THE

COLLIERY MANAGER'S HANDBOOK

A COMPREHENSIVE TREATISE ON THE LAYING-OUT AND WORKING OF COLLIERIES

DESIGNED AS

A BOOK OF REFERENCE FOR COLLIERY MANAGERS

AND

FOR THE USE OF COAL-MINING STUDENTS PREPARING FOR FIRST-CLASS CERTIFICATES

BY

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Containing over 700 Plans, Diagrams, and other Illustrations

FOURTH EDITION, REVISED AND ENLARGED

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that the best and indeed the only satisfactory preparation for the efficient discharge of the duties of the Colliery Manager is that which is to be gained in the laborious school of experience; but he knows also that the wise use of a carefully prepared and comprehensive HANDBOOK—such as he ventures to believe the volume now in the reader's hand will be found to be—would have been to him an incalculable boon in the earlier years of his course, and hardly less so subsequently as a book of reference and of practical guidance. No doubt excellent handbooks already exist, dealing with various branches of the subject; but the Author's aim has been in the following pages to prepare a work more complete in itself than any of its predecessors, and one comprising the most recent information as to the ever-progressing science and art of Coal Mining.

In the use of the work by students, it is believed that the questions and answers given on the various subjects will be found specially helpful.

The Author gratefully acknowledges the assistance which has been given him in preparing this work for the press by his friends Mr. J. T. Robson, H. M.'s Inspector of Mines for South Wales, and the Rev. J. E. Flower, M.A., of Wandsworth; and also the courtesy shown by the Council of the Mining Institute of Scotland in granting him permission to quote the result of their Commission of Mining Engineers, as given in their *Transactions* (vol. iii., pp. 51 to 124). Besides much valuable information which has been obtained from the *Transactions* of the North of England and South Wales Institutes of Mining Engineers, he has also received considerable assistance from the columns of the *Practical Engineer* and the *Colliery Guardian*, and from other sources, reference to which will be found in various pages of the work.

The greatest care has been taken to ensure accuracy in every respect, but if any mistake has escaped notice the Author will be thankful to have it pointed out to him.

The preparation of this HANDBOOK—which for several years has occupied such time as the writer has been able to give to it—has been to him a most congenial task; and he ventures, in sending it forth, to express the hope that the work may be found as useful to the student, and to those engaged in the management of collieries, as the preparation of it has been pleasant and profitable to himself.

PONTYPRIDD,
January, 1891.

PREFACE TO FOURTH EDITION.

A fresh edition having been called for, the Author has seized the opportunity carefully to revise the whole work in order to bring it in every respect up to date.

It will be found that an important addition, involving about eighty pages, with thirty new illustrations, has now been made to Chapter XII., owing to the special attention of the Author having been recently directed to the subject of Mine Ventilation and the use of Regulators in airways.

This further enlargement of the volume, without any addition to the original price, will, it is believed, greatly enhance the value of the work as a whole, both to the practical Colliery Manager and to the student.

In spite of the utmost care in preparation, it can hardly be

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THE

COLLIERY MANAGER'S HAND-BOOK.

CHAPTER I.

GEOLOGY.

Stratified and Unstratified Rocks—Dip—Rise—Outcrop—Strike—Diversified arrangement of Rocks—Faults—Classification of the Rocks into Systems—Igneous Rocks—Description of the different systems of Stratified Rocks.

THE Science of Geology in its widest sense comprises all that is known concerning the constitution and history of our globe. It has been the life-study of many eminent men, and as one of the youngest of the sciences has made amazing progress; but our knowledge, gained by direct observation, is still limited to within a mile or two of the surface, and there is consequently room for much speculation as to the condition of the interior of the earth.

An examination of the rocks soon led to their being divided into *stratified* and *unstratified*. The former have the appearance of having been deposited in layers one above another, by aqueous action; whilst *unstratified* rocks

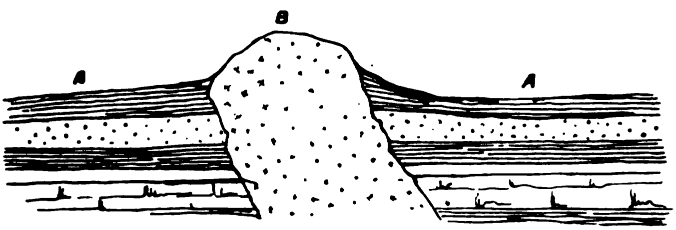


Fig. 1.—STRATIFIED AND UNSTRATIFIED ROCKS.
A A Stratified, and B Unstratified.

occur as amorphous masses and seem to have been more or less completely fused. Granite is a specimen of the latter group of rocks. It is now generally held by geologists, physicists, and astronomers, that the earth must, at one time, have been entirely in a state of fusion, and that the crust in cooling became a hard unstratified mass. In process of time the waters began to destroy this crust, and to deposit layer after layer of the derived material. This process of disintegration by mechanical and chemical agencies, and deposition of the resulting detritus, with intermittent eruption of heated matter from the earth's interior, has gone on through countless ages until the earth's crust has obtained its present condition. The sedimentary rocks were not always deposited on horizontal surfaces, and the result of deposition on an inclined surface would be to form a bed at nearly the same angle as the surface on which it was deposited.

As beds were deposited over extensive areas there may have been many variations of surface causing the beds to be inclined at different angles and in different directions, but in most cases where the strata are inclined, they were originally deposited in a position more or less horizontal, and have been subsequently disturbed by earth-movements. Inclined strata are shown at B in Fig. 2.

In almost the earliest of the aqueous rocks signs of vegetable and animal life are found in the shape of *fossils*. As the organisms died, the waters covered them, depositing again other layers of sand, clay, &c. In course of time the waters receded, or, more probably, the land was elevated, and fresh forms of life appeared, to be themselves buried in their turn. The fossils represent either the

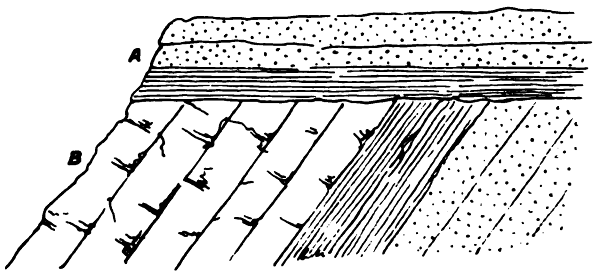


Fig. 2.—Horizontal and Inclined Strata, Showing Unconformity.

creatures which inhabited the waters in which sedimentation was proceeding, or those which lived on the neighbouring land, and had their remains carried down and entombed in the sediment.

The great fact to be remembered in regard to these stratified rocks is, that they have been deposited in regular order, and that each has its characteristic fossils. Not that every formation of rock was deposited all over the world, but the order of their occurrence, if there, remains generally the same. Sometimes,

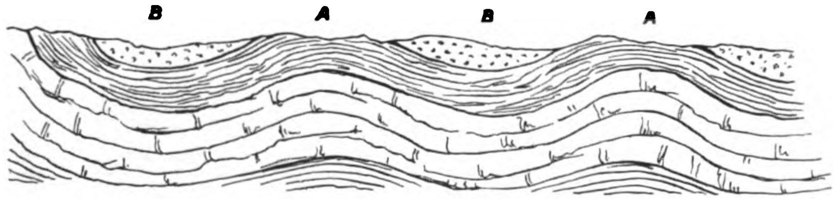


Fig. 3.—Curved Strata.
A, Anticlinal. B, Synclinal.

indeed, a formation is altogether wanting, or the order of their position reversed, but these are exceptions and can usually be accounted for by denudation or by volcanic agency, or by other earth-movements.

The angle at which a stratum inclines to the horizon is called its *dip* when viewed in the direction of the fall, and the *rise* when viewed in the contrary direction. When an inclined stratum comes to the surface its edge is called the *outcrop*, and the line of outcrop along a horizontal surface is termed its *strike*.

The dip is always at right angles to the strike, so that if the dip be given, the strike is also known; but if the strike only be given, the dip cannot be known from it, because the dip may incline to either side of the strike. When strata dip in opposite directions from a ridge or axis of elevation they form an *anticline* or *saddleback* as at A, Fig. 3, and when they dip towards a common line of depression as at B, they are said to be *synclinal*, and the depression so formed is spoken of as a *trough* or *basin*. These terms apply only to the folds of the strata and not to the features of the surface of the ground. When strata are bent and twisted they are termed *contorted*. When strata lie upon each other in parallel order, they are said to be *conformable;* but when one set reclines upon another at a different angle they are termed *unconformable*. Thus, in Fig. 2, the strata A are conformable to each other, and so are the strata B, but there is a strong *unconformity*, or discordance of stratification, between A and B. The strata are said to be *monoclinal* when, though inclined, they all slope the same way; and *periclinal* when dipping in every direction from a common centre. When strata terminate abruptly in a bold bluff edge, they are said to form an *escarpment*.

The dislocations, fractures, and fissures produced in the rocky crust by subterranean movements are known by such terms as *faults*, *slips*, *hitches*, *heaves*, *leaps*, *throws*, *troubles*, *&c.*

Faults may be so thin as to be easily mistaken for the ordinary jointing of the rocks they traverse. More often, however, there is a considerable space between their "walls" or "cheeks." This space is sometimes filled up with *débris* from the adjoining rocks, or with mineral matter deposited from solutions circulating within them, such as iron pyrites, oxide of iron, and other metallic substances, together with quartz, clay, &c. When a fissure is filled with matter injected from the earth's interior, such as basalt, it forms a *dyke;* when filled with mineral matter, like quartz, it constitutes a *vein*, and if a metallic mineral, or *ore*, be present, the vein is often termed a *lode*.

Fissures due to these disturbances may contain carbonaceous matters where they traverse the coal measures, which may at times be accompanied by metallic minerals. There is no essential difference between faults found in the coal-measures and the mineral veins ("fissure veins") of a metalliferous district. For instance, the great Minera lead lode is, throughout a part of its course, the chief fault of the Denbighshire coal-field.

The displacement in faults is not necessarily caused at one time; there is frequently clear evidence of repeated movement. They may occur in groups of two or three, with parallel or nearly parallel bearings, sometimes all dipping in the same direction, but usually two sets dip in opposite directions. There is often connected with these groups another series at right angles to them of a somewhat later origin.

The Igneous Rocks are now usually divided into a *plutonic* series, including such rocks as granite, which seem to have solidified at a great depth under enormous pressure; and a *volcanic* series, including rocks like basalt, which have consolidated at or near the surface.

The study of the igneous rocks is attended with much difficulty. Their mineral composition is usually complex, and often two rocks having the same composition present a totally different appearance, owing to the fact that they were cooled under different circumstances. They have no order of superposition, but have been erupted from below, and are found traversing the stratified rocks in veins or dykes, sills, or intrusive sheets, and irregular bosses. Moreover they contain no organic remains by which their relative ages can be satisfactorily established. The igneous rocks contain many valuable minerals, building stone, &c.

The Metamorphic Rocks form an obscure group, often regarded as the lowest of the stratified rocks. Natural agencies like subterranean heat, water, and pressure, have so profoundly affected certain rocks that their original condition cannot be determined: hence the origin of such rocks as gneiss and mica-schist is still open to much discussion. Some metamorphic rocks are undoubtedly altered sediments and others altered igneous rocks. Serpentine, for instance, is believed to result from the chemical alteration of certain igneous rocks; whilst clay slate, regarded by some as a metamorphic rock, evidently results from the mechanical alteration of sedimentary matter of an argillaceous character. The Metamorphic rocks are very productive in a mercantile point of view, yielding marble, slate, serpentine, and quartz; whilst metallic veins containing copper, lead and tin frequently traverse these rocks. Gold and silver and many precious stones are among the valuables they contain. Plumbago or graphite, one of the three forms of carbon, is also a product of the Metamorphic System.

Rocks which contain no fossils are sometimes termed *azoic* or *hypozoic;* whilst those which yield traces of only the earliest forms, or what are supposed to be such, have been designated *eozoic*.

The STRATIFIED ROCKS have been arranged in three large groups, named according to their chronological succession *Primary*, *Secondary*, and *Tertiary*. But these names are now often replaced by the respective terms *Palæozoic*, *Mesozoic*, and *Cainozoic* (or better, *Cænozoic*), these terms having regard to the nature of the organic remains in the rocks, and indicating respectively the "ancient," "middle," and "recent" types of fossils. The following table of strata shows the sequence of the great groups, the oldest being placed at the bottom of the scheme:—

QUATERNARY	Recent. Pleistocene.
TERTIARY OR CAINOZOIC	Pliocene. Miocene. Oligocene. Eocene.
SECONDARY OR MESOZOIC	Cretaceous. Oolitic. Liassic. Triassic.
PRIMARY OR PALÆOZOIC	Permian. Carboniferous. Devonian. Silurian. Ordovician. Cambrian.
ARCHÆAN	Pre-Cambrian.

Each of these groups will now be briefly described, commencing with the oldest and proceeding thence in regular ascending order.

The ARCHÆAN SYSTEM includes an enormous thickness of rocks, principally gneiss and crystalline schists, which are generally regarded as metamorphic rocks. They are largely developed in the Scotch Highlands, in the outer Hebrides, and in the North of Ireland. It has been shown, in recent years, that the archæan rocks of the North-west Highlands have been subjected to stupendous earth-movements, whereby huge masses of rock have been violently disturbed and moved for miles along thrust-planes. The archæan rocks of Canada, studied by the late Sir W. Logan, consist of a lower group, about 30,000 feet thick, and an upper group which in places may be 20,000 feet in thickness: the former has been called the *Laurentian* and the latter the *Huronian* series, both names having geographical significance. Certain bands of crystalline limestone in the Laurentian series have yielded a curious structure, regarded by Sir W. Dawson as a gigantic reef-forming foraminifer and termed *Eozoon*—the oldest known form of life. Its organic nature is, however, extremely doubtful. It is notable that the Canadian Laurentians contain much graphite, or plumbago, and apatite, or phosphate of lime.

In certain parts of Pembrokeshire, Carnarvonshire, Anglesey, Shropshire and elsewhere are some very ancient rocks, consisting of altered stratified deposits and contemporaneous igneous rocks, which are generally termed *Pre-cambrian.* They have been extensively studied at St. David's by Dr. Henry Hicks. The Torridon sandstone of Scotland is also now regarded as Pre-cambrian.

THE CAMBRIAN SYSTEM, named by Sedgwick, is largely developed in North Wales, where it comprises (reckoning from below upwards) the *Harlech* and *Llanberis beds*, forming a great thickness of slates, grits, and sandstones; the *Menevian* beds, named by Hicks and Salter from the classical name of St. David's; the *Lingula flags*, so called from a little fossil formerly known as *Lingula ;* and the *Tremadoc slates*, named by Sedgwick from a typical locality in Carnarvonshire. Cambrian fossils, though not numerous, are of profound interest, inasmuch as they represent the earliest known types of life. The most interesting are the Crustaceans, called *trilobites*, and a recent classification of the Cambrians, now commonly adopted, is based on the occurrence of characteristic trilobites. The Lower Cambrians form the *Olenellus* zone; the middle Cambrians the *Paradoxides* zone; and the Upper Cambrians the *Olenus* zone. Most of the best slates occur in the Cambrian system; those of Llanberis and Penrhyn being worked in the Llanberis series. The gold of North Wales is found in quartz veins, associated with the igneous rocks of the Cambrian system.

THE ORDOVICIAN SYSTEM, named by Lapworth after the old tribe of the Ordovices in North Wales, comprises the strata formerly termed Cambro-Silurian, and equivalent to the "Lower Silurian" of the Geological Survey. The Ordovicians include the *Arenig series* of slates and volcanic rocks, developed in the Arenigs, the Arans and Cader Idris; the *Llandeilo group* of shales and sandstones, so named from the locality in Carmarthenshire; and the *Bala* or *Caradoc series*, which includes the famous Bala limestone, rich in marine fossils. The old igneous rocks of Snowdon are referable to the Bala period; whilst those of the Lake District of Cumberland belong to the Llandeilo, and probably in part also to the Bala, age. The characteristic fossils in many of the Ordovician shales are the little *graptolites*, allied apparently to the modern sertularians. Phosphatic nodules occur in the Bala beds of North Wales.

THE SILURIAN SYSTEM, named by Murchison from the ancient British Silures, is typically developed in parts of South Wales, and in Monmouthshire, Gloucestershire, Herefordshire and Shropshire. It comprises the *Llandovery series*, known also as the *May Hill group*, both being geographical names; the *Wenlock series*, with the well-known limestone, at one time largely quarried at Dudley; and the *Ludlow* group, consisting of an upper and a lower member, separated by the Aymestry limestone. Both limestones yield splendid fossils, especially corals, crinoids, trilobites, brachiopods, and cephalopods. The veins of silver-lead ore, formerly of great value in Cardiganshire, occur in the Lower Llandovery rocks.

THE DEVONIAN SYSTEM received its name from Sedgwick and Murchison in 1836 in consequence of its development in Devonshire. It consists of a lower group of conglomerates, shales and sandstones; a middle group of limestones; and an upper group, principally of sandstone. The limestone, known as the Great Devon Limestone, is rich in corals, and is quarried as a marble as well as a building-stone. The Devonian slates of West Devon and Cornwall are disturbed by huge bosses of granite, forming a series of heights stretching from Dartmoor to the Land's End. The slates known locally as *killas* are, for the most part, imperfectly cleaved, but good roofing slates are worked at Delabole and Tintagel in North Cornwall. Granite is extensively quarried. Near the junction of the killas with the granite rich lodes of tin- and copper-ores occur; and it is notable that while these veins usually course in a nearly E. and W. direction, the lead-lodes run almost N. and S.

THE OLD RED SANDSTONE is probably the equivalent of the Devonian rocks, but appears to have been deposited in lakes and not in the sea. In Scotland a series of these ancient lakes has been mapped out by Sir A. Geikie. A large area in Herefordshire, Monmouthshire, and part of S. Wales, is occupied by Old Red rocks, consisting of marls, sandstones, pebble-beds, and impure nodular limestones termed "cornstone." The Caithness and Arbroath paving-flags belong to this system. The Old Red Sandstone yields the remains of curious ganoid and other fishes, and large extinct crustaceans. In the South of Ireland fresh-water lacustrine beds occur of the age of the Upper Old Red. The upper and the lower groups in Scotland are divided by a strong break. It is notable that the upper old red passes conformably into the overlying carboniferous rocks, and is sometimes regarded, locally, as representing the shore-deposits of the early carboniferous sea.

No geological era has bequeathed to us a more valuable deposit than the CARBONIFEROUS PERIOD, which succeeds the Old Red Sandstone. Interstratified with the rocks we find those valuable seams of coal which are the greatest wealth of a country.

The system is typically separable into four well-marked groups—viz., the Lower Limestone Shales or Carboniferous Slates, the Mountain Limestone, the Millstone Grit, and the Coal Measures.

The Carboniferous strata throughout are composed of frequent alternations of sandstones, shales, limestones, coals and ironstones.

The *Lower Limestone Shales* form a series of beds between the Old Red Sandstone and the Carboniferous Limestone. The *culm measures* of Devonshire may belong partly to this series. In Northumberland the Lower Carboniferous strata have been called the *Tuedian* beds. The Lower Carboniferous beds are very scantily developed in some districts and in others attain a great thickness, so that it is difficult to state their average thickness, although frequently given at 1,000 feet, the Mountain Limestone at from 500 to 3,000, the Millstone Grit at 600, whilst the true Coal Measures vary from 3,000 to 12,000 feet thick. The *Calciferous Sandstone series*, of Scotch geologists, includes the lower part of the Mountain Limestone and the Lower Limestone shales.

The *Carboniferous* or *Mountain Limestone* is, perhaps, the most distinct and unmistakable group in the whole crust of the earth. Its limestones and fossils are so marked and peculiar as to form a guiding post to the miner and geologist. By far the greater part of the workable coal seams of Scotland is included in the Mountain Limestone group. The Mountain Limestone is absent from both the Shrewsbury and South Staffordshire coal-fields, the true coal measures there resting on the Cambrian and Silurian rocks; whilst in Worcestershire, the Forest of Wyre Coal Measures rest on a bed of Old Red Sandstone. In Shropshire the Silurian rocks form the general foundation to the carboniferous formations.

Between the Mountain Limestone and the Millstone Grit in Yorkshire and the North of England is a thick series of black shales with sandstones, limestones, and even coal seams, called by the late Prof. Phillips, the *Yoredale rocks*. They take their name from Yoredale, or Wensleydale, in Yorkshire; and are otherwise known as the *Upper Limestone Shales*. The *Bernician* beds of Northumberland were named by Prof. Lebour, and include representatives of the Yoredale rocks as well as of the Carboniferous Limestone.

The *Millstone Grit* is composed of a series of hard and coarse sandstones and shales, usually of a grey, white, or yellow colour, but occasionally red. It is sometimes absent and rarely attains a greater thickness than 1,000 feet, although it is said to reach, in some places, a maximum of 5,000 feet. Occasionally it contains thin seams of coal, but is known to miners by the name of the Farewell Rock, suggesting that, on a sinking reaching this rock, farewell has been said to the coal seams of the true Coal Measures.

In Derbyshire the Mountain Limestone consists of an enormous mass of calcareous rocks almost destitute of sedimentary matter and entirely so of coal. Further north in Lancashire and Yorkshire workable coal seams are found at a stage earlier than the true Coal Measures—in the Millstone Grit. Still further north, in Northumberland, several beds of coal are found near the base of the Mountain Limestone formation. The coals of the Mountain Limestone of Scotland occupy a position similar to that of those in Northumberland, but in Scotland the sedimentary strata are more largely developed.

The true *Coal Measures* furnish us with those valuable beds of coal which contribute so much to our country's prosperity and power. The series sometimes occurs immediately above the Mountain Limestone, the Millstone Grit being then wanting. One of the most notable features in its composition is the frequent recurrence of seams of coal, or of bituminous shale, all speaking of an enormous profusion of vegetable growth. The organic remains of the Coal Measures are peculiarly well defined. The fishes are chiefly of large size, and in certain fields there are evidences of terrestrial life in the skeletons of amphibians, fragments of land shells, and remains of insects. The great feature of the period, however, is the abundant flora, which comprises forms which are now only distantly represented in tropical swamps and jungles, and point possibly to a tropical condition of climate. The coal-measure flora consists chiefly of lycopods, or club-mosses, some of gigantic size; horse-tails, or *equisetaceæ;* ferns and conifers. In some coals, and associated rocks, the resinoid spores of lycopods are well preserved.

It must be borne in mind that the coal is composed of carbon, hydrogen, and oxygen, elements which enter into the composition of vegetable organisms, so that the coal seams are not the result of direct mechanical deposition of mineral matter, like the associated sandstones and shales. The most reasonable theory as to the origin of coal seems to be that it is the remains of vegetable matter which became decomposed and mineralized on the spot where it grew, and where it is now found. This vegetation might have grown on the borders of great lakes, estuaries, and vast lagoons. In process of time these layers of vegetation, might, by subsidence, be carried down beneath the sea-level, and the water consequently flowed over them, depositing layers of sand, silt and mud, which we now find alternating with our coal seams as beds of sandstone and shale. The downward movement must have been irregular or intermittent, and in the long pauses the sediment would fill up the lagoons, and fresh jungles would spring up, which, when the downward movement set in afresh, would be in turn entombed beneath the silt and mud of the sea waters. Each coal seam thus represents the vegetation of an old land surface, and the alternations of sandstone, shale, fireclay, &c., represent the different sediments which were brought together by the combined action of the sea and estuarine waters during the slow and irregular subsidence of their beds. It will be noticed that nearly all coal-fields are basin-shaped. The synclinal form is, however, usually due to earth-movements subsequent to the formation of the coal seams. Each coal seam has characteristics peculiarly its own, often prevailing over very wide areas. Besides the distinctive features in the formation of coal seams, the coal obtained from one differs from that yielded by another, and frequently one coal seam gives different qualities of coal. The differences arise chiefly from the difference of chemical composition, as the hydrogen, oxygen and carbon are present in the coal in varying states of combination.

The theory that coal has been formed in its present position, so strongly advocated by Logan, after his observations on the underclays with their stigmarian rootlets, has of late been warmly opposed by certain French geologists, who feel justified in reverting to the old view that most coal consists of altered vegetable matter drifted into estuaries and lagoons, and therefore removed from its original site of growth.

Different samples of coal give out amounts of heat during combustion which are not always anticipated from their chemical composition, or expected from analysis. For this reason the different coals used in boiler tests should be experimented on, to ascertain the actual heat of combustion, as well as that estimated from analysis, otherwise the results of the tests made may be quite misleading.

Commercially the Carboniferous is the most important and valuable system to man. It yields building stone of the best quality, flagstones for paving, grits for grindstones, limestones for many purposes, marbles, fire-clay, ironstone, ochre, alum, copperas, and coal of various qualities. The Mountain Limestone yields ores of lead, zinc, antimony, and sometimes silver and gold.

Succeeding the Coal Measures we have the PERMIAN SYSTEM, so named from its development in the ancient kingdom of Perm in Russia. The system consists of red sandstones and marls, conglomerates and breccias, with limestones which are usually magnesian, or dolomitic. In some places they contain 44 per cent. of carbonate of magnesia associated with the carbonate of lime. The Permian fauna represents that of the late palæozoic era. In Germany the *Kupfer-schiefer*, represented in this country by the marl-slate of Durham, has been long mined as an ore of copper.

The dolomite or magnesian limestone of the North of England is used as a source of magnesian salts, and also as a building-stone. The Permians vary in thickness, but may be as much as 3,500 feet in this country.

The TRIAS or NEW RED SANDSTONE, the latter name being given because reddish hues prevail throughout its sandstones and shales in the British Isles, consists of two members—a lower group of sandstones and conglomerates, called the *Bunter*, in allusion to the variegated tints of the rocks; and an upper group, consisting chiefly of red marls and sandstones, and known as the *Keuper*. Where typically developed, as in Germany, these two members are separated by the *Muschelkalk*, or shell-limestone; and the term Trias has reference to this three-fold character. The limestone is wanting in England, and as the best fossils are to be found in this rock we have not the local facilities for an extensive knowledge of the flora and fauna of the period. The plants of the Triassic strata have a strong resemblance to the flora of the Lias and Oolite above, consisting of ferns, cycads, and conifers. The industrial products yielded by the system are sandstones of varying quality, gypsum, and rock-salt. All the salt of Cheshire, whether mined as rock-salt or pumped as brine, is obtained from the Keuper or New Red Marl: the Lower Keuper sandstones occasionally contain copper-ores and other metallic minerals, as at Alderley Edge, in Cheshire. The Trias is at least 4,000 feet thick in England.

The LIAS, which follows the Trias, or is separated from it by a thin series of strata known as the *Rhætic* or *Penarth beds*, is composed of dark argillaceous limestones, and bluish clays, forming the *Lower Lias*, separated by the marlstone, or *Middle Lias*, from the group of shales and limestones which constitute the *Upper Lias*. The fauna of the period is diversified and interesting. Upwards of 120 species of ammonites have been discovered, and belemnites are also very numerous. The most interesting feature in the life of the age is the appearance of marine reptiles of extraordinary size and structure, such as the Ichthyosaurus and Plesiosaurus.

The well-known ironstone of Cleveland, in Yorkshire, which lies in thick beds, is in the Middle Lias. Ironstones in other parts have also been found in the Lias. The argil aceous limestones are largely quarried for mortar and hydraulic cement. Jet is obtained from the Upper Lias of Yorkshire, and alum-shales were formerly worked to a large extent near Whitby.

The Lias is about 1,200 feet thick.

The Oolites.—Above the Lias in the South and West of England are the Oolites. These consist of alternations of oolitic limestones, calcareous grits, shelly conglomerates, yellowish sands, and thick bedded, bluish grey clays more or less calcareous. The Oolites and the Lias are now frequently united as one large group, known as the *Jurassic system*.

The flora of the oolitic era was extensive, as may be gathered from the fact that at Brora, in Sutherlandshire, there is the thickest stratum of coal found in any British Secondary rock. It has been worked intermittently for a long period, the seam being 3½ feet thick. In Yorkshire there is a group of coal-bearing rocks of Lower Oolitic age, consisting of sandstones, shales, and iron-stones, with seams of coal, some of which may reach a thickness of 18 inches. These "Moorland coals" rest on underclays, and are associated with many vegetable fossils, especially the remains of cycads, ferns, and horse-tails. The Oolites yield limestones like those of Bath, Portland, Ketton, and Ancaster, excellent for building purposes. They also yield Fuller's earth. Lignite or wood-coal is likewise found, but is of no economic value. The Oolites are about 1,800 to 2,000 feet thick.

The Cretaceous System.—The lowest member of the Cretaceous group is the Wealden; this deposit is of fresh-water origin, the bones of terrestrial animals and the relics of land plants, testifying that the Wealden owes its existence to some great river which brought down mud, sand, &c., from the continent it drained, depositing the detritus in a delta. The Cretaceous beds above the Wealden are of marine origin. They include the Lower Greensand, Gault clay, and Upper Greensand, followed by the Chalk, which is the most conspicuous member of the system. Some geologists unite the Lower Greensand with the Wealden beds, under the name of the *Neocomian* group. The Chalk is for the most part a white earthy limestone, upwards of 1,000 feet thick, with bands of flints in the upper portion. The organic remains are sponges, corals, star fishes, molluscs, crustacea, fishes and reptiles. Fossil plants are comparatively rare. Industrially, the chief products of the system in Great Britain are chalk and flint. Several of the workable coal seams of Vancouver Island, and British North America, belong to the Cretaceous epoch. The Cretaceous system is over 4,000 feet thick.

The Tertiary System embraces a large series of clays, sands, pebble-beds and other strata, overlying the Cretaceous rocks, and well developed in the two areas known as the London and Hampshire basins, the latter extending into the Isle of Wight. The Tertiaries have been arranged in a number of groups, of which the principal are termed in ascending order, *Eocene*, *Oligocene*, *Miocene*, and *Pliocene*—these names having reference to the proportion of recent and extinct mollusca in their fossil fauna.

At the base of the Eocenes are the strata known as "The Lower London Tertiaries," comprising the Thanet sands, the Woolwich and Reading series, and the Blackheath or Oldhaven beds. Then follows the London Clay—a stiff clay of bluish or brown colour, reaching a maximum thickness of about 500 feet, and containing zones of septaria, or nodules of argillaceous limestone. Iron pyrites and selenite are also common in this clay. The Bagshot beds, above the London clay, include the famous potters' clays and pipe-clays of Poole, in Dorsetshire, and Bovey Tracey in Devon. At the latter locality are beds of lignite, known as "Bovey Coal," associated with numerous plant remains, but of little or no industrial value. The Bovey Coal was formerly regarded as miocene, but most geologists now hold that although miocene strata are very largely developed in parts of the Continent, they are not represented in Britain.

During the early Tertiary period volcanic action was energetic in the British area, as attested by the thick sheets of basaltic rocks, or old lava-flows, in the N.E. of Ireland and the West of Scotland. The celebrated columnar basalt of the Giant's Causeway belongs to this period.

The Oligocene strata are represented in this country by the fluvio-marine series of the Isle of Wight. In the absence of Miocenes, we then pass to the Pliocene beds, which are fairly developed in East Anglia, where they consist chiefly of the shelly sands known as "Crags." The total thickness of the British Tertiaries may reach about 2,000 feet.

Most geologists regard the strata above the Tertiaries as forming a distinct group known as the POST-TERTIARY or QUATERNARY SYSTEM. This group includes the Pleistocene deposits, which consist chiefly of clays, sands, and gravels, in the formation of which ice is believed to have played a more or less important part, and which are consequently referred to the Glacial epoch, or Great Ice Age. This period of the earth's history was marked by the prevalence of arctic conditions over a large part of the northern hemisphere, but it is not always easy to determine whether the "drift" has been accumulated by the action of land-ice or of floating ice. Ice-borne boulders and erratics are common in many parts of Britain, and the great Northern drift comes down as far south as Finchley, on the north of London.

A remarkable instance of rock believed to have been deposited in glacial times, is that at the Stevens Mine, in Mount M'Clellan, California, its altitude being 2,500 feet. At a depth of from 60 to 200 feet, the vein, consisting of silica, calcite and ore, and also the surrounding wall rocks, are a solid frozen mass. Many other mines in the same vicinity have similar belts of frozen ground, and as they are at considerable depths from the surface, where there are no openings for air currents, the theory is that the frozen mine is due to imbedded icebergs of the glacial period. There has been no diminution in the frost in descending, and the frozen material is so hard as to render the miner's pick and drill of no use in excavating it.

All the accumulations formed since the Glacial period are known as *Recent* or *Modern* deposits. At its close the present distribution of sea and land seems to have been established, and the earth to have then had its present flora and fauna, with the exception of some local removals of certain plants and animals and the general extinction of a few species. Theoretically the accumulations of the present era are not only of high interest in themselves, but of prime importance in furnishing a key to the complicated phenomena of former epochs. Practically they present many important features to the farmer, engineer, and navigator, and furnish us industrially with such products as brick clay, sand, marl, and peat, and in other countries pumice, sulphur, brown coal, and amber.

Fig. 13 shows a group of improved boring tools, as manufactured by Messrs. Thornewill & Warham, of Burton-on-Trent, and supplied by them singly or in sets.

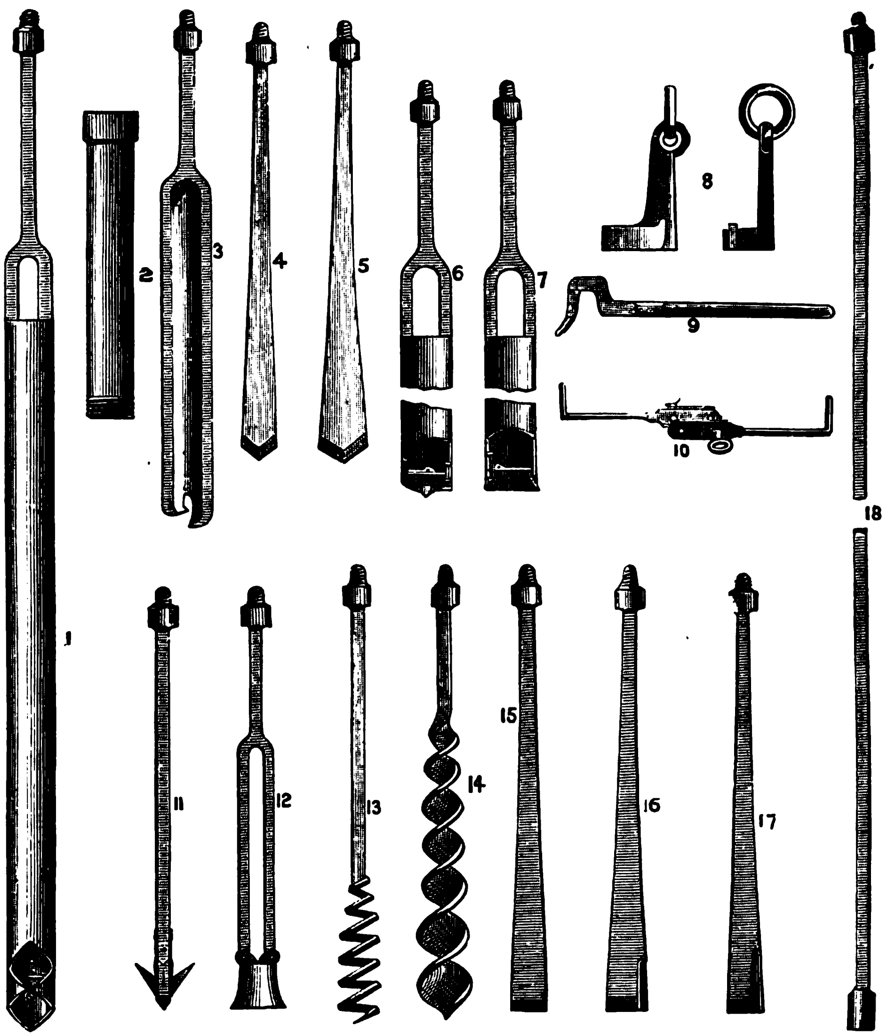


Fig. 13.—Messrs. Thornewill and Warham's Improved Boring Tools.

List of Boring Tools in Fig. 13.

1. Shoe-nose shell with valve for bringing up loose stuff.
2. Wrought-iron screwed well bore pipes.
3. Auger for clay and stiff soil.
4. and 5. V-nose chisels for hard ground.
6. Shell-auger with valve for loose and wet soil.
7. Bell-shell with valve for loose gravel.
8. Lifting dog for raising rods.
9. Pair of rod-wrenches for screwing and unscrewing rods.
10. Levers for turning rods.
11. Spring dart for drawing pipes in bore-holes.
12. Bell-box for bringing up broken bits.
13. Spiral worm for extracting broken rods.
14. Worm auger for loosening stuff in bore-holes.
15. Square-nose chisel.
16. S-nose chisel for hard strata.
17. T-nose chisel for hard strata.
18. Rods with screw joints in 5 and 10-foot lengths.

but these prices would not apply to basalt, whin, or other excessively hard rock.

The depth of a bore-hole is measured by the number of rods, and the kinds of strata judged by the borings brought up by the tools used. Changes in the stratification are noted by the charge-man, who is guided by the sound and sensation through the hands when the rods fall, and he marks each change on the rods.

As the depth of a hole bored in this way increases the rate of progress decreases, and when a great depth has been reached a considerable amount of time is occupied by the necessary changes in the tools, and very little in actual boring. In cutting hard rocks far beneath the surface this system is extremely slow and expensive. A steam engine may be used, so as to give more power in lifting the weight of rods, which becomes excessive as the bore-hole reaches a great depth, but even then the process does not admit of rapid boring. The

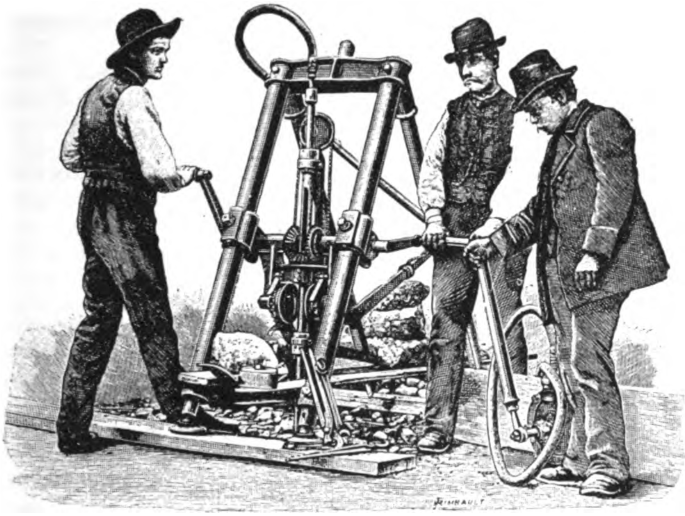


Fig. 14.—Schram's Diamond Prospecting Drill.

vertical engine usually gives motion by means of belting to a shaft carrying a cam roller, and this lifts a wooden beam or lever near that end from which the boring rods are suspended. At every two or three revolutions of the small engine the cam roller makes one, thus raising the beam and allowing it to fall. The other end of the beam rests in a trestle, which acts as the fulcrum, and allows of change by sliding the trestle nearer the rod-end of the lever. Between it and the cam roller is placed a strong, upright column to receive the lever on its descent, and this is protected on the top by a cushion of india-rubber to break the shock.

The ordinary method of hand boring is slow and costly, especially in hard rocks, and there is a great disadvantage in the form of drill which cuts the rock into dust or slime in the bore-hole, so that when samples are brought to the surface for examination it is difficult, if not impossible, for even experienced borers to classify the strata. The solid cores of the rock obtained by the Diamond drill afford a much more satisfactory means of determination, as the samples withdrawn may be judged by their appearance, their markings, fossils, thickness of bed, and chemical tests of composition.

A hand-power prospecting diamond drill is arranged in the manner shown in Fig. 14, which is suitable for boring holes at any angle to a depth not exceeding 300 feet and a diameter of $1\frac{3}{8}$ inches, yielding cores of $\frac{7}{8}$ inch.

washed from under the crown as the boring proceeds. As shown in the drawing, the crown is slightly larger than the other portions of the boring tool, so as to reduce friction. The manner of procedure is as follows:—

The boring tool, with one length of rod attached, is lowered into the bore-hole by means of the chain over the shear-legs pulley, and is then suspended by clamps placed over the bore-hole until one or more lengths of rods (which consist of ordinary tubes threaded at the ends) are raised by means of the chain,

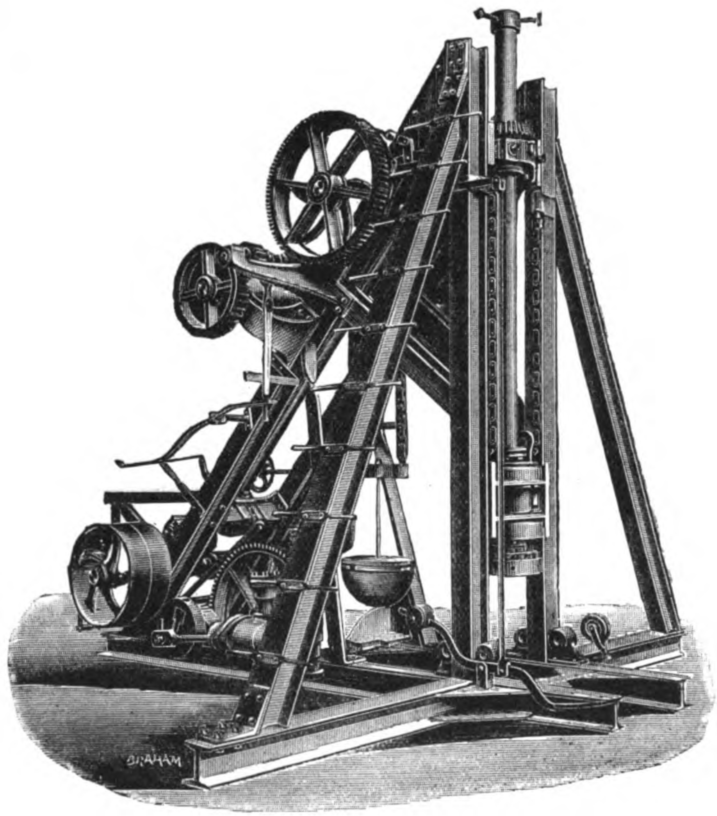


Fig. 16.—The Diamond Rock Drill.

and attached by screwing to the rods held by the clamps. All is then lowered together, and again suspended by clamps. This process is repeated again and again until the crown reaches the bottom of the bore-hole. The machine is then moved forward on the rails into its working position, the cross-head lowered, and the hollow shaft screwed on to the boring rods.

The rods are then given a rapid rotary motion (from 200 to 300 revolutions per minute), and the motion of the rods being transmitted to the crown an annular channel is cut by the diamonds, and as the rods descend the core formed inside the channel enters the core-trap, and afterwards the core-tubes. As the rods are revolved the pump is set in motion, and water is forced down the hollow rods, A, as shown by the arrow, at a pressure which must be sufficient to cool the crown, and keep it clean by sweeping away the eroded material. After passing under the crown the water returns to the surface by the annular space formed between the walls of the bore-hole and the boring rods, as shown by the arrows.

Unless the ground is soft and friable the core formed is solid, and will not break till the act of withdrawing the rods takes place, when it parts at the base and is retained within the core-tube by the expanding ring. Upon reaching the surface these cores are all accurately numbered and safely housed by the attendant in charge, who also enters all notes as to stratifications between the numbers of cores not extracted, and is responsible to his employer for the preservation of an accurate account of the strata.

Fig. 16 shows Appleby and Beaumont's patent diamond drill, which is capable of boring holes up to 16 inches in diameter to depths of 2,000 feet, and of bringing up solid cores 30 feet long weighing more than 3 tons.

The framework of the drill is built of steel girders, and consists of a strong tripod mounted on horizontal girders, provided with all the appliances required to rotate the drill, to counter-weight the rods, and give a uniform pressure on the cutting-head.

The boring rods are made of drawn steel tubes with screwed flush joints, and are caused to rotate by being clipped in a universal chuck to a revolving quill, which has a stroke of 6 feet, and works in the vertical slides attached to the upright side-frames of the machines. The pump is connected with the top of the boring rods by means of a flexible tube. The weight of the rods is counterbalanced so that an even pressure is maintained on the boring crown.

The crown is screwed on to the core tube, and this latter for hard rock has a length of about 20 feet. The core tube is of a slightly less exterior diameter than that of the crown, and may be of much greater diameter than that of the boring rods which are screwed into it. The boring rods are of course hollow, and, when at work, a constant current of water is forced down them in order to keep the crown cool and clear away the borings, which are carried up to the surface as fine sand.

When a certain depth has been bored, varying according to the strata and the length of the core tube, the core clip is used for detaching the solid portion left standing within the annular groove cut by the crown, and the core is then raised to the surface.

The following is a complete specification of the outfit necessary for a machine ready for work, for a boring 2,000 feet deep:—

Boring machine, as shown in the illustration (about 395*l*.).
Portable engine of 10 horse power.
2 engine straps.
6 1¼-inch flexible hoses.
3 pairs of unions.
2 water unions.
150 feet of ¾-inch chain.
1 18-inch top sheave, with spindle and bearings.
2,000 feet of steel boring rods and joints.
300 feet of 6-inch steel lining tubes, with steel joints.
400 " 5 " " " "
600 " 4 " " " "
800 " 3 " " " "
Special connectors—6-in. to 5-in., 5-in. to 4-in., 4-in. to 3-in.
6-inch to 3-inch steel driving shoes.
2 15-foot core tubes.
2 special connectors of boring rods to core tubes.
2 " " of rods to 3-inch tubes.

of the core-tube with a view to take the weight of water off the whole core, and possibly some of the Boring Companies adopt this practice now.

The diamond used is in a different state from the gem, and is technically called *carbon*, *carbonate*, or *carbonado*, but the substance is really carbon in an imperfect state of crystallisation, and this rough diamond, while as hard as the ordinary diamond, does not so easily break. It is black or dark grey in colour, and granular in texture; it is supplied only from the mines of Brazil in the district of Bahia, where it was discovered in 1842; and no diamonds of the same class have been found elsewhere, except, it is said, in Borneo. Experiments have been tried with other substances, such as bort, or rough splintery diamond, but none compare with the carbonate in fitness to cut hard rocks. A piece of carbonate the size of a large pea, will cut a hole in sandstone, half a mile deep or more, without sensible abrasion to its surface, and although in harder rocks than sandstone the abrasion is more, it is still very slight. The loss on the crown from the act of drilling half a mile is very slight, but the diamonds get broken from other causes. If a jar breaks one, it causes more jarring and leads to the breakage of others, and as the diamonds are costly, the loss to the Boring Company becomes serious.

A great advantage of the diamond-drill boring is that the hole is kept true and vertical, and this cannot be assured in other systems. Then the cores frequently contain whole and uninjured fossils characteristic of the strata, which but for this evidence would remain uncertain of classification. Thus the sub-committee of geologists connected with the Sub-Wealden bore-hole near Hastings, state that the determination at which they arrived with reference to the position of the strata was "due to the manner in which large cores were brought to the surface by the Diamond Rock-Boring Machine, so that a number of fossils were obtained entire, the species of which could be accurately determined."

The percentage of cores obtained by the diamond drill seems to vary from about 60 to 90 in bore-holes of small size. The boring rods are subject to accident, occasioning delays, such as breakages and jams, but they are not more difficult to deal with than are the breakages inseparable from other modes of boring. When a trial of the diamond was first made to ascertain its suitability for the purpose of boring, it was given a percussive motion, but experience soon showed it to be better adapted for abrasion.

On the introduction of the system the holes drilled were under 2 inches in diameter, and the cores produced were about $\frac{3}{4}$ of an inch in diameter. Consequently difficulty arose in securing cores in soft strata, although the result in hard rocks was quite satisfactory. On this account and also to give room for lining tubes, the bore-holes have been gradually increased in size. Where they have been of large size, 100 per cent. of cores of the strata have been obtained even with the softest of rocks. For instance at Caerphilly, in South Wales, in a hole put down to a depth of $1{,}007\frac{1}{2}$ feet in 1874, the cores yielded showed a complete section of the strata passed through, and the samples of coal were satisfactory.

Bore-holes of 26 inches in diameter yielding cores $23\frac{1}{2}$ inches in diameter have been put down by the diamond drill.

A Diamond Rock Boring Co., prospecting for minerals, usually stipulates that the employer must find water and engine power, or, instead of the latter, pay a fixed sum per month, according to the size of the engine, but the company find everything else that is necessary. Boring to any specified depth is not guaranteed, but the company uses its best efforts to reach it. Unless a certain prescribed depth be bored the employer must pay the cost of carriage to and from the boring site or sites, and all damage caused in moving the machinery on and off the ground.

The schedule of prices is in arithmetical progression over certain zones of depth, and only apply to strata of ordinary character.

One of the deepest bore-holes known is that made at Schladebach, Leipzic, to a depth of 5,736 feet. It was commenced 11 inches in diameter in the Trias, and after passing through the Permian entered the Old Red Sandstone, the size of the hole having been reduced to 1·22 inch at the bottom.

The following very interesting particulars of this borehole appear in the *Colliery Guardian* of April 22nd, 1892.

"At the closing *séance* in 1891 of the Société Industrielle de Mulhouse, M. Charles Zundel gave some particulars, gathered from two reports sent to the society by the German Mine Department, of the deepest borehole which has yet been put down—viz., that at Schladebach, near Kötschau in Merseburg, Prussia, which attained the great depth of 1,748 metres, or 956 fathoms.

"The boring was begun in August, 1880, by the Royal Division of Prussian Mines, and finished in the autumn of 1886, having occupied 1,247 actual working days, with a mean daily advance of 1·4 m. (4 feet 7 inches), and at a total cost of 212,304 marks (£10,615), or say, £6 per yard of boring. A 25-horse power portable engine sufficed for the motive power necessary to work the boring rods and also the pumps, a tower 27 m. (89 feet) high, permitting of 20 metres of tube lining being raised or lowered at a time.

"The initial diameter of the hole was 280 mm. (11 inches). After 20 m. of sand, gravel, and marl, passed through by the system of *Schappenbohrung*, or tube boring, the variegated sandstone was struck, which was attacked by the *Hohlfreifallinstrument*, or free-fall apparatus, a cast-steel drill with end in the form of a cross with guides, water flowing down outside the tubular rods and rising up the inside. At the depth of 57 m. (31 fathoms), the lining could not be got to descend further, so an attempt was made to go on without lining; but the *débris* detached from the sides of the hole rendered it necessary to put in a tube lining of 230 mm. (9 inches) diameter. At 164 m. (90 fathoms), strata of gypsum and anhydrite were encountered, and also water charged with sea salt. At the depth of 175 m. (95 fathoms), the free-fall apparatus was abandoned for rotary boring by means of a soft iron crown, of 210 mm. (8¼ inches) diameter, set with borts or carbonate diamonds, producing cylindrical cores of 140 mm. or nearly 6 inches diameter. In the dolomitic limestone, it was found impossible to detach one of these cores; and the crown became fixed so fast that it became necessary to draw up the rods and bore into the core with a small crown, in order to detach it, a work that occupied three weeks.

"After several accidents and difficulties—such as the rods bending and having to be cut and lifted, a crown sticking in the bottom of the hole and having to be perforated and ground up by the free-fall apparatus, or a tool falling by accident into the hole and necessitating the cutting of the rods at great depths—the depth of 1,070 m. (585 fathoms) was reached, the diameter of the hole having been reduced seven times. Between this depth and that of 1,724 m. (943 fathoms), the hole was continued with a diameter of 48 mm. (2 inches), yielding cores of 23 mm. or less than 1 inch diameter. The Old Red Sandstone had been passed through between 327 and 1,630 m., when the Devonian rocks were traversed, without, however, encountering a seam of coal, which might have been met with at that geological depth. A stratum of cuprose schist, struck at 326 m., was not considered rich enough to warrant its being worked.

"Nevertheless, the hole was continued with a purely scientific object, and at the beginning of 1886, a lighter tubing of 33 mm. (1⅜ inches) diameter, was put in, in which boring was carried on with a crown of 31 mm. (1¼ inches) giving cores of 12 mm. (½ inch bare) only. But finally, after successive accidents—such as breakage of the rod and fall of the crown, and breakage of the screw of the core-extracting apparatus—at the depth above mentioned it was found necessary to stop the work, which could only have been continued at a great expenditure of time and money. The following are the rocks passed through, with their depths and thicknesses:—

	Thickness Metres.	Depth Metres.
Vegetable earth	0·6	0·6
Sand	4·27	4·87
Clay	17·76	22·63
Variegated sandstone	141·89	164·52
Gypsum and anhydrite	16·03	180·55
Permian limestone (Zechstein)	46·36	226·91
Gypsum	10·41	237·32
Anhydrite	89·19	326·51
Cuprose schist	0·9	327·41
Old red sandstone	1,302·59	1,630·
Devonian rocks	118·4	1,748·4
	1,748·4	

"Careful thermometric observations were made as the work proceeded, and naturally delayed its progress considerably. From a depth of 1,200 m. they were continued regularly every 30 m., and before the tube lining was put in, so as to avoid any disturbing influence from conduction, while, to counteract that of water-currents, the thermometer was immersed in a fixed column of water between two clay plugs, and left there for sixteen hours. To prevent the glass from being broken by the great pressure, it was enclosed in an iron case; and there were always three superposed thermometers from which a mean of the readings was taken. The observations of the first 1,200 metres were taken subsequently. The observations went to show that the increase of temperature does not diminish with depth, as had before been imagined, but follows a constant arithmetical progression. The last reading taken, viz., at 1,716 m. (938 fathoms) was 45·3° Réaumur, or 56·6° Cent. or 134° F., which shows an increase of 1° Réaumur for every 46·09 m. equivalent to 1° Cent. for every 36·87 m. At this rate potassium, the fusion point of which is 48° Réaumur = 60° Cent. = 140° F., would melt at a depth of 1,845 m. (1,009 fathoms), and grey foundry pig (1,240° Réaumur = 1,550 Cent.) at 56,775 m., or about thirty-five miles, while the greatest temperature of blast furnaces, which, according to Schérer is 2,230° Réaumur (4,037° Cent.) would be attained at a depth of 104,708 m., or a little over fourteen geographical miles.

"M. Zundel concluded his interesting communication by calling upon the Société Industrielle to urge the putting down of a deep borehole in Alsace-Lorraine, or the deepening of two Hasenrain and Dollfus holes in order to prove the existence of coal or bituminous shales yielding petroleum, or even to demonstrate practically, that a source of heat and power exists beneath our feet, that shall maintain warmth in our bodies, and keep our mills and factories going, when all the coal deposits shall be exhausted."

The deepest boring is that made by the Prussian Government at Paruschowitz, near Rybnik, in Upper Silesia, particulars of which were communicated to the Société Industrielle de Mulhouse, by M. Charles Zundel. This borehole, begun on January 26, 1892, was finished on May 17, 1893, after having attained a depth of 6,572·7 feet, or 2,003·34 metres. The surface level of this boring being 498·7 feet higher than that of Schladebach, it only penetrated 338 feet nearer the centre of the earth than the Schladebach boring. The Paruschowitz borehole proved the existence of eighty-three coal-seams, many of them being of considerable thickness, and giving a total thickness of 89·5 m., or 293·6 feet. Begun with a diameter of 12 inches, and lined with a tube $\frac{13}{32}$ inch thick, the boring was put down to a depth of 230 feet, from which point to 351 feet the diameter was reduced to 10 inches. At this depth the blue marls encountered became so compact as to necessitate the use of the diamond drill for the further boring. Under the action of the water injected into the hole, the marl swelled and subjected the lining tubes to such compression that it was found necessary to gradually reduce their diameter. Shifting sand, met with at the depth of about 656 feet, also caused great difficulty.

The greatest difficulty of all, however, was the great weight of the boring-rods as the depth increased. Although a reduction in the weight of these was obtained by substituting steel for iron, yet at the depth of 6,560 feet the total weight of the tools was 13·7 tons. With such a weight and so great a length, ruptures of the rods were frequent, and an accident of this nature finally stopped the work. When the diameter was $2\frac{23}{32}$ inches, and that of the cores brought up $1\frac{25}{32}$ inches, about 4,500 feet of rods fell to the bottom, and became jammed in an unlined portion under the tubing, so that it was impossible to withdraw them. Consequently the borehole was abandoned. During the 399 working days a daily advance of a little over 16 feet was made. The total cost of the boring was £3,761, or 34s. 4d. per yard. Temperature observations made showed 12·1° Cent., or 54° Fahr. at the surface, and at a depth of 6,572·7 feet the temperature reached 69·3° Cent. or 157° Fahr. This gives an average increase of 1° Cent. for 34·14 m., or 1° Fahr. for 63 feet, differing from that observed at Schladebach, where the mean increase was 1° Cent. for a depth of 35·45 m., or 1° Fahr. for every 64 feet 7 inches.

During thirteen years the Prussian Government have made 400 boreholes for exploring purposes, of an aggregate depth of 80 miles, and at a cost of £650,000.

A bore-hole at Sperenberg, Berlin, was made by means of rigid rods to a depth of 4,170 feet, and took $4\frac{3}{4}$ years to accomplish. A boring for salt near Lubtheon, in Mecklenburg, was carried out by a diamond drill to nearly 4,000 feet, and was completed within 6 months. The boring was not only wonderfully successful in the speed with which it was accomplished, but was further distinguished by the hole yielding 100 per cent. of cores, one specimen of rock-salt being over 20 feet long.

No virgin property of a size to require any borings on it should have less than three, and it may be wise to put down more; but in a neighbourhood free from faults and dykes, and proved by winnings all round, none will be necessary. It is natural to let the first be nearest the rise. It must be remembered that the depths as proved by boring through inclined strata do not give the true thicknesses of such, as the bore-hole is perpendicular and the line of stratification does not often form a right angle to it. The true thickness of a stratum is found by multiplying the thickness as proved in the boring by the cosine of angle of dip. Thus, if a stratum was proved to be 1 fathom thick in the boring where the dip was at an angle of 30°, the true thickness would be ·866025 of a fathom or 5·19615 feet. Tables of incline measure may be obtained showing the comparative lengths of the hypothenuse, horizontal, and vertical legs of a right-angled triangle for every degree of the quadrant.

Supposing three bore-holes to be put down on a property at equal distances apart, say 500 yards. No. 1 being nearest the rise and proving the coal at a depth of 15 fathoms, No. 2 fully to the dip of No. 1, as well as can be ascer-

No. 4 is S. 40 W. of No. 1, 250 yards distant from No. 1, and proves the coal at $64\frac{1}{4}$ fathoms, all the bore-holes being at the same surface level. The direction of full dip and its extent may be determined as follow.

By plotting as shown in Fig. 17, or by trigonometry, it will be found that the distance between No. 2 and No. 3 would be 383·62 yards. The difference in level between the seam at No. 2 and No. 3 bore-holes is 220 yards — 160 = 60 yards and $\frac{383 \cdot 62}{60} = 6 \cdot 39$, that is the seam dips from No. 2 towards No. 3 at the rate of 1 in 6·39. Now, to find the level course of the seam, follow along the course from No. 2 towards No. 3 until a point is reached 5 fathoms or 10 yards below the seam at No. 2 bore-hole, because the difference in the depth between No. 1 and No. 2 is 85 — 80 = 5 fathoms. To gain 10 yards of fall along a course dipping 1 in 6·39 follow it for a distance of $6 \cdot 39 \times 10 = 63 \cdot 9$ yards. Therefore a line drawn from No. 1 bore-hole to a point in the line connecting No. 2 and No. 3, and 63·9 yards distant from No. 2, gives the level course of the seam, and by means of plotting or from the computed value of the angles, it may be seen that the bearing from Nos. 2 to 3 bore-holes is N. 36° 48′ 9″ E., and that of the level course of the seam S. 4° 52′ 29″ E. from No. 1. The full dip would be at right angles to this or N. 85° 7′ 31″ E. To get the amount of dip, draw a line connecting No. 3 bore-hole with the level course of the seam and at right angles to it. Such a line would measure 212·6 yards, and since the difference of level in the seam between the two ends of it is 110 fathoms — 85 = 25 fathoms, or 50 yards, we have $\frac{212 \cdot 6}{50} = 4 \cdot 25$, so that the full dip would be 1 in 4·25 in a direction N. 85° 7′ 31″ E.

In the same way it can be shown that a line drawn from No. 4 bore-hole to join the line representing the level course of the seam and at right angles to it would measure 176·4, and taking the dip at 1 in 4·25 as proved on the other side of the level course $\frac{176 \cdot 4}{4 \cdot 25} = 41 \cdot 5$ yards or 20·75 fathoms, and 85 — 20·75 = 46·25 fathoms as the depth of No. 4 bore-hole to prove the seam, which is the depth given, a reasonable conclusion from which would be that the coal on the property was fairly free from faults. It is only right to say that these calculations are often upset by faults running between the position of the bore-holes, and when these are suspected to be on the property, more bore-holes should be put down before the sinking is decided on.

In coalfields having carboniferous rocks exposed on the surface the outline of the basin may be traced by following the course of outcrop of the upper edge of millstone grit or other strata underlying the carboniferous, if conformable to them. Any estate being within the coal measure or productive area as shown on a geological map would usually be expected to comprise coal seams. If situated to the rise of the lowest seam's outcrop, no coal would be available, although the rocks might belong to the lower members of the carboniferous period.

Where a coalfield is known to exist over a certain extent of country by its coal-bearing portion of the carboniferous rocks outcropping in some places and disappearing in others under Permian or newer rocks, which lie unconformably on them, there is much room for speculation as to the extent of the coalfield. Under such circumstances the search for coal of course becomes far more intricate.

As an instance of the difficulties of such a search may be cited that of the South Staffordshire Coalfield under the "red rocks."* A fault which exists in the neighbourhood of West Bromwich appeared to cut off the coal measures

* See *Colliery Guardian*, May 14th, 1875.

The red rocks at Bullock's Farm and other West Bromwich sinkings were proved by the Sandwell Park sinkings to be coal measures instead of Permian as previously supposed. The Sandwell Pit is a mile to the eastward of Bullock's Farm. As fixing the position in the pit where the coal measures were struck, fossil plants found at a depth of 110 yards in red measures were stated by authorities on the subject, to be Permian. At a depth of 200 yards a thin seam of coal 7 inches thick was proved. Above the coal was a black shale containing fossil plants, some of which were identical with those found at 110 yards, whilst beneath the coal was a bed of fire-clay containing *Stigmaria*. At 230 yards a second coal seam 6 inches thick was struck, and a third of the same thickness at a depth of 244 yards, while at 418 yards the thick coal, the object of this patient and enterprising search, was attained. The shaft therefore entered the coal measures at a depth of 110 yards or less, showing that there is a greater thickness of coal measures over the thick coal at Sandwell Park Colliery than was proved by the pits previously sunk through the Permian strata.

As another instance of a successful search for coal we may mention that near Dover, in Kent. This discovery is full of interest to the geologist, and as it is within a workable depth it may create a very important industry in the South of England.

Previously, no true coal had been found in England to the south of a line joining Bath and Stamford and continued to Great Yarmouth, though lignites were known to exist in the Wealden strata of Kent, Surrey, and Sussex. An examination of the formations (which are much newer than the carboniferous) prevailing over most of the area indicated would of itself not appear hopeful, yet geologists have reasoned for many years that there was a possibility of finding coal under these newer formations and within a reasonable distance of the surface.

About the year 1855 Mr. Godwin-Austen started this theory, but Sir R. Murchison disputed it. In 1871 the Royal Coal Commission published its Report, including a very elaborate communication from Professor Joseph Prestwich, who had accepted the theory suggested by Mr. Godwin-Austen.

The main feature which gave rise to the theory, is the fact that a great axis of elevation extends from the South of Ireland to Westphalia, a distance of 850 miles. Its existence can be traced at one extremity from Ireland to Frome in Somersetshire. Along the strike the lateral pressure elevated the fractured ends of the strata into ranges of hills, of which there now remain two remnants, viz., the Mendips, in Somersetshire, and the Ardennes, in Belgium. The Ardennes have many features in common with the Mendips. On the northern flanks of both ranges the palæozoic strata are highly inclined as if tilted up from the same disturbing cause, and both are overlaid by newer strata reposing horizontally on them. From Westphalia to the North of France there is a series of coalfields, the more important being those of the Ruhr, Aix-la-Chapelle, Liége and that of Charleroi, Mons, and Valenciennes, whose longer axes succeed one another along the same line of strike. In all these the coal measures are highly inclined on the south against the Mountain Limestone, while on the north they disappear under newer formations. Westward of Valenciennes no palæozoic strata are exposed on the surface, but the coal measures have been proved to pass beneath the chalk and tertiary strata to Enquin within 30 miles of Calais, whilst further west the older rocks subtend the chalk. A boring at Calais, however, proved carboniferous strata at a depth of 1,032 feet from the surface after passing through the chalk.

Passing to a point near the other extremity of the same axis of elevation, a striking resemblance in the geological features is seen. From Milford Haven to Tenby are found old red sandstone and mountain limestone in a contorted condition, resting on which to the northwards is the Pembrokeshire coal-field, considerably disturbed, though not to the same extent as the old red sandstone and mountain limestone; from which it may be inferred that the forces disturbing the strata decreased in proportion to their distance from the line of upheaval. Proceeding eastward along the axis of elevation to the Somersetshire coal-field, the Mendip Hills give evidence of the same tilting and denudation of the older rocks observable in those of the Ardennes range, and on their northern flank are covered by mesozoic strata laid horizontally over the highly inclined coal measures and mountain limestone.

The South Wales coal-field is not hidden by newer strata, but a very large portion of the Somersetshire coal measures is covered by permian, lias, and oolite, and proceeding eastwards these in turn are overlaid by the chalk. The Pennant rock, yielding but few seams of coal, and the associated sandstones, shales, and coal seams above, and also below, with the addition of ironstone, form strong features of resemblance between the coal-field of South Wales and that of Somersetshire.

The similarity in the structure of the rocks of the Mendips and Ardennes, and in the direction of their strike, points to their being due to a common cause, namely, an upheaval of irresistible might and affecting an enormous mass of the earth's crust. The upheaval was too powerful for any opposing force to interrupt, and its continuity under the newer rocks which hide it from view in the South of England, is almost a matter of certainty.

The mountain limestone is continuous throughout the line of elevation, and wherever the older rocks come to the surface, the coal measures show signs of having covered the mountain limestone at the time of disturbance and to have accompanied the latter in its foldings and movements. This fact, taken in conjunction with the unconformability of the permian, enables geologists to fix the age of the axis of elevation, as being after the coal measures and before the permian. The strata of the coal measures were altered from their originally horizontal position by subterranean forces, causing the line of elevation. As the latter proceeded in a slightly irregular, wave-like direction, mostly east and west, the rocks were doubled up or folded into a number of anticlinal and synclinal curves proceeding northwards from the main line of disturbance. The effect of this would be to change what was once a gigantic coal-field into a number of isolated basins separated by ridges of the older rocks, similar to those known to exist. A proof of the series of existing coal-fields having once formed part of a larger, lies in the fact that their edges contain beds of equal thickness with the more central portions. Were they separate coal-fields there would be a thinning out of beds near old lines of shore.

If this reasoning be accurate, then the amount of coal which will be found underlying the newer formations in the South of England will depend entirely on the amount of denudation the coal measures have been subjected to over that area anterior to the deposit of the newer strata. The borings hitherto made point to a southern slope of the surface of the palæozoic rocks in the south of England; the denuded surface can of course give no indication of the dip in the rocks beneath.

Professor Hull shows in a section from Gloucestershire to Oxford that all the newer rocks below the great oolite thin out rapidly to the S. E., the total thickness of the overlying rocks diminishing from 1,880 feet in Gloucestershire to about 600 feet at Oxford. At Burford, in Oxfordshire, coal-measures have been reached at a depth of 1,184 feet.

In 1854–5, a boring at Kentish Town, on the north side of London, passed through tertiary and secondary strata and proved red sandstones, believed by some geologists to be of palæozoic age, at a depth of 1,114 feet. In 1854–7, a boring at Harwich proved lower carboniferous rocks at a depth of 1,026 feet, after passing through tertiary strata and the chalk and gault, the lower greensand, the oolites and the lias being absent. These borings presented the same order of superposition as that at Calais and in other places in the North of France and Belgium.

The borings alluded to were made for water, but in 1872–5, to test the theory of Mr. Godwin-Austen and Prof. Prestwich, a boring was made near Battle, in Sussex, under the direction of the Sub-Wealden Exploration Committee. It passed through 200 feet of Purbeck strata and 1,705 of oolite below, the bottom of which was Oxford clay, and was then abandoned. It proved that the palæozoic rocks did not lie within 1,905 feet of the surface at Battle. In 1886, Sir Edward Watkin, chairman of the Channel Tunnel Company, acting on Professor Boyd-Dawkins' report, which recommended that a bore-hole be made in the neighbourhood of Dover (at the Channel Tunnel works, under the Shakespeare Cliff), gave orders to commence the boring. The site was about 30 miles west of the boring near Calais, which had established the fact that coal measures extended under the newer strata to the French coast line at a depth of 1,092 feet from the surface. After penetrating 500 feet of cretaceous rocks the boring passed through 613 feet of oolitic rocks, when the coal measures were reached, being therefore 1,113 feet below high water mark and 1,157 feet from the surface. The boring, after passing only 27 feet further, met with a good seam of coal on the 15th February, 1890. This result, whilst very encouraging to all concerned, has by no means cleared up all the points connected with the hypothesis started by Mr. Godwin-Austen in 1855, but it has clearly established the fact that at any rate one coal-field exists in south-eastern England, where it is covered by the newer strata, and that within a workable depth. This alone is no slight reward for the patient toil and thought of the geologist. It will doubtless act as an incentive to other trials along the most likely course of the line of elevation crossing the south of England, between Kent and Somerset, in order to search for other coal-fields or to discover the detached portions of what at one time was one immense coal-field, and is believed now to exist in a more or less denuded state. Indeed, a company was registered on March the 27th, 1890, with a capital of £2,000 in £1 shares, to search for coal and other minerals in Kent, Surrey, Sussex, and elsewhere in Great Britain.

In this country, however, there is little encouragement offered to private enterprise in searches for coal of the importance attached to the search instituted at Dover. If successful, it is the landowners in the immediate vicinity who derive the greatest benefit, whether they promoted the search or contributed towards its cost or not. They may make what use they like of the information obtained (without reference to the bold spirits who pioneered the venture or those who carried it through), whether by fixing a high royalty price, or by sinking shafts themselves to work the coal so found. In France, where the minerals belong to the State, private enterprise meets with the encouragement which is due to it, and many agricultural districts have through this means been converted into busy mining centres.

When the law relating to royalties has been revised, searches of national importance may be expected to proceed more rapidly in this country, but all things considered, it is perhaps not surprising that so many years should elapse between the expression of Mr. Godwin-Austen's theory and the result to which it has eventually led.

The boring for coal near Dover has been continued to a greater depth since the above remarks were written. In a paper read before the Manchester Geological Society on February 2, 1894, Prof. Boyd Dawkins gave details of the Dover boring up to that date. It appears that twelve seams of coal, having an aggregate thickness of 23ft. 5ins., have been proved. The first was a seam 3ft.

6ins. thick, struck at a depth of 1,136ft. 6ins. below high-water mark, and described as "bright and bituminous, with sandstone parting," yielding 2ft. 6ins. of good house-coal. At a depth of 1,200 feet, the second seam occurred, but this was only 6ins. thick. Then at 1,229 feet occurred a 2ft. seam, and another of the same thickness at 1,279 feet, both described as "blazing, with specks of iron-pyrites." At 1,311ft. 9ins. the borer touched a seam of bituminous coal, 1ft. 3ins. thick; at 1,433 another seam, 1ft. thick; and at 1,456ft. a 2ft. 6ins. seam of "good house-coal." The eighth seam, described as clean bright coking coal, was touched at 1,570ft., and found to be 2ft. 3ins. thick; the ninth seam, 2ft. 9ins. thick, also clean bright coal, occurred at 1,763ft. 9ins.; the tenth seam, of "clean, bright coal," 1ft. 8ins. thick, was struck at 1,831ft.; the eleventh, a 1ft. seam of hard coal; and, finally, at a depth of 2,177ft. 6ins., a seam of "bituminous coal" was discovered, having a thickness of 4ft.

Having regard to the opinion of Mr. McMurtrie, of Radstock, and of M. Watteyne, a Belgian engineer, Prof. Boyd Dawkins concludes that "the horizon of the south-eastern coalfield is that of the valuable upper measures of Somersetshire, and of the important middle coal-measures of Belgium and Northern France." At the same time he points out that the beds under Dover have an exceptionally gentle dip. It is notable that in the course of the Dover boring a bed of oolitic iron-stone, 12 feet thick, was discovered, and it is probable that this ore, worked in connexion with the underlying coal, may become of great industrial value. A shaft has been sunk 44 feet from the surface to high-water mark, and the boring is continued from the bottom of the shaft.

It will be useful to compare with the results at Dover the number and thickness of seams, and the thickness of coal measures in the French, Belgian, and Westphalian coal-fields as given by Professor Joseph Prestwich in the following table:—

	WEST OF DOVER.		EAST OF DOVER.		
	South Wales.	Somerset.	Mons.	Liège.	Westphalia.
Number of seams	75	55	110	85	117
	FEET.	FEET.	FEET.	FEET.	FEET.
Total thickness of workable coal	120	98	230	212	294
	FEET.	FEET.	FEET.	FEET.	FEET.
Mean thickness of coal measures	11,000	8,400	9,400	7,600	7,218

The English coal measures appear to contain fewer seams than the Continental, although they are of considerably greater thickness. This is partly due to the Pennant rock in the Somerset and South Wales coal-fields, where it is from 2,000 to 3,000 feet thick and contains but little coal. In Belgium the Pennant rock is replaced by productive measures. It frequently happens that there are marked changes in the quality and thickness of the coal seams in the same coal-field, as well as in the associated strata. In districts where these changes defy all attempts at co-relation, the seams are often known by different names at different points in the same coal-field.

The difference shown in the particulars of the coal-fields in the above table, does not therefore prevent our associating the Dover field with them, or exclude the probability that it was at one time continuous with them. This probability, indeed, is strengthened by the coal already proved in the boring, the thin seams of which correspond with those of the North of France, Belgium, and Somersetshire, which belong to the true carboniferous series. Thicker seams will in all probability be met with at greater depths as in the lower series of the Somersetshire coal-field, and also in the department of the Nord. That the Dover field will prove of great value there can be little doubt, even though the smallest number of seams given in Professor Prestwich's table may never be proved and the total coal area may be very limited. The development of this field both in the east and west is awaited with great interest.

The length of the coal-field of Northern France from east to west is about 94 miles. Its width is very irregular, varying from about 11 miles near Douchy to about half a mile at Fléchinelle, Pas de Calais. This coal-field is everywhere masked by tertiary or cretaceous strata. In 1890, it employed 29,000 persons, and produced nearly 8,000,000 tons of coal.

The Somersetshire and Gloucestershire coal-field extends from the north side of the Mendip Hills to Cromhall, near Wickwar, a distance of 26 miles. Its greatest width, from Ashton near Bristol to Twerton near Bath, is 12 miles; gradually diminishing toward the north until at Cromhall it is only 1 mile. Four-fifths of the whole area are covered by the New Red Sandstone, Lias and Oolites. The official returns for the year 1890 for the production of this area, show a total of 1,425,071 tons of coal, beside some fireclay, &c., and 7,982 persons employed.

The greatest length of the South Wales coal-field is 89 miles, from Abersychan on the east to St. Brides Bay on the west. Its width is irregular, but it nowhere exceeds 21 miles, and the extent is taken approximately at 1,000 square miles, 846 of these being exposed, 153 lying beneath the sea, and one square mile covered by newer formations. The quantity of coal worked in this area during 1890, according to official returns, was 29,310,295 tons, beside fireclay, ironstone, &c., and the number of persons employed above and below ground, 109,404.

The importance of the discovery at Dover is emphasized by a consideration of these facts. No doubt, a considerable area of the coal-field lies under the sea between Dover and Calais, the minerals of which under the foreshore are the property of the Crown. Interposed between the bed of the sea and the coal measures are impervious strata, of sufficient thickness to protect any coal workings which may be carried on. At Whitehaven, coal has been worked for two miles under the sea with 300 yards of cover below the ocean bed. This distance might be indefinitely extended but for the cost of haulage, and the still greater difficulty of proper ventilation. As it is not practicable to sink shafts at sea, computations relating to under-sea coal are usually limited to 3 miles from the shore. But in the case of the Dover coal-field, unless thicker seams are met with, the working of under-sea coal will be at a greater cost per ton than in most of the other coal-fields in this country. This no doubt will be duly considered in fixing terms which will create a precedent in the district. No insuperable difficulties are likely to be met in sinking shafts for coal through the cretaceous and oolitic strata overlying the coal measures, for chalk of a similar character has been successfully pierced in the Belgian and French sinkings. In these cases, considerable difficulties arose at times owing to large quantities of water in the thick tertiary and cretaceous sands. Again, in Somersetshire, although similar difficulties were encountered in the sinking of shafts through oolitic and triassic strata, the coal measures were successfully reached and pierced to the required depths. The lower greensands of the London basin are known to

The Ecclesiastical Commissioners, representing a Department of the State, own considerable areas of land, as do also, to a smaller extent, other public bodies and corporations, such as universities and colleges; municipal corporations; charities and hospitals; and the Board of Admiralty, who possess the forfeited estates granted to Greenwich Hospital, the minerals under which are leased by them on terms that are usual in the district, or that are within their power of granting.

All gold and silver belong to the Crown, but copper, tin, lead, and the other base metals are not mines Royal. In early times the Crown claimed all minerals.

Only one gold mine is now in active operation in the United Kingdom, viz. the Gwynfynydd or Morgan Gold Mine, near Dolgelly in North Wales, belonging to the Morgan Gold Mining Company.

The following are the royalties chargeable by the Crown in the British Isles:—

If lease granted to owner	$\frac{1}{30}$th	of gross produce.
,, ,, ,, others	$\frac{1}{20}$th	,, ,,
,, under Crown lands	$\frac{1}{15}$th	,, ,,

These rates have not been invariably the same, and may possibly undergo further alterations.

The Crown does not, as a rule, give licenses to search for gold on private lands unless a gold miner has first obtained a license from that owner to work the baser metals which belong to him. There seems to be some doubt as to the right of the Crown to enter upon the lands of private owners, without their permission and to erect machinery thereon or divert the water courses, or if the right exists it is never exercised. The gold found in Wales is associated with the baser metals running in the same veins, so that it may be impracticable to work one but not both. If a mine is worked exclusively or mainly for the gold or silver obtained it is of course a mine Royal, and that even though copper or other base metal may have been the original object. The gold and silver may be claimed by the Crown by virtue of the right of pre-emption of the ore brought to the surface. The lessee of the mine would exercise his previous right of mining ore and bringing it to the surface, but as long as the base metals were present in merchantable quantities, they would be paid for by the Crown at certain fixed prices, and possession of the rest claimed. When not present in merchantable quantities there can be no pretence that the mine is not worked primarily for gold or silver, but it may sometimes be extremely difficult to decide whether the base metals are sufficiently plentiful to constitute merchantable quantities, owing to fluctuations in the market. If the Crown does not exercise its right of pre-emption, the precious metals together with the baser ones are taken possession of by the lessee of the mine. It is believed that the right of pre-emption has never been exercised.

The veins carrying metallic ores may continually change their character as to quality and yield.

Owing to the dual ownership set up on private lands, the gold miner has practically two royalties to pay. In some of the Australian quartz no baser metal of any market value is found, and the quartz is crushed for the sake of the gold only.

There are no mines which can be classed as silver mines working in this country. It is true that silver is obtained, but the mines yielding it do not work chiefly or solely for silver. It is contained in varying degrees of richness with lead ore and obtained almost exclusively from this source. In Crown leases for lead mines, the royalties charged are determined by the amount of silver in the lead, but in other cases the yield of silver is so small that the Crown foregoes its claims.

Freehold properties may be inherited or acquired otherwise. The former are either inheritances absolute, called fee simple, or inheritances limited which are either qualified or base fees, or fees conditional. Fee simple is the largest estate or interest which is possible in landed property; the owner is free to dispose of it to whom he pleases in his lifetime by deed, or by will and if he dies intestate it descends to his heirs.

A qualified or base fee has some qualification or limit annexed which may determine the tenure of the estate.

A conditional fee restrains to some particular heirs exclusive of others. A person who holds land which by deed or will has been tied up so as to be descendible to the heirs of his body is called a tenant in fee tail and his estate a fee tail.

A freehold not of inheritance is an estate held by the owner for his own life only, or during the life of some other person, or until the occurrence of some uncertain event.

Copyhold tenure is a survival of the feudal ages, and originated from the king in olden times granting large estates to his lords, who thereupon let out to tenants parts of their lands in return for services rendered. In the course of time money payments took the place of those services. At the present time the title to copyhold property is altogether constituted by custom. It is supposed in law, to be held at the will of the lord of the manor; the mines and minerals belong to him; without his license the property cannot be let on lease for more than a year, unless there is a special custom of the manor to the contrary.

Copyhold tenure has certain liabilities peculiar to itself in the shape of heriots and fines; it is also subject to quit-rents. A heriot is the render of the best beast or other chattel (according to the custom) to the lord on the death of a tenant. Fines may be due on the death of a tenant, and on the alienation of the land: sometimes they are fixed by custom and are alluded to as "fine certain," sometimes arbitrary and alluded to as such. Where it is arbitrary the lord of the manor is allowed to inflict what fine he thinks fit upon the descent or alienation of the land, but in practice it has been decided that double the annual improved value of the property, after deducting the quit rent, is a reasonable sum; and no more is allowed to be taken unless under very exceptional circumstances. A fine is the sum payable to the lord of the manor upon any sale, mortgage, or other transfer of the property, or by the heir to whom the property descends on the death of the owner without will. In negotiating these transfers, beside the fines, stewards' fees have to be paid, which in some manors are very extortionate.

Quit rent, in law, is a small rent payable by tenants of manors which leaves them free of other charges.

In many copyhold manors there are special customs not generally known, and applicable only to a particular manor.

The owner of the land in fee-simple who received the property intact, and who has not disposed of any interest he has, claims everything lying under it in a perpendicular direction from the surface to any depth.

A mining company with a large capital may acquire by purchase the whole of the surface with the minerals they intend working, but the heavy outlay which this necessitates, and the length of time which must elapse before a dividend can be earned, preclude the general adoption of this course.

Minerals are in some instances worked by the hereditary or other owners for their own profit, but usually they are leased from the owners (the lessors) to others (the lessees) whose interest it is to produce them skilfully and economically, and then dispose of them so as to yield a profitable return on the capital.

On an unproved estate, or where there is much room for speculation as to the existence of minerals in paying quantities, a *take-note*, or license to search, is first granted by the owner to an applicant. Usually take-notes are for a period of from one to three years, and occasionally for five years. A nominal rent is payable half-yearly, and a provision is inserted whereby the prospector, if successful, is increased royalty rent, which he may have been able to obtain in a more flourishing state of trade than existed at the date of his lease.

The lease is subject to forfeiture if the lessee neglects to pay his rent within 40 or 60 days of its becoming due, or on any breach of covenant, or on his becoming bankrupt when the lessor may re-enter, but this provision is frequently modified by others.

A royalty or lordship on the tonnage is paid, ranging in Northumberland and Durham from 2½*d.* to 10*d.*, the average being about 5*d.* The difference in prices fixed for royalties arises from the leases having been granted at times when trade was depressed, prosperous, or unduly inflated; or from greater difficulties presented in working some mines than others; or from variety in the quality of the coal-seams to be worked; or from the difference in the depth at which they lie; or from the geographical position of the property under which the seams are to be worked. In an entirely undeveloped district, without railways or other suitable means of transport for the coal, a lessee would probably succeed in getting an unusually low royalty price fixed. If the coal to be worked is subject to a wayleave rent, it influences the royalty fixed. At the same colliery the royalty rent may vary on the different seams worked, in order to meet the increased difficulties and cost of working thin seams or those having other disadvantages such as a bad roof and floor. Where, however, there are a number of competitors for a property, the lessor of course obtains a high royalty rent.

Fireclay and ironstone where worked are subject to royalties in precisely the same way as coal.

In some leases the royalty is paid on all coal raised, in others a certain percentage is allowed to be deducted for colliery consumption. In Monmouthshire, if a lessee has power to gob any portion of the small coal, his tonnage royalties are made proportionately higher. In Northumberland and Durham leases the rent is "tentale" instead of on the tonnage, a ten sometimes consisting of 48·583 tons and sometimes of 50 tons. In these counties as the small coal is of much less value than the round, it is subject to a separate tentale rent which varies from a half to a quarter of the tentale rent on the large coal.

In order to act as an incentive to diligent and energetic working of the minerals, a *certain annual rent* is fixed, which is termed dead-rent—so that in periods of depressed trade, when the usual yearly sale is not reached, or the production is reduced owing to unexpected difficulties, so that the royalty does not amount to the certain rent, that rent is paid to the owner and there remains what are termed "short workings." These are the deficiencies for which the fixed rent necessitates payment in advance, and may go on from year to year. On the other hand, if the quantity of coal worked in any year exceeds the standard fixed to make up the certain rent, the excess is called "over workings." Usually power is given in the lease to make up "shorts" during the whole term of the lease, that is, the short workings of any year or years may be made up in any subsequent year or years of the term, though in some instances the shorts may only be liquidated during triennial, septennial or other fixed periods. If quantities in excess of the dead-rent are worked in any one period into which the lease is divided, they are not allowed for in the next, but a fresh start is made for each period. If there are overpaid dead-rents at the expiration of the lease the lessor retains them.

The certain rent reserved in leases is determined by the number of acres in the property or the probable output of the colliery, and is imposed as a guarantee that the lessee will work the minerals under the lands leased and not keep them locked up. The surface area to be leased and the probable thickness of the seam over that area enable a calculation to be made as to the probable quantity of many years before any coal can be obtained from those areas furthest from the shafts, the consequence of which is that immense sums of overpaid rents are paid by the lessees, to their serious disadvantage if not actual loss. In some cases these large sums of overpaid rents have accumulated through lessees taking a large number of properties with no immediate intention of working, but rather to prevent competition from others taking them. Consequently they lie dormant for many years before they are worked, during which overpaid rents keep on increasing.

When the coal in the original royalty on which the pits are sunk is nearly exhausted, the lessee of a single property, who has erected valuable works and is pressed hard by reason of a diminished coal area on which the dead-rent continues payable, may seek to utilise his capital by taking adjoining properties. As a result the lessor of the original royalty on his part tries to obtain the best terms he can exact for wayleave rent, if this item has been entirely omitted from his lease.

The coal under the foreshore of the sea and under the foreshore of tidal navigable rivers usually belongs to the Crown, and the under-sea coal can only be brought to the surface by means of shafts situated on property adjoining or near the foreshore, the foreshore being that portion of a river or ocean bed between high and low tides.* From their favourable geographical position, the owners of this property are able to demand a somewhat high wayleave rate on the under-sea coal, as much as $1\frac{2}{3}d.$ per ton being paid in Northumberland and Durham, where it is considered a matter of arrangement that the Crown takes $\frac{2}{3}$rds and the land-owner $\frac{1}{3}$rd of the gross rent of $5d.$ per ton paid by the lessee. Before taking it, however, the terms of the wayleave are well known by the lessee, and he urges this as an argument for reducing the royalty payable on the under-sea, or similarly unfavourably placed coal to as low a figure as possible so as to enable him to work it.

Practically, the royalty owner who is unfavourably situated generally has to pay the wayleaves demanded by the owner, who holds the key of the position. If the circumstances of the lessees are such that they urgently require the coal, hoping thereby to obtain a return on their capital outlay, and the owner of the unfavourably placed royalty remains firm in his demand for a full royalty value, the whole of the wayleave burden may have to be borne by the lessees, more especially if the coal underlying the property is accessible to the workings of two or more colliery companies who become competitors for any particular area.

Outstroke is the privilege of breaking the barrier and working and conveying underground the coal from an adjoining royalty. A lessor granting a lease of his property may insist on an unbroken barrier of coal being left next its boundaries. Another whose lessee may be anxious for a larger taking than the lessor owns, grants this outstroke privilege, which gives facilities for the lessee to work the adjoining coal and bring it through to that on which the shafts are sunk. For this privilege a rent in addition to the wayleave rent is sometimes paid.

Instroke rent is a charge in addition to the ordinary royalty rent made by the owner for the privilege granted to the lessee of going underground into his workings from an adjoining property, and is made on all coal extracted therefrom and raised at a shaft on other property. This rent is not usually charged by lessors

* Ancient grants from the Crown in some instances conferred privileges of an extensive description on the lord of a manor upon the seashore, such as the exclusive right to take sand, stone, and seaweed, besides wreck of the sea and fishery rights. The seashore, therefore, may be claimed as a portion of a conterminous manor. Where the claim cannot be proved by adducing an actual grant, or by evidence of acts of ownership, the seashore and the minerals thereunder belong to the Crown.

of small properties, nor always by owners of large ones. Where it is charged, it is upon the plea that the lessee is not put to any expense in sinking shafts on a property of considerable extent.

A shaft rent is paid for the privilege of raising coals at a shaft which have been obtained from an adjoining or outlying property, or is charged by a lessor to his lessee for the use of shafts sunk before his lease was granted.

This rent is not invariably demanded where these privileges are given.

Aircourse and watercourse rents are paid for the privileges of conveying air or water to or from an adjoining property and inwards or outwards through another property along roads which may have to be carefully preserved, thus deferring the working of a certain portion of coal next them.

Aircourse and watercourse easements on the various properties of one lessee may be very useful, but should always be kept distinct from each other. Many owners object to watercourse facilities through their properties on account of the danger of flooding. Each colliery usually deals with its own water as a whole, and for this reason barriers of coal are left by the lessee next the boundaries of his takings. If this were not done there would be no protection from water to coal workings in a property lying to the dip of another, so that although the lessor of the colliery property situated to the rise may not consider barriers next boundaries to be necessary, the owner of the coal situated to the dip would naturally expect and insist on having barriers left, so as to prevent the water from flowing to his coal. In South Staffordshire the difficulties of each colliery pumping its own water have led to a scheme being carried out by the South Staffordshire Mines Drainage Board under an Act of Parliament for dealing with the district as a whole and pumping the water at common pumping stations, for which purpose the barriers between collieries are cut through.

In fiery districts, the ventilation of each colliery is much better kept separate from that of others, and clauses have been inserted in leases in order to ensure this. By this means the damage resulting from an explosion is confined to the colliery in which it occurs.

Usually, in England and Wales all buildings necessary for the working of a colliery are erected by the lessee. A clause is usually inserted in the lease which enables the lessee to take possession of lands for building or other colliery purposes, such as a spoil bank for tipping rubbish on, by paying double the agricultural rent for the surface damage. At the end of the lease he must restore the soil or pay its value. The buildings, such as engine and boiler houses, coke ovens, offices, workshops, stables, store rooms, &c., are for the most part placed close to the pit's mouth, and being constructed of brick or stone, revert unconditionally to the royalty owner at the termination of the lease, as do also the shafts and permanent underground works for keeping the mines open. If the minerals are all worked out, these buildings, &c., may be of no value whatever to the lessor on his obtaining possession of them.

The lessor allows the lessee to take the clay, stone, and sand necessary for the building from the land without payment. The lessee endeavours to erect such structures as will last the term of lease. At its expiration the machinery still belongs to the lessee, but the lessor has the right to purchase, and if no agreement is come to between the two interested parties as to the value, it is settled by arbitration. Engine-beds are usually removable by the lessee. The buildings are often not kept in an efficient state of repair towards the end of the lease, and at its termination may be further dilapidated by the removal of the lessee's machinery, so that they are then not worth much.

In Northumberland and Durham each married workman is allowed a house, rent free, and has an allowance of free coal, paying *6d.* per fortnight for the leading only, but these privileges are considered part of his wages. In order that these houses may be near the work, they have in many cases been built on the property leased, and then form part of the surface buildings which revert to the lessor.

If, however, other land can be obtained for the purpose near at hand, it is of course usually preferred. This land may belong to the same or another owner, who grants a building lease for a much longer term of years than the colliery lease, from which it is quite distinct, and thus the cottages remain the property of the lessees subject to the payment of ground rent after the pit buildings have reverted to the royalty owners. In some districts the workmen's houses are erected upon land bought for the purpose by the lessees. Occasionally cottages are built by the lessors, the lessees paying an annual rent for them for a term of years concurrent with the mining lease. Excepting in Northumberland and Durham, it is customary for the workmen to pay rent for the houses they occupy.

In Scotland it is usual, though by no means general, for the lessees to have power to remove all buildings at the end of the term unless the owner chooses to take them at a valuation, and in some rare cases this is so in England; the lessees, however, in that case are not allowed to get the clay, stone and sand for the buildings free of charge, and very possibly the landlord when negotiating the lease fixes a higher rent to meet this arrangement.

In cases where the surface and mines ownership are severed, the buildings would not revert to the lessor at the expiration of the term, unless some special arrangement were made to that effect.

Where a property is leased on which the shafts are already sunk and all the machinery belongs to the lessor, a higher royalty is sometimes charged in accordance with their value. In districts where coal mining has been carried on for many years, perhaps centuries, as is the case in Northumberland and Durham, nearly all the original leases have expired, and the royalties paid under existing leases include the value of shafts and buildings in the lessor's possession.

There is generally a clause in the lease by which the lessee is obliged to work the coal in an approved and economical manner and in accordance with the views of a mining engineer appointed by the landlord, who inspects the mine periodically.

An arbitration clause provides for the settlement of disputes which may arise between the parties respecting the covenants or agreements contained in the lease. Each party has power to appoint an arbitrator, but if either fails to do so after a stipulated number of days the other party's arbitrator acts alone, and his award is binding. If each party appoints an arbitrator, then the two arbitrators before proceeding to the matter in dispute appoint a third. If the third arbitrator is not appointed within a stated number of days by the others, he is appointed by a Judge of any of the Superior Courts of Law or Equity, and the award of the sole arbitrator or of the two arbitrators to be first appointed without the co-operation or interference of the third or of any two of the three arbitrators is final and binding on all the parties. For the thorough investigation of the matter in dispute, witnesses are examined upon oath by the arbitrators, and documents in their custody bearing upon the subject may be demanded.

In Yorkshire, and also in some other districts, a large farm is sometimes taken in connection with the colliery leased, and where the land is good this may be of benefit; but in districts where the land is poor and it is thrust on the lessees to cultivate it may be a considerable burden. The taking of the farm should be optional, and distinct from the mining lease, although running concurrently with it.

If toward the termination of a lease the coal is not all worked out, the lessee, if desirous of continuing to work the colliery, approaches the lessor for a renewal of the lease. The lessor, although not obliged to renew, generally meets him in a reasonable spirit, so that the lease is renewed on terms satisfactory to both. No doubt hardships have been inflicted on lessees under such circumstances, but these cases are by no means common. When conducting the negotiations overpaid certain or dead-rents are sometimes taken into consideration, and carried forward into the new lease, at others they are absolutely forfeited. It would not be usual to carry shorts forward in case minimum rents had been charged in the expired lease. On the renewal of a lease, a rent is sometimes charged for the shafts and surface works which have now become the property of the lessor, so that the lessee pays for improvements which he has himself carried out during his previous lease.

A lessee may have property of his own on which he sinks his shafts and erects all his machinery, although most of his coal may be obtained from the properties of other owners. Where this is so, it may assist him in obtaining better terms for the renewal of his lease, than he otherwise would get.

The life of a colliery is sometimes prolonged by a lessee not renewing his lease, but by his leasing coal seams below that exhausted, and within the same property, so that the same shafts and plant are used for raising the coal.

In times of extreme trade depression, appeals have been made by lessees to the royalty owners for a reduction of rent, which in some cases have been granted. In some districts there has been difficulty in obtaining relief owing to the fact of estates being vested in trustees whose legal obligations deter them from giving relief in cases where the lessor acting for himself would probably have acceded to the request.

Difficulties in working the coal may arise from joint-ownership of the minerals. For instance, a lease may be held of the coal under common land which has been enclosed and belongs jointly to two or three owners as lord of the manor in well-known proportions which are not partitioned off. In case mining difficulties should prevent the coal from being worked out during the term of lease, and render its renewal desirable, the joint-owners may not be unanimous in their opinion as to carrying the shorts forward or on other items submitted for their consideration. Again, they may take different views respecting concessions which are asked for during times of great trade depression.

In West Yorkshire the terms of leases vary from 21 to 40 years. The rent is paid at a price ranging from £50 to £300 per acre per annum, or at a fixed sum per foot thick and varying from £20 to £40 per acre per annum, a fixed minimum rent being stipulated for, and no allowance being made for free coal for colliery use. There is usually power within the period of the lease to make up short workings. An acreage rent is payable to the lessor on whose land the pits are sunk for coal worked from adjoining properties and raised at the pits referred to. This shaft-rent includes underground wayleave which is not separated from it. Surface wayleaves are paid generally by the acre, but sometimes by the ton.

In South Yorkshire the terms of leases vary from 30 to 60 years. The rent is paid on the acreage worked, according to quality and thickness. The rent for the Barnsley Bed varies from £120 to £375 per acre, for the Silkstone seam it to the conditions of the lease. There is usually no wayleave on the surface where one underground, this one being considered sufficient. Unless the lessee breaks the conditions of the lease, or fails to pay his rent within a specified time, the lessor has no power to anticipate the natural termination of the lease, but the lessee can abandon his lease at any time on proving that the minerals are unworkable to profit, and there is a clause in the lease which enables him to abandon it on giving proper notice at the third, fifth, or other year without assigning a cause.

In the South Wales coalfield, leases are usually granted for a term of 60 years, some of the earlier ones having been granted for 99 years. In the old leases there is power to surrender upon the exhaustion or unprofitableness of the coal. In modern leases, the lessees have power to surrender on giving twelve months' notice at the end of certain fixed periods such as the third, sixth, ninth year, and so on throughout the term of the lease. There is power of assignment subject to the lessor's assent, which is not to be unreasonably withheld. The dead-rent is fixed at from £1 to £2 per acre and up to £5 in exceptional instances.

The royalty rent is paid on the tonnage, the ton consisting either of 21 cwts. of 120 lbs. = 2,520 lbs., or the imperial ton of 2,240 lbs. The average royalty paid on the steam coals is about 8*d.* per ton for large, which forms about 85 per cent. of the yield, and 4*d.* per ton for small, the remaining 15 per cent. The royalty on house coal is higher than on the steam coal, being as much as 1*s.* 6*d.* per ton in exceptional cases for all coal obtained, large and small mixed. Where a sliding scale has been adopted it is usually from $\frac{1}{10}$th to $\frac{1}{12}$th of the selling price of the coal at the pit's mouth, a minimum royalty of 6*d.* per ton being usually reserved.

In many of the early leases granted when the coalfield was undeveloped, a light dead-rent was provided for and no royalty. A portion of these properties has since been sublet on terms which pay the dead-rent, and the remainder is in the hands of lessees, practically royalty rent free.

In some cases short workings may be made up throughout the whole term of the lease, in some they may be made up for any year or years within three to five years of any year in which the certain rent has exceeded the amount of royalties, in others a provision in the lease divides the term into fixed periods of three to five years, within which period only the recovery of over-paid rents is possible.

No shaft rents are charged, but surface and underground wayleaves are, the former usually being light owing to the proximity of a railway to the shafts. The latter, including waterleave, airleave, and shaftage, are charged only on coal and ironstone, and not upon rubbish or materials.

Coal for colliery consumption is free from royalty, sometimes the exact quantity used being allowed, in others the practice is to limit the quantity to from 5 to 10 per cent. of the output.

The whole of the colliery consumption is apportioned to the several properties under lease, according to the quantities worked.

In the Somersetshire coalfield, leases are granted for terms varying from 30 to 40 years. The usual fixed rental is from £1 to £2 10*s.* 0*d.* per acre. In some instances the royalties are charged by the ton, being from 5*d.* to 9*d.* per ton, the average being about 6*d.* In some cases the royalties are paid on the selling price of the coal at the pit's mouth, varying from $\frac{1}{10}$th to $\frac{1}{30}$th and averaging about $\frac{1}{16}$th. Short workings may be made up in some instances within periods of 3 years, in others of 6 years, and in some cases every year throughout the whole period of the term. The lease may be surrendered on the exhaustion of the coal, or when it becomes unworkable to profit, or by notices varying from one to two years.

jury of free-miners, and documents over 200 years old are now in existence, containing a record of their deliberations. The powers of the free-miners' jury were insufficient to prevent confusion and disputes arising from trespasses on one another's coal. Consequently in 1838 a Commission was appointed, by whom three separate awards were made, one for coal, one for ironstone, and a third for quarries. The award was signed in 1841, and its provisions have been in force ever since.

By an old custom "every free-miner, duly qualified, claims the right to demand of the King's gaveller a gale—that is, a spot of ground chosen by himself for sinking a mine—and then, provided it does not interfere with the working of any other mine, the gaveller considers himself obliged to go, receiving a fee of 5*s*.. and inserts the name of the free-miner in the gale book."

This custom still continues in force, subject to modifications introduced by Act 1 & 2 Vict. c. 43, entitled "An Act for regulating the opening and working of mines and quarries in the Forest of Dean and Hundred of St. Briavels, in the county of Gloucester."

The free-miner lodges his application for a gale on a printed form at the gaveller's office. If the gale applied for is not already granted, or in charge,—that is, does not belong to any party other than the Crown—the application is entered into the book kept for the purpose at the gaveller's office; if the gale should be in charge or owned, the application is not entered but filed. If the free-miner thinks his application is the first one on the book for the gale he applied for, he sends its number to the gaveller, begging that it may be granted to him. If he proves to be the first applicant and no objection arises, instructions are sent from the Office of Woods and Forests to the gaveller's office to prepare the approbation paper, which contains the terms upon which the gale will be granted. These terms, in the case of coal mines, allow two years in which to open the mine, during which no royalty and no dead-rent are payable, and afterwards no royalty is payable on coal consumed by colliery engines or for other necessary colliery purposes. In the case of iron mines, a period of four years is allowed for opening the mine, during which no royalty or dead-rent is payable.

The terms vary, but usually include payment of a certain annual dead-rent of about £20 to the Crown, for which 1,600 tons may be worked free of any other rent, being at the rate of 3*d*. per ton; and for any quantity beyond 1,600 tons or "overs" 3*d*. a ton. It is provided also that the gale shall be opened according to the rules and regulations then in force or be forfeited. The approbation paper, together with the draft grant, is returned to the Office of Woods and Forests for approval. When approved, the grant is advertised for two consecutive weeks in three local papers.

After the second week's advertisements, a clear week must elapse after which the gale is granted to the grantee, galee, or free-miner, a fee of 5*s*. being payable for the parchment. Thus, from the time of application to the receipt of the grant, five or six weeks must elapse. In the case of a quarry free-miner, nearly the same routine is gone through, excepting that he is not entitled to apply for coal or iron gales.

No free-miner can have more than three gales granted to him. If one or more allotted be forfeited, the number may be made up again to three, if the miner is otherwise entitled to it.

A gale of coal and a gale of iron are not granted to the same man, but are kept distinct from each other. Also the upper series of coal seams are kept distinct from the lower, separate gales being granted for each.

The area of the gale is at the discretion of the gaveller and deputy-gaveller, and depends upon the depth at which the minerals lie, the probable capital required in sinking and for plant, &c. The intention is, that the smallest gale shall be large enough to justify the expenditure of capital for the purposes named. One of the deepest gales granted is about 2,000 acres in extent, whilst some of the shallow ones are only about 100 acres.

As to royalties, the maximum is the right of the Crown to put in a fifth man to work the coal for the Crown's profit after the mine has been opened; that is, assuming the gale to be a partnership property, the Crown would have the right to put in one partner to four, but would bear no part of the expenditure incurred in winning the mines. There is then this peculiarity in the partnership that the Crown pays no share of winning, but is entitled to receive one-fifth of the profits after the minerals are won.

The value of this fifth share is reduced to cash by the gaveller or his deputy, and if the dead-rent and royalty are fixed by them at what the free-miner thinks excessive figures, he can go to arbitration with the Crown, and both are bound to accept the decision of the arbitrator.

The royalty is only fixed for 21 years for each gale from any assessment, and at the expiration of that time it is open to the gaveller on the one hand, or the free-miner, or the person who has succeeded to his interest on the other, to object or express a desire to alter the rent; if the interested parties cannot agree, the matter is referred to arbitration.

In an arbitration case of an unopened mine, it would devolve on the Crown to prove to the arbitrator what, in its judgment, it will cost to sink the pits, to erect the machinery, &c.; what afterwards will be the probable quantity worked, and the probable profit realised in order to ascertain the probable value of the Crown's fifth proportion.

The licence once granted to the free-miner, unless forfeited, is granted in perpetuity, and is in the nature of a freehold; it is transmissible by will or by sale to one who is not a free-miner.

A capitalist may therefore buy a gale or as many of them as he likes. A free-miner, who seldom has much capital, may be able to work a crop gale successfully, but to win the coal in the deep gales requires a considerable outlay, and if the coal is 500 or 600 yards from the surface, the miner has to part with most of his interest in order to induce any capitalist to find the money for sinking the pits. In many instances, the free-miners merely sell the grant they have obtained to some speculator, who retains it in the hope of finding some one else to whom he may sell his interest at an enhanced price.

The following advertisement appeared in the *Dean Forest Mercury* of Feb. 5th, 1892, and explains the way in which a gale becomes subject to forfeiture:—

"V. R.—DEAN FOREST MINES.—*Notice to Free-miners.*—Whereas, the Persons entitled to or in possession of the Gale of Coal called NAG'S HEAD COLLIERY have desisted from working the same for a space exceeding Five Years at one time after the vein of Coal has been gained, contrary to the rules and regulations contained in the Second Schedule to the Dean Forest Mining Commissioners' Report of Coal Mines, dated the eighth day of March, one thousand eight hundred and forty one, whereby the said Gale has become liable to Forfeiture, I HEREBY GIVE NOTICE, that unless the working of the said Gale is *bonâ fide* resumed on or before WEDNESDAY, the tenth day of FEBRUARY, 1892, it is my intention, soon after Five o'clock on that day, to declare the Gale FORFEITED to Her Majesty.

"I FURTHER GIVE NOTICE, that in the event of the Forfeiture of the said Gale, applications for a grant of a Gale of a Licence to get the Coal in the said Gale will be received at the Gaveller's Office, at Coleford, at and after Ten o'clock a.m., on THURSDAY, the Eleventh day of FEBRUARY next.

"GEO. CULLEY, Gaveller of Dean Forest.

"Office of Woods, &c., 22nd January, 1892."

The deputy-gaveller's surveyor makes all the surveys, and keeps up all the plans of every one of the collieries and iron mines in the Forest of Dean.

In two districts of Derbyshire, very curious ancient mining customs have existed from time immemorial, and are now confirmed by Act of Parliament. The districts are Wirksworth and High Peak, in both of which lead is mined. The lead in them belongs to the Crown through the Duchy of Lancaster.

Some of these mines are believed to have been worked since the time of the Romans, whose inscriptions have been found on pigs of lead in several places at Matlock and Cromford.

In the wapentake or hundred of Wirksworth, by an Act of Parliament passed in 1852, an equal right is given to every one, strangers and inhabitants alike, to search for lead ore upon nearly all lands, whether cultivated or moorland, without the permission of the owner or tenant, and without paying compensation for damages. The same Act contains clauses enabling the miners to obtain grants of mines on easy terms from the barmaster, and subject to the tenure of workmanship they maintain a complete title to them. The appointment of barmaster is made direct from the Crown. His duties are to superintend the measuring of the ore (in measures peculiar to the neighbourhood), to choose the grand jurymen and small jurymen, to summon them when required, and to perform other business, such as setting out, giving away, and regulating the working of the mines.

The Wirksworth district is leased by the Duchy of Lancaster, but it is merely on nominal terms, the lessee receiving no pecuniary benefit from it at all, for the dues payable go direct to the Duchy of Lancaster. The dues are called "lot" and "cope." In the Statute the lot is defined as consisting of a thirteenth part, but as a matter of fact a twenty-fifth only has been taken for a long period. The "cope" payable is 6*d*. per load. These dues are received from the miners upon the gross produce when it is washed, cleaned, dressed, made merchantable and fit to sell for smelting. Under the Act fees are payable by the miner to the barmaster.

If on a search being made ore is found, the searcher applies to the barmaster, who can then set out a mine. The proportion of surface set out by the barmaster is in his discretion, and that of the two grand jurymen he selects for the purpose. The grand jurymen are practical miners, who consider each case on its merits, and set out the area they think reasonable for the commencement of operations. A large area would not be set out in advance, but sufficient to meet the probable requirements for two or three years, after which, if the necessity arise, a further allotment may be marked out. The miner has a right to the water or to the highway, and the surface allotted gives him sufficient room for buddling the ore and making all other surface arrangements. This has reference to the surface only. There is no restriction underground within the wapentake, and the miner can follow the lode as far as he likes. Besides setting out the mine, the grand jurymen inspect it in case of dispute, their practical knowledge of the veins worked being valuable. In case of a dispute on a point of law, of title, of trespass, &c., the small jury, consisting of mine owners who are a more highly educated class, act as arbitrators. A steward is appointed as judge, and the jurymen give their opinion on certain points laid before them. The judge decides and gives sentence in accordance with the opinion of the jury. The two juries and the court are recognised by law, and are established by Act of Parliament. The law for the most part merely confirmed ancient customs, it being thought that a special Court so established would be more suitable to deal with mining customs than any ordinary one. The Act does not enforce the keeping of plans of the workings, so that, unfortunately, there is no record of old workings, consequently much money is wasted in driving long exploring places.

If within a reasonable time after allotment, a person does not work his mine, through another's concession, or on the surface over someone else's land, which power is similar to that exercised in the case of the occupation of the land. If an amicable arrangement cannot be made with the neighbouring mine owners, or surface owners, the concessionaire can ask for a decree of public utility. If the public utility of the passage for his mineral through another man's mine is greater than that of the mine through which it would pass, the right of passage is obtained, valuers are again appointed, and compensation given.

The workings cannot legally be carried within a distance of 40 metres of a building, but most of the concessions are on common land.

A miner is obliged by law to allow another miner to drain or ventilate through his mine, the principle of the law being to encourage the development of the mining industry, by giving every facility for the working of the mines when a concession has once been given.

In the case of injury to the surface through subsidence, the concessionaire must buy the land or give compensation. If a friendly settlement cannot be arrived at, he has power to purchase the land, by a certain process of the law.

As a concessionaire can so readily transfer his property to others, a large number of the mines are not worked by the concessionaires themselves. When leases are granted the terms are usually from 10 to 15 years and provision is made for the payment of an annual royalty of so much per ton, an annual minimum dead-rent, power to surrender the mine on its becoming exhausted; the exhaustion to be proved by a technical committee composed of mining engineers of the province who have to declare that the mineral remaining is not workable at a profit. Some leases have no terms granted for making up shorts, but usually from one to five years are given. Any form the lease may take is purely a matter of bargaining between the lessor and lessee, and the Government has no part whatever in such negotiations.

The State levies a special tax of two per cent. on the value of iron ore at the quarry or mine mouth, which is paid by the lessee, and is not recoverable from the concessionaire.

In France all the minerals except a small quantity reached from the surface by opencast, or which are of an alluvial character, belong to or are at the disposition of the State. Only the surface owner may search for them on his own land unless another searches with the owner's consent or with authorization from the Government. The search is generally limited to two years, and is prohibited within a certain distance of buildings and enclosed spaces. A surface owner must give notice to the prefect of the department before commencing open workings. In the case of a mine, however, neither he, nor any other, can work without first obtaining a concession, which is given to the discoverer or another applicant at the discretion of the Government. Geological and other considerations influence the Government in deciding the limits of the concession, which may be bounded by imaginary lines crossing several surface properties. Concessions vary in area from 8 to 17,442 hectares. A concession is indivisible and is granted in perpetuity for the mineral specified. It may be forfeited under certain circumstances. Two or more concessions may be amalgamated by consent of the State. Land outside the concession is obtained for a railway, &c., on the plea of public utility; within by paying compensation. No underground wayleave questions arise. The rents paid by the concessionaire are a fixed and proportional royalty to the surface owner, which are heavy in the St. Etienne coal-field, and a fixed and proportional royalty to the State. The State fixed royalty is ·10 franc per hectare, payable annually whether the concession is worked or not, and the proportional royalty, including an extra tax for special purposes, is 5·5 per cent. on what is practically the profits of the mine.

In Belgium and Germany the mineral law is for the most part the same as that of France, but in Belgium the owner or combined owners of the surface, has a preferential right, subject to satisfying the claims of the discoverer, to a concession. In Germany the concession is granted to the discoverer of the minerals. Here the royalties paid to the Government are 2 per cent. on the annual profits obtained from the coal sold.

In the United States of America a broad distinction exists between public lands which belong to the Republic and are under the care of the Central Government, and private lands held by individual owners. If this distinction is not kept clearly in view, confusion and misunderstanding will arise. Lands which were private property when the Federal Government acquired its general rights are subject not to that Government, but to the laws of the State in which they are situated.

The United States Government possesses all public lands, some being in the States and some in Territories which have not yet been formed into States. Concessions have been made to the older States, by liberal grants of land sometimes comprising all the public lands within their area. But this course has not been adopted with the newer and less thickly populated States, where the Federal Government retains possession of most of the public lands.

No sovereignty is claimed by the central Government in the minerals. Before the Declaration of Independence (July 4, 1776) the thirteen original States were mostly colonies of Great Britain, and had received by royal grant, in one form or another, a delegated sovereignty in the minerals. The rights conceded by the English Crown in the original grants to the colonies were subject to certain royalties payable to the Crown. The colonies were virtually sub-lessees under the sovereignty grant. When their independence was declared and recognised by treaty Sept. 13, 1783, the colonies succeeded to all the rights of sovereignty which Great Britain previously enjoyed. Any rights not afterwards ceded by the original States to the Federal Government were reserved to the States, and as the right of sovereignty in the minerals was not so ceded, it has been claimed since that time by the original States, and although the laws authorising it are mostly dead letters the right has to some extent been exercised. Great acquisitions of territory have taken place since 1783. Large portions of this territory, in which the original thirteen States had conflicting claims, were presented by them absolutely to the central Government, including of course all sovereignty in the minerals. These lands were at the time occupied by Indians and their forests infested with wild beasts. After incorporation with the United States, Congress authorised the Secretary of the Treasury to lease lead and copper mines, and for forty years the experiment was tried, the copper mines being situated in what is now the State of Michigan. These experiments proved unsuccessful, and in the year 1847 an Act was passed by Congress, abandoning this practice and authorizing the sale of mineral land, the surface and all beneath being sold together.

Further extensions of territory followed at the end of the war with Mexico by purchase under the treaty then made. California, Arizona, and New Mexico were added. The Mexican law was the old Spanish law under which the Government never alienated the ownership of any of the precious metals. Any grants made previously by Mexico to private persons were simply for agricultural purposes. These grants were of large extent and defined by natural boundaries. The discovery of gold in California followed closely on the Mexican war, and the "gold fever" spread rapidly, bringing a large influx of population from all parts of the world. Previously mining had not been carried on in California, so that the Mexican mining laws had obtained no footing there, and the comparatively insignificant original population of California had little influence in framing the laws and customs. A great deal of trouble followed through the grants having been defined by natural boundaries and not actual survey; this gave rise to fraud, as the natural boundaries could be so regarded as to exclude or include certain gold lands and gulches. The Government had the grants tested, the boundaries properly fixed by survey, under the authority of Congress, after which regular patents were granted for those areas constituting them private property. These patents were deeds of fee simple, and although they contained no clause as to reservations, it became a question whether they were subject to the reservation contained in the original grant from Mexico. In cases which were brought before the High Court of California, and afterwards carried to the Supreme Court of the United States, it was decided that every grant of land from the United States Government includes all that is on and under the soil, unless otherwise specified.

Since the unsuccessful experiment of leasing lead and copper mines in the Mississippi valley, the Federal Government has simply fixed the terms of sale for its mineral lands. and legalised certain peculiar methods of mineral searches and working on public lands without purchase. This legislation arose from the peculiar circumstances following the discovery of gold and silver in 1847 in California. The country was over-run by large numbers of adventurers, and was without any court, or resident Government officers, and without means of communication with the central authority by railroad or telegraph. Mass meetings of the miners were held and laws agreed upon and in this way order was maintained. This, however, did not alter the fact that they were all trespassers on the public lands of the United States Government, a state of things in which the Government itself acquiesced, so that its rights were in abeyance. This continued from 1847 to 1866. In 1847 possessory titles in the public domain of the United States sprang into existence. A statute was then passed embodying the peculiar principle that so long as the Government rights remained in abeyance, all subordinate rights should be dealt with as if there were no Government right. Any mining operations on public land were regulated by local custom or law, and until 1866 it was held in cases of disputed trespass both by Local and Superior Courts, that until the United States superior title was asserted the occupier who conformed to local law had all the remedies of a fee-simple owner.

The Federal Mining Act was passed in 1866, legalising all operations on public lands according to local customs, and also providing for the sale of the mines with a patent.

The advantage of purchase from the Government over a possessory title is obvious, and is accentuated in the case of mines becoming very valuable or requiring large capital for development. In this case a capitalist might well hesitate to invest his money in land held only on a possessory title, because this was entirely dependent on local law which might be passed one day and repealed the next: whereas a Government title would afford absolute security.

Other circumstances tended to the same conclusion. Occasionally the raids of Indians would necessitate a stampede of the miners, who, in the hope of a safe return, passed a stay-law, which prevented the appropriation of their property by others. Under this law districts have been locked up for a considerable time, and when re-opened conflicting claims have arisen, owing to the fact that some owners have not returned with the rest, and their mines have been claimed by others.

In 1872 an Act was passed by which it became still more advantageous to take title by patent. That Act provides for a permanent and accessible official record of the boundaries of properties, an immense advantage to the owners and others. The same Act makes the sale of mineral lands as simple as in the case of agricultural land, with one important exception. The grant of a mineral location which is made upon the edge of a lode or vein differs from the absolute title of the fee simple, inasmuch as the grantee has an extra lateral right. The two end lines of his claim must be parallel, and he has no right to follow the vein beyond vertical planes drawn through those two lines, but with regard to the lateral boundaries, the grantee has a right to follow outside these to any depth whatever vein may be outcropping within that surface. He is unable, however, to trespass upon his neighbour's surface, and is liable in damages if he injures it. Every grantee is of course subject to the exercise of the same lateral rights on the part of his neighbour. These lateral rights are referred to as lode claims.

A placer mining location or a tract located for alluvial mining, and granted by the Government, has its boundaries defined on all sides by vertical lines drawn through the boundaries of the surface. The land is sold as mineral land out-and-out in the case of placer claims, or with the peculiarity in the class known as lode claims of having the lateral right of the grantee or patentee added to his fee simple, and the lateral right of his possible neighbour subtracted therefrom.

Coal lands on the public domain are sold by the Government by the acre, surface and mineral going together; the price if situated within 15 miles of a railway is 20 dollars per acre, if beyond that distance 10 dollars per acre. The Government will sell the land on the completion of its surveys, but a man may enter upon possession beforehand, with preferential right of purchase. It claims no seigniorage or royalty.

The Act of 1872 to a great extent extinguished the vagaries of local customs by laying down regulations to be observed by local Governments. One clause insists on the performance of a certain amount of work annually on every mine held by a possessory title. The discoverer of a vein on the public land locates it, and if no one else has a prior claim he marks out the area of the land he wishes to hold. By doing a certain amount of work every year upon that claim, he for an indefinite period may retain possession of it without purchasing. The amount of work must not be less than that prescribed by the Government; more may be required if the local customs happen to be onerous.

The Government having once sold mineral land does not afterwards hamper its working. No law regulates the method of working or the payment of taxes, or interferes with the freedom of the owner of the property to use his surface or underground right as he may think fit. He is in the same position as the owner of land under English law. One very important provision, however, has been made by the Government with respect to wayleaves on public lands. First, there is absolute freedom of way across the public land for mining ditches in which water is conveyed, and also for mining roads, tunnels, or anything necessary to mining operations. Secondly, there is a right-of-way easement, which does not include the freehold. In this case compensation for the land is determined as in other cases where private property is acquired by the State for public use. If A. is carrying on mining operations and can only get out by crossing the claim of an adjoining neighbour B., he may do so whether B. is willing or not. If, however, the crossing is done so as to injure B.'s mining operations, a jury assesses the damages, or the courts take charge of the case. The Government makes mining claims and lands subject to all rights of way already existing over the public land, and also subject to rights of way, easement, or drainage enacted by the local Legislature; these are invariable reservations from the grant. If an easement is obtained by legal process compensation will be accorded in proportion to the damage sustained and not in proportion to the benefit accruing to the party seeking the right of way.

All questions affecting the safety of miners and other local regulations are left to the individual States.

On a mine being developed and proving valuable the high price which it will command in the market is equivalent to a capitalised royalty. Of this Government of course receives no portion There are cases on the public lands where a regular tonnage or per centage royalty is paid. This may arise from the first adventurers having lost money; the mine may have proved poor, even if rich for a time. The owners not being able or willing to find sufficient funds to carry it on, let it on lease at a royalty rent.

In the Eastern States there are no laws to govern wayleaves or easements between neighbouring owners; questions relating to these are settled by common law, on the basis of freedom of contract. In these States two classes of iron and coal mines exist, viz., those worked by proprietors who pay no royalty, and those worked upon a system of mining leases under royalty, the grantor being the owner of both the surface and the minerals, who does not wish to be burdened with the management both of the land and the coal working. The system of leases is very similar to the English. The leases in the case of iron mines are for terms of 20 years or upwards, and in coal mines usually from 25 to 50 years and occasionally 99. They are subject to dead or certain rent with power to make up short workings for any year throughout the term. The powers of renewal at the expiration of the term, wayleave rights, and right of sale or of re-letting are matters of arrangement between the parties. The royalties are sometimes paid according to a sliding scale which is regulated by the selling price of the coal.

It is not customary for the proprietors to provide houses for their workmen rent free, though a lower rent is usual to their workmen than would be the case to others.

The coal mines of the country are divided into three classes, viz., the anthracite, the bituminous coking coal, and a third class consisting of all other coals. The anthracite and the bituminous coals are subject to royalties which vary in proportion to the value of the coal yielded, and the facilities for working it. The lignites are found for the most part in the Rocky Mountains, and are not subject to royalty rents. They are worked by adventurers who take up the land, which is sold cheaply by the Government.

In some States under the general railway laws a right of way across intervening lands may be obtained by proceedings in condemnation if amicable arrangements fail. In this case damages are assessed on the basis of injury done, and not of benefit accruing to the promoters. The roads authorised by condemnation are always assumed to be public carriers; they must carry the freights brought to them. Private wayleaves are obtained by negotiation and in almost all cases by the payment of a lump sum.

The Government reserves all public lands officially reported to contain minerals. Other land is sold to agricultural applicants at from \$1¼ to \$2½ per acre. Mineral lands are sold at \$2½ to \$5. Coal lands are sold at \$20 per acre if within 15 miles of a railway, and \$10 per acre otherwise. No agricultural claim can be made on mineral land, but if a tract should be purchased as agricultural land and mineral be afterwards unexpectedly found on it, the agricultural claimant is entitled to the mineral interest. He becomes absolute owner and may work the mineral himself, lease it to others, or keep it locked up. No agricultural land is sold by the United States Government as a rule, until the surveys are completed, but in the case of mineral land that survey is not made until an application is received from a purchaser, who is charged by the Government officials with the cost of the official survey. This forms the principal expense in acquiring the land, the price per acre being so small. In addition to this the Government requires proof of an expenditure of 500 dollars under the possessory title before it will complete the sale, as an indication of the discovery of a lode. The maximum size of a mining claim is about 20 acres, 2 roods, and 26 perches, being 1,500 feet by 600. Placer locations are more extensive, running up to 120 acres. A small capital, therefore, is sufficient to purchase a mining claim.

In the colony of Newfoundland, the minerals belong to the Crown. Any person may search without licence, and for a fee of \$50 may obtain a temporary lease, which gives the exclusive right of further search for a year over one square mile. If the search proves successful within that year a further lease is granted for a period of 5 years, the only condition being that a specified sum per annum shall be expended in *boná fide* prospecting or operating the mine. The aggregate amount is $6,000, being $800 during each of the first four years, and $2,800 in the fifth year. Failing this, the lease is forfeited and the land reverts to the Crown. These conditions being fulfilled, a grant is made in fee of the minerals (excepting gold which is reserved to the Crown) contained within the one square mile, and also a freehold grant of 50 acres within the same area, the Crown reserving all surface rights over the remainder. When the property is granted in fee to an individual, it remains his whether he works the minerals or not, and he has the same right as a freeholder in this country. The Government reservation of the greater portion of the surface does not take effect until the purchaser has had ample time to complete his search and verify his discoveries. The greater portion of the island is not yet colonized, and all the minerals known are in unoccupied land.

When a grant is made the minerals vest absolutely in the grantee, who has the power to sell or let. Copper is the chief mineral worked, though lead and silver are also mined. Coal has been found, but is not worked owing to difficulties in that locality respecting the French treaty rights, which make the title doubtful.

No local or imperial taxes are levied on the mines, the principle of the legislation being to encourage the investment of capital in the mineral wealth of the colony.

In British Columbia, which now includes Vancouver's Island, there is a royalty charged on Crown lands of five cents (about $2\frac{1}{2}d.$) per ton upon coal raised. The coal belongs to the surface owner, subject to this royalty. Minerals other than coal are searched for by free miners who take up claims. The largest claim measures 1,500 feet by 600. Within his claim-limits all veins and lodes belong to the miner holding the licence. He may obtain a Crown grant to include all minerals within the boundaries of his concession, and have power to run drains through unoccupied mining lands subject to paying compensation.

In the colony of New South Wales, the Government possesses all unlet minerals, but a great many of those which have been worked belong to private owners. The grants made before 1861, under Orders in Council, contained various reservations, and were not of a uniform character; some contained a reservation of timber for bridges, some of roads, and some of all minerals. About 1854 the Governor issued a proclamation cancelling the reservation of minerals. A large company called the A. A. Company, having a grant of a million acres, thus became possessed of very valuable coal seams and other minerals which existed under a considerable portion, though not the whole of their grant. This property forms a part of the Newcastle coalfield, the most valuable in the colony. In a Land Act passed in 1861, two conditions were laid down for granting lands, (1) required the payment of £1 per acre and residence for a specified term of years, but conveyed no interest in the minerals; (2) the other, under the Mineral Conditional Purchase Clause, provided that on payment of £2 per acre, and expenditure of £2 per acre in working minerals other than gold, the land should be granted without reservation of the minerals. This clause did not long remain in force, as it was prejudicial to the public interest. Under its operation a number of private people obtained possession of mineral lands.

In 1884 the Crown Lands Act was passed, by which all grants of land for settlement contain a reservation of the minerals to the Crown, the interest of grantees being restricted entirely to the surface. By it the Crown also reserves all rights to give authority to work the minerals. "Minerals" according to the Act, "include coal, kerosene, shale, and any of the following metals or any ore containing them, viz., gold, silver, copper, tin, iron, antimony, cinnabar, galena, nickel, cobalt, platinum, bismuth, and manganese, and any other substance which may from time to time be declared a mineral within the meaning of this Act by proclamation of the Governor published in the Gazette."

Owners of the soil, including the minerals, are much in the position of a private owner in England, having power to sell or lease the coal. Companies have been formed to take advantage of such powers. Where the coal is leased, a tonnage royalty is paid, and a minimum annual dead-rent. The properties are usually large, and seldom give rise to questions of wayleave. The law of subsidence of the surface is the same as in England, and as some of the seams are very thick their working in some places causes injury to buildings, &c., more especially as the pits are generally shallow. The local taxes are paid by the lessees. On the coal becoming exhausted or unworkable to profit there is power to surrender the lease, the question of its being unworkable to profit being determined by arbitration. The lessor has usually power at the termination of the lease to purchase the fixed machinery and rolling stock at a valuation.

The Crown Lands Act of 1884 divides the surface and the minerals into two departments, one the Crown Lands Department, dealing with the surface settlement, the other, the Mines Department, with the mining. It provides that "The Governor shall, notwithstanding the provisions of the Mining Act of 1874, impose a royalty of not less than *6d.* per ton on coal raised from land which may be hereafter leased; and such royalty shall be in addition to or in substitution of any rent payable by such lessee under the said Act, but shall not affect or prejudice any other condition of the lease."

It is in the discretion of the Minister for Mines whether in addition to this royalty the rent of the land shall be paid or not. The object of this discretionary power to charge royalty and rent is to prevent the taking up of large tracts of land for trading purposes. If a lessee does not work the minerals the Government can insist on payment of the yearly rental. If the land is of a rough description, and the lessee undertakes to spend money for its improvement, the royalty alone is required. The rent is £1 per acre per annum for gold-bearing lands, and for other minerals, *5s.* per acre. The Crown does not work the minerals, but gives the right to private persons under certain rules and regulations. A man may by paying £1 obtain a miner's right, and for another £1, a mineral licence. Provided with these, he can enter upon any Crown lands in the colony to prospect. If successful, he can make application for a lease, stating what minerals he wishes to work. He is permitted to continue operations during the official survey, and in three, six, or nine months, he receives his lease, and as long as he complies with its conditions, retains absolute possession of the minerals. If there are two or more applications, the Mines Minister deals with them as he thinks right. In the case of a gold lease, the area is from 1 to 25 acres for reefing gold, but much smaller areas for alluvial gold. In the case of any other mineral the largest grant is for 640 acres. More than one block is seldom granted to the same person, the desire being to prevent the taking of land for speculative purposes. Any number of persons may apply collectively and have a corresponding acreage allotted them; thus six may have 3,840 acres.

The lease is for a term of 15 years, with the right of renewal for 15 years, the intention being to extend the lease indefinitely so long as its conditions are fulfilled. No compensation is allowed to the lessee for his expenditure of capital at the termination of the lease. A lessee has the right to sell his interests, subject, of course, to the payment of royalty due to the Crown.

No royalty is charged except on gold and coal; for other minerals a fixed rent is charged.

At the expiration of the term of 15 years, the royalty of *6d.* remains the same on renewal, but the rent of *5s.* per acre on coal may be increased.

A Mines Regulation Act, passed in 1874, is in force throughout the colony, which applies equally to mines held under the Crown, and to those under private individuals. Government inspectors are appointed to enforce the operation of the Act.

The Crown reserves in its leases all easement rights which it grants in case of need, to the lessees of adjoining mines. Thus a lessee may obtain shaftage, way-leave, ventilation, or watercourse privileges, if the Mines Minister is satisfied with the urgency of the need. No lessee can dispose of his interest, or mortgage it, without the approval of the Crown.

The Crown retains the power at any time to take from the lessee land which is necessary for a public purpose.

In Crown leases of coal under sea or tidal water, the lessee is compelled to carry on his working by pillar and stall method in order to leave support for the surface.

Forfeiture of the lease may result from breach of covenant, arrears of rent, or using the land for purposes other than those specified in the lease.

If an outlet is required from a mine to a port or railway station, a private bill which provides for payment of all proper claims, is passed by Parliament on the plea of public utility.

Crown lands conceded with full mineral rights, may be resumed by the Crown on certain conditions somewhat similar to those below mentioned, prevailing in South Australia and New Zealand.

In Queensland, the Government has power to mark off mining districts, within which licences are granted for working all minerals other than gold. The licences are for 21 years, at a rent of *10s.* an acre. Under the Coal Mining Law of 1886, a licence is granted for one year, allotting land up to 640 acres at *6d.* per acre, within which a search may be made for coal. This is renewable on the same terms for another year. If the licensee discovers coal, he may obtain a lease for 320 acres at a rent of *6d.* per acre, and a royalty of *3d.* per ton of coal raised during the first ten years, and then *6d.* per ton on the output for the remainder of the lease. A law of 1881 regulates labour, ventilation, shaftage, fencing, machinery, inspection, and safety-lamps.

In South Australia, leases for minerals, other than gold, are granted by the Crown over an area of 80 acres (formerly 640 acres) for a term of 99 years at *1s.* per acre, and 2½ per cent. royalty on the net profits. Licences to search are also granted preparatory to a lease as in Queensland. In the Northern Territory a different law prevails to suit local conditions. There, by a law of 1888, private lands may be resumed for mining purposes by the Government provisionally for six months; after that period the resumption ceases, unless it is previously made absolute. If absolute, the owner receives payment of the purchase money and compensation for the loss of his land, but the price paid does not include anything for the minerals. Royalties received by the Crown from the workers of those mines are paid over to those who, but for the resumption, would be exercising the mining rights, less a commission of 2½ per cent. retained by the Government.

In Victoria, there is a Mining Act of 1890. Here, as in Queensland and South Australia, a licence to search upon Crown lands for minerals other than gold is granted. For the purposes of cutting races, making dams and reservoirs, and diverting waters, licences are also given on Crown lands and on lands leased to private persons, compensation being paid. Mineral leases on Crown lands are granted for 30 years to the extent of 640 acres with a right to cut races and make ton of marketable coal. No royalty to be charged for the first two years of the term. Proper books showing the output must be kept by the concessionaire and produced for inspection. The grant is for either 20 or 30 years, and is subject to withdrawal after two years, if a specified output is not reached, or at any subsequent time if it is not maintained. The concessionaire is bound to comply with the Government's requirements as to plans and method of working. Provision may be made to prevent any claim by the concessionaire for damages resulting from any subsequent alteration in the mining laws. There is no specified power of renewal, but in the event of non-renewal the concessionaire is allowed a given time in which to sell or otherwise dispose of his plant. There is no power of subletting or transferring the concession without the consent of the Government; such consent is, however, usually given in case the transferee is a British subject having means to carry on the mine. Where the surface is waste land and in the hands of the Government, complete power is given to the concessionaire over the surface, but where it is occupied or subject to easements, the concessionaire must pay all damages resulting from his works to the surface occupier or owner. In order to convey the produce of his mine over land belonging to some one else, the concessionaire may arrange the terms of compensation amicably by private negotiation with the surface owner. Failing this, the Government may, if necessary, take action under the expropriation law which would throw the full compensation awarded under the law upon the concessionaire. Ordinary cesses or rates are levied on coal mines for local purposes; neither of the two cesses at present levied in Bengal may exceed one-half anna in the rupee, or about 3 per cent. of the net annual profits of the mine.

The Government are considering the passing of a law for the regulation and inspection of mines.

There is no fixed rule as to terms of concessions or as to the rights given to a concessionaire, so that intending applicants are at a disadvantage as compared with those in European countries, where full information is given in regard to these matters.

to one shaft by boring a hole downward from the bottom of the second shaft, and connecting it by means of a drift with the leading shaft at different points as the sinking proceeds.

The lower strata of a deep colliery are mostly dry, and it is frequently found that after the shafts are tubbed through the water-bearing strata no further pumping is required. If tubbing is not resorted to, in consequence of seams too near the water-bearing strata having to be worked, the pumping may be more advantageously met by a third or pumping shaft sunk only through the water-bearing strata, and used solely for raising water.

Shafts must be fairly large where a daily output of 1,000 tons is to be landed, and if much more than this is aimed at, and the winding is from a great depth, possibly more than two shafts may have to be sunk on the property. If the sales are assured, a large output is more profitable than a small one: an output below a certain point results in the colliery working at a loss instead of at a profit. Very large outputs from a single shaft are only possible when there are the natural advantages of thick seams which can be easily worked, and then the shaft must be of large area, and, if deep, must be well equipped for rapid winding with cages constructed to take large trams. If a mechanical ventilator be employed both the downcast and the upcast shafts may with advantage be used for winding from a different seam of coal. Either of the shafts may be used for pumping water, but if only one shaft is used to wind, the pumps should be placed in the other, so that accidents to winding or to pumping appliances may interfere as little as possible with the operations in the other shaft.

Very large takings are pierced at different points by such shafts as are thought necessary by the engineer in charge, who allots to each pair of shafts the area that can be most economically dealt with, having regard to the exhaustion of the coal within the period of the lease, the commercial value, and possibly the divisions made by large faults or by old workings.

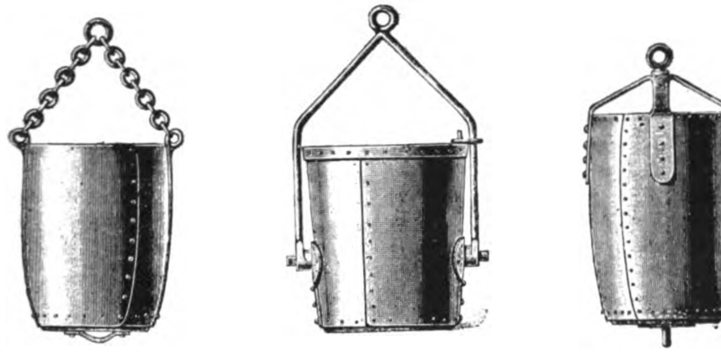
In South Wales, where favourable conditions prevail, a daily output of 1,000 tons is in many instances obtained, and in rare ones even 1,500 tons have been raised up a single shaft. Here the thickness of the seams allows of the use of large trams, having a carrying capacity of from 30 cwt. to 2 tons. Single-decked cages are for the most part used so as to facilitate the changing of the trams in them. In order to receive two trams ranged one behind the other the cage must be of great length. This of course necessitates large shafts, which are frequently from 18 to 20 feet in diameter in the clear. These large shafts also admit of the workings being well ventilated, which is a matter of great importance in the working of fiery seams, and requires to be kept in view as much as the question of output.

Where thin seams such as those at Radstock are worked, it is found impossible to raise large quantities of coal. In that case faults add to the difficulties of working these seams, and shafts rarely if ever exceed 10 feet in diameter. Such shafts under the prevailing circumstances meet all the demands made on them, whether for winding, pumping, or ventilation.

Deep shafts frequently take years to sink, but larger shafts, although more expensive, are not usually longer in sinking, because of the larger number of men and machinery that can be employed. It may safely be assumed that shafts varying from 10 to 20 feet in diameter will meet all the requirements of the different districts in this country.

In all new winnings two shafts are required by law, and their site is a matter requiring much thought. Other things being equal, that site which gives the largest amount of the field to be worked to the rise is preferable. The underground conveyance is thus rendered easier, and water in the workings will naturally gravitate to the shafts. If the seams of coal to be worked are lying in a horizontal or nearly horizontal position, there is an advantage in placing the main shaft in the centre of a property of moderate size so as to necessary for the discharge of the contents which takes place whilst the kibble remains suspended. The kibble is made of wrought-iron, and the catch prevents it from tilting during the winding.

A kibble not intended to have its contents discharged in this way, but for transport to a rubbish tip after reaching the surface, is made as shown in Fig. 19 (C), but without the valve, and sometimes with chain attachment to the sides, as in Fig. 19 (A). Care must be taken in loading the kibbles only to place the broken material to within a few inches of the top. This is a matter of supreme importance to the sinkers in the pit bottom, to whom the falling of an ounce or two of stone or other substance may mean serious injury or instant death. Tipping kibbles were formerly used for winding water, and were made with a smaller bow fixed to the sides as well as with the larger one. The contents were discharged on the surface by the banksman who released the catch and pulled the kibble over by means of the smaller bow. This kind of kibble is unsuitable for winding



A B C

Fig. 19.—A, Kibble for Winding. B, Tipping Kibble or Bowk. C, Barrel for Winding Water. with Valve at Bottom.

a large quantity of water, because it does not fill easily, the water having to flow into it over the top. It is sometimes used, however, when there is only a small amount of water in the shaft bottom. The water is then baled into the tipping kibble and wound to the surface. But baling is a tedious process, and therefore costly. Sometimes these tipping kibbles are made with an ordinary chain link placed so as to slide over one link of the bow and long enough to reach to a short vertical pin riveted to the inside of the kibble. Over this pin the link is placed to hold the kibble firmly in place. When the link is lifted the kibble turns over and empties itself. Tipping kibbles, whatever their form, need only to be disconnected from the rope at the bottom of the shaft. The other form of kibble must be taken from the rope and replaced by another at the surface as well as at the bottom of the shaft. The *water-barrel* is much the same as the kibble, but it has a valve in the bottom, and is used for sending the water to the surface, where the banksman pulls a handle communicating with the valve which allows the water to run out; or in another form of water-kibble the valve-spindle projects below the level of the kibble, so that when the engineman lowers it on to the runner the valve is opened by the action, and the water runs out.

Fig. 19 (C) shows a water-barrel which is made of wrought-iron with a water-tight valve in the bottom moved by a central spindle extending beneath to allow the water to run out. The barrel is lowered into the water in the shaft, and if this is of sufficient depth the barrel fills through the valve, which is lifted by the pressure of the water. When the barrel is full, the valve drops into its place and

Tipping waggons are constructed in various forms to suit various purposes, such as the single end tip, the double side tip, and the all-round tip waggon; the general construction is the same, but they are so arranged that the contents may be discharged as described.

The body of the double side-tipping waggon shown in Fig. 21 is made of steel, and is carried upon four pairs of trunnions. A large tipping angle is thus obtained, and a free discharge of the material ensured. The sides and bottoms are formed with one plate, which is bent to further assist in the discharge. The trunnions are firmly riveted through the end plate of the body, which is strengthened by an additional inside and outside trunnion plate. Each end of the body is made with one plate, which is flanged by hydraulic machinery, and is riveted firmly to the sides. A strong half-round welded ring is riveted round the top, and holds it rigidly together. The body as thus constructed is very strong and durable. The under-frame is of channel steel, with steel bowed buffer ends, and steel angle stays across the underframe; the trunnion supports are of

Fig. 20.—Spring Hook.

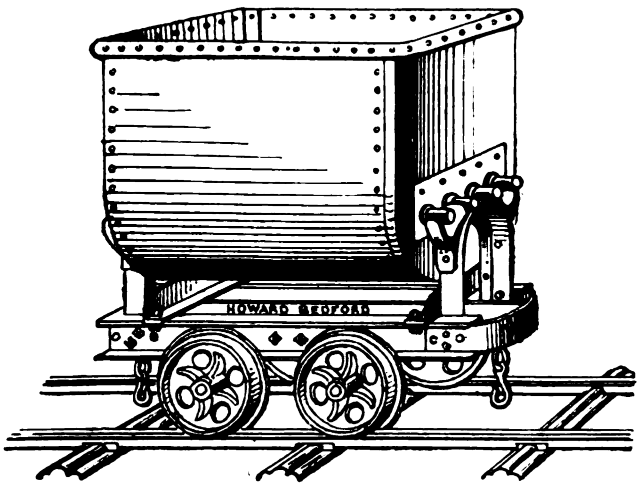


Fig. 21.—Double Side-Tipping Mining Waggon.

angle steel; the draw-bar passes throughout the length of the under-frame, forming a central stay; the axles are of steel and the wheels of chilled iron or cast steel. Waggons with round buffer ends are far superior for light railway work to those built with corner buffers, as the liability to derailment on curves is greatly lessened. This may be of no importance in tip waggons for a sinking pit, and other buffers, if preferred, can be substituted. The trunnions are placed at such a distance apart as to ensure steady running, thus dispensing with safety chains whilst still maintaining an easy tip.

The usual capacities of these waggons are 10 and 16 cubic feet on gauges of 18 inches to 24.

After the first 6 feet of sinking has been done, which will most likely be through soil or clay, *curbs* or *cribs* must be put in. These are segments of wood (see Fig. 22) cut out to the circle of the pit, and are generally 6 inches square and made of oak or elm, the former being preferable. The joints must radiate truly from the centre of the shaft, and cleats, as shown in the sketches, secured to them at the surface in order that the joints may be brought into proper contact when the curb is fixed in the pit. Sometimes a scarfed joint is made between anything but pleasant to those riding in a sinking shaft, and may possibly cause an accident through giddiness.

If the shaft be fitted with wooden or iron conductors as the sinking proceeds, these may be used to guide a slide carrier or rider. It consists of a horizontal cross-bar, the ends of which fit the guides. This is carried by inclined struts to a circular piece fitting loosely over the winding rope. A buffer-catch is secured to the rope at its lower extremity, which fits into the circular portion of the slide carrier, and raises it in its ascent.

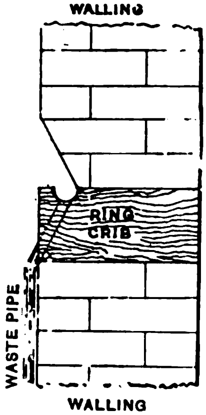


Fig. 23.—Ring Crib.

The guides at their lower ends have stoppers, so as to arrest the carrier in its descent, and hold it there whilst the buffer-catch becomes disengaged from the rider, and the kibble is free to descend without guides to the bottom of the shaft. Similarly it ascends unguided again until it reaches the carrier, which is afterwards lifted up with it.

Where wooden or iron guides are not available, wire ropes may be used as guides for the carrier. The guide ropes may be secured in the shaft to a balk stretched across the shaft; the balk also serves to arrest the further descent of the carrier.

The guide ropes are wound on the drums of steam or hand crabs on the surface, and pass over pulleys at the shaft there, so as to admit of their being easily lowered; they are long enough to reach the expected depth of the sinking.

The objection to this system is that the balk in the shaft requires frequent change. Mr. W. Galloway has patented a great improvement of this system, in which the ends of the guide ropes are secured to the cradle, beyond which the kibble descends unguided through a door in the cradle. In some instances a single guide rope has been used in deepening shafts already at work, where the circumstances were such as to make it undesirable or impossible for

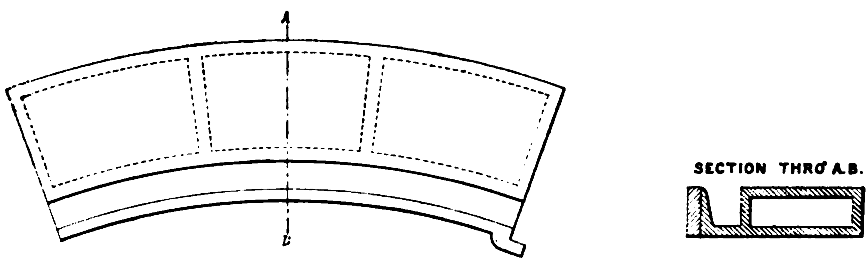
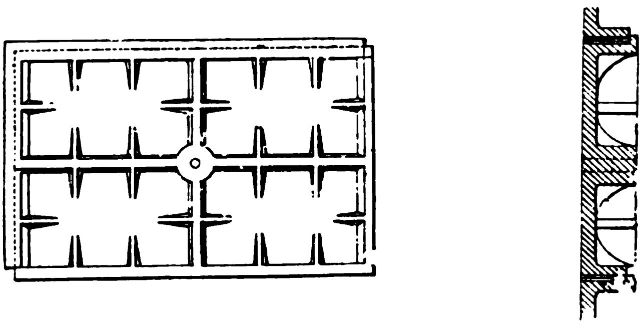


Fig. 24.—Combined Wedging and Ring Crib.

sinking and walling to proceed simultaneously while coal was being drawn from a higher level. The guide rope is wound on the drum of the capstan engine at the surface, and uncoiled to lengthen it as the sinking proceeds. During the walling it is used to raise and lower the cradle. While sinking is in progress the cradle is removed and a heavy weight attached to the lower end of the rope, which is lowered almost to the pit bottom. The rope thus forms a guide for the kibble all the way. One end of a horizontal bar is made with an eye to run freely up and down the guide rope, while the other is connected to a bar below the winding rope, so that the kibble in its ascent and descent is continually kept an equal distance from the guide rope. This very largely prevents oscillation of the kibble, which, however, cannot be run in the shaft so steadily with a single as with a double-rope guide. There are objections, too, to weights suspended in sinking shafts, as they must occasionally be raised above the sinkers working in the bottom.

A *ring-crib* (Fig. 23) is frequently used in wet pits. It consists of a crib hollowed out in the shape of a gutter, and is built into the shaft, the first two or three courses of brickwork upon it being inset or shorn back, so as to leave a portion of the crib in which the channel is cut exposed as shown in the sketch. The water which trickles down the sides of the shaft runs into this ring-crib, and from thence it is conveyed down the shaft by means of a "waste pipe," one end of which is let obliquely through a hole in the crib to the bottom of the channel, and the other is placed over the cistern, from which the pump takes its water. Sometimes a wedging crib is used as a "ring-crib" (see Fig. 24).

Frequently large quantities of water are found in the hard rocks, and if circumstances admit, these should be tubbed back with cast-iron tubbing. If,



(A) Elevation. (B) Cross Section.

Fig. 25.—Segment of Metal Tubbing.

however, the bed of rock on which we are able to place the bottom part of this tubbing is near a seam of coal, or even if it is some distance away, but in communication with it by fissures, or the rocks between are not impervious to the passage of water, there will be no advantage derived from the tubbing, because the water will find its way down to the seam, and remain as much a burden to the colliery as if the tubbing had not been put in. Assuming an impervious bed of rock to be available, and that there is a large quantity of water below the walling, the sinking would be continued from the last wedging crib at the reduced diameter of 15 feet for a few feet, so as to leave a support under the wedging crib, and then gradually enlarged to a size suitable to take the tubbing, and on approaching the impervious rock alluded to, the shaft should be again reduced to a diameter of 15 feet, and the sinking carried a few feet, say 6, into it, to serve as a sump, and allow of the water kibbles being used whilst efforts are directed to the tubbing operations. Explosives should be avoided in sinking past the point where the wedging curb will be fixed.

A bed should be carefully prepared at the commencement of the impervious rock for the wedging crib, or cribs (for often two are laid), which is somewhat similar to the walling curb, but of cast iron about 6 inches deep, and 13 inches in the bed. The pit is shorn back so as to admit of the wedging crib being placed, and also leave a small annular space round it; the bed for the reception of the wedging curb must be dressed with hacks perfectly smooth and level. The curb is then laid and securely wedged, the space behind having been filled with fir sheathing, and behind that again moss or oakum. Sometimes both the single and double cribs are provided with escape valves to release the air as it escapes from the back of the tubbing. The segments of *metal tubbing* (see Fig. 25), having first of all been tested by sounding them all over with a hammer and punch on the surface, are next to be proceeded with. These segments are cast with a smooth inner surface, and are flanged so as to fit into each other, a hole being left in the centre of each to allow the water to run out whilst building them up, and also for convenience in sending down the pit. Pitch-pine sheathing is first laid on the wedging crib, and then the segments are fitted round it, 10 or 12 forming the circle, and they are usually 2 or 3 feet high. The ground is shorn back near the crib where required as the tubbing is built up. The thickness of metal in the tubbing will depend upon the height the tubbing has to be carried, and varies from $\frac{3}{4}$ of an inch to $1\frac{1}{2}$ inches. Mr. Greenwell gives the following formula for estimating the thickness of metal tubbing, the height of the segment being 2 feet:—

Let $x =$ the required thickness in feet.
$P =$ the pressure or vertical depth in feet.
$D =$ the diameter of the pit, also in feet.

$$\text{Then } x = \cdot 03 + \frac{P \times D}{50{,}000};$$

so that if we had 60 fathoms of water-bearing strata to tub through in a 15-foot pit, we have

$$\cdot 03 + \frac{360 \times 15}{50{,}000} = \cdot 138 \text{ of a foot} = 1 \cdot 656 \text{ inch.}$$

In practice the thickness of tubbing is generally reduced every few feet upwards. Sometimes it is necessary to cover the tubbing with tar or wood lining, to help preserve it; if the shaft is afterwards to become an upcast, ventilated by a furnace, a brick lining over the tubbing will be necessary. But this would not be placed until the building of the tubbing in the shaft was quite completed. A second course of segments is now proceeded with, a sheathing of pitch pine having been laid all round on the top of the first, to allow of wedging when all is built up, and care must be taken in the building to break the joints of the tubbing. This method is continued until approaching the rock left to support the wedging crib above, a part of which must be shorn back to allow of the tubbing fitting in truly under the wedging crib. All the vertical joints in going upwards should have strips of wood laid behind them an inch thick, and about 6 inches broad, and a wedge-shaped piece driven behind, to force the segments well together at the joints, and the space behind the tubbing should be filled up with concrete. When all the tubbing has been thus placed in position, the important operation of wedging the joints is commenced, from the bottom upwards, leaving the centre holes till last. The plugging of these is upwards also, and requires skill and care if much water is coming through them.

Sometimes the tubbing, when carried above the water-bearing strata, is left open-topped, but this does not allow of such good wedging as the close-topped tubbing. Sometimes it is necessary with close-topped tubbing to put a pipe into one of the upper segments, and either allow it to remain open, and the water to run constantly down the pit, or to continue the pipe up the pit above the level of the water behind the tubbing. This allows a vent for the air from behind, which, when no provision was made for its removal, has been known to do considerable damage. The segments should have proper pieces cast on them to which to fasten the buntons, when the pit is fitted up with guides.

It has been stated that the Shireoaks pits have more tubbing than any other pits in England, viz. 170 yards. The pressure at the bottom is about 196 lbs. per square inch.

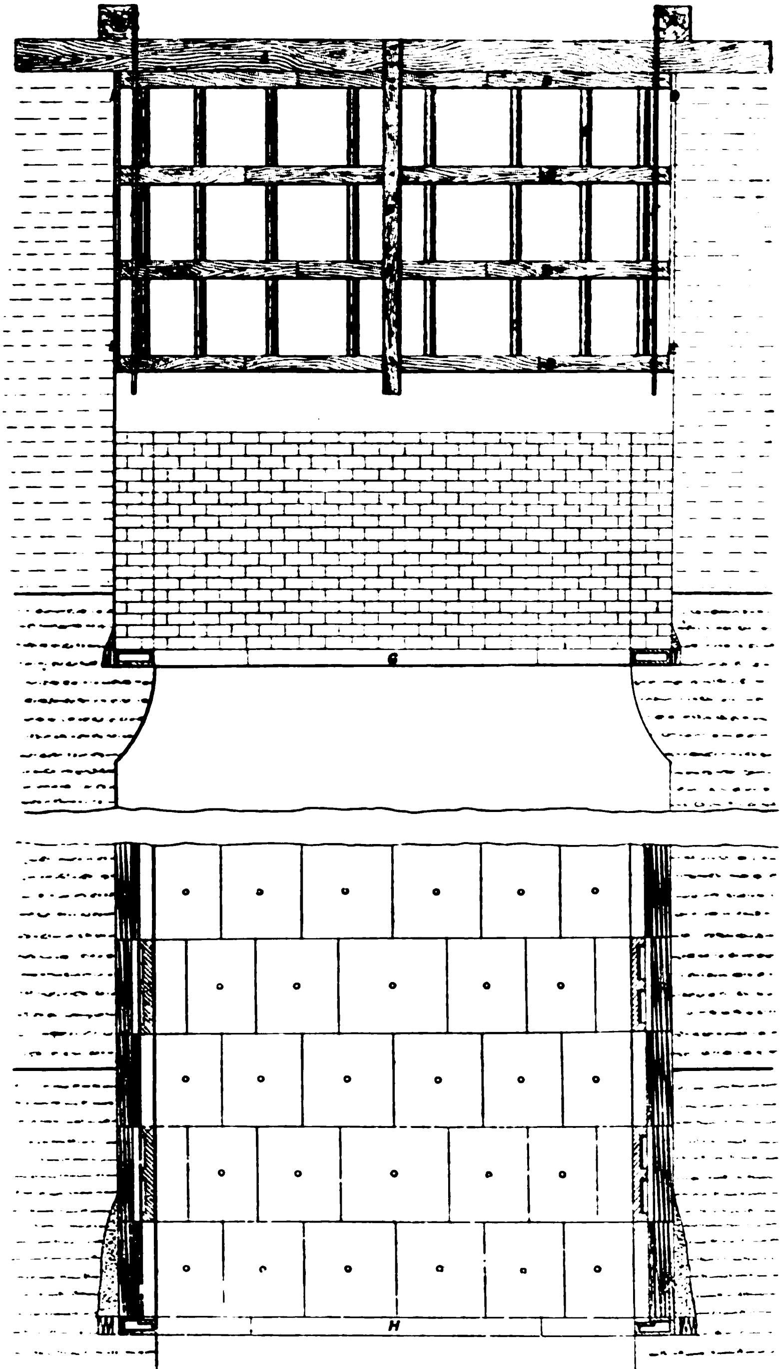


Fig. 26.—TIMBER, WALLING, AND TUBBING IN A SINKING SHAFT.

A. Balks laid across the top of the shaft. B. Timbering curbs. C. Punch props. D E. Backing deals. shown in section but otherwise omitted for the sake of clearness. F. Stringing deals. G. Walling curb, with the walling shown above it. H. Hollow cast-iron wedging curb with cast-iron tubbing resting on it.

18 inches. With 15 feet length of piles, a fresh course is required every 12 feet, so that the depth of the quicksand in feet divided by 12 and the result multiplied by $1\frac{1}{2}$, gives the reduction in size. The reduction in a quicksand of 84 feet would be $\frac{84}{12} = 7$, and $7 \times 1\frac{1}{2} = 10\frac{1}{2}$ feet; if the diameter of the pit is to be 17 feet 6 inches, to allow of a 15-foot net size, it would require to be 10 feet 6 inches + 17 feet 6 inches = 28 feet in diameter at the top of the quicksand.

It will be necessary on reaching the quicksand to put an additional crib-bed 6 inches less in diameter than the diameter of the crib last put in; it must be concentric with it, and allow of the piles being driven between the two. Care must be taken to drive them down vertically, and after getting them down a few feet, as much of the quicksand as is practicable is taken out and a crib put in. Again the piles are driven down a few feet, the quicksand removed and a crib put in. These operations are continued until the piles are fully down, and the quicksand removed to within 3 feet of the bottom of them, a crib laid to support them and another laid inside and concentric to it, 18 inches less in diameter, to allow the next course of piles to be driven in the annular space between the two. The same operation of driving, excavating and laying cribs is again gone through. When within 3 feet of the foot of these piles another curb 18 inches less in diameter is put in to allow of another course of piles, and so on till the stone head is reached, when the wedging curb is laid and the walling or tubbing run up as expeditiously as possible through the treacherous ground.

Another method of getting through quicksands, consists in sinking by means of hollow cylinders of cast iron, pressed down by heavy weights piled on the top, but sometimes there is considerable difficulty in keeping the cylinders in a vertical position, especially if large boulders are met with.

An ingenious and efficient method of sinking through quicksand is POETSCH'S FREEZING SYSTEM, whereby the quicksand is transformed into a solid mass. The quicksand in its changed and solid form is then sunk through in the ordinary manner.

A refrigerating liquid, consisting of a solution of chloride of calcium, is produced on the surface by means of proper machinery.* Through tubes this liquid is conveyed into the shaft required to be sunk through the quicksand. A zone of quicksand must be solidified round the shaft and downwards sufficiently far to form a wall or barrier all round the part to be excavated, and sufficiently thick to resist the surrounding pressure.

The thickness of this wall is determined beforehand by taking into account the depth at which the quicksand occurs, and the thickness of the quicksand itself.

Fig. 28 shows the principle of applying Poetsch's sinking process to a quicksand 20 feet thick. The tubes A A are of wrought iron 8 inches in diameter, provided at their lower extremity with a circular blade D 8 inches high and slightly tapered in form. These tubes are sunk vertically at distances apart varying from 1 foot to 4 feet. Upon their reaching the solid rock below the quicksand, the lower end of each tube is closed with a leaden plug C, fitting into the tapered end-piece and covered with several alternate layers of cement and pitch E to ensure the closure being water-tight. When each of the large tubes A A has been treated in this way, an inner tube B, $2\frac{1}{8}$ inches in diameter, is inserted within the larger tube, and is provided at its lower extremity with an opening F. The large tubes are flanged at G, and by this means attached to a cast-iron branch with three outlets and flanges H, H and J. The outside tubes

* See Transactions, South Wales Institute of Mining Engineers, vol. xv. pp. 143—151.

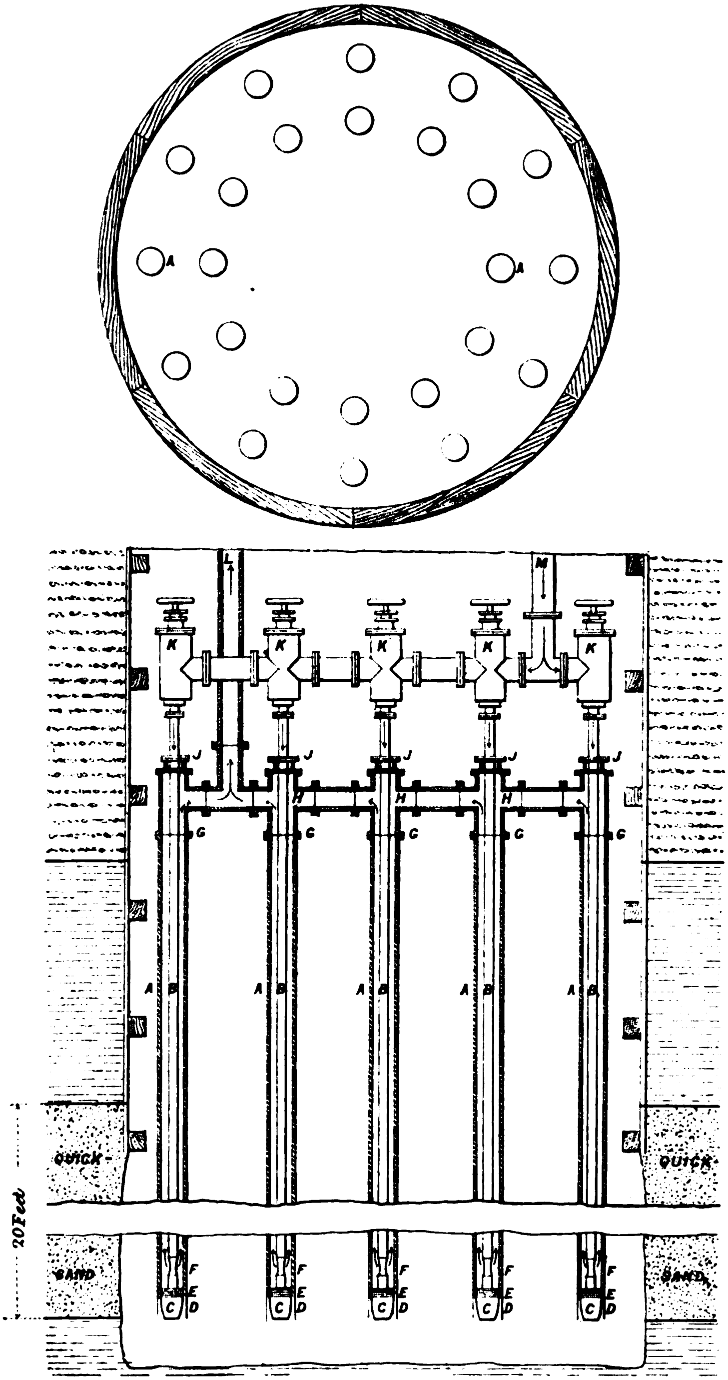


Fig. 28.—Poetsch's Sinking Process

hole for the rods to work through) at the level of the working floor, turn the rods by means of the cross bars at each stroke. The smaller *trépan* (Figs. 43–46) requires modification in its construction, according to the nature of the material to be cut. If soft, the bar to which the teeth are attached is suspended by a fork of wrought iron, but for hard rock it is forged in a single piece and weighs about 8 tons. In ordinary ground this cutter advances about 8 feet per day. The teeth are well steeled, fit into sockets in the main bar, and are further secured by a pin easily removed when the teeth require sharpening or renewing.

The cutter is driven about 9 or 10 strokes a minute usually, but sometimes more, and after being some hours at work, it is raised by a small capstan-engine, with a flat hemp rope of $14\frac{1}{4}$ inches wide by $2\frac{3}{8}$ inches thick. The rods require unscrewing (as in the ordinary manner of boring rods) as the cutter is being withdrawn. To clear the whole of the cut material a sheet-iron cylinder 6 feet long, with two valves in the bottom, is lowered and raised by the rods.

The larger *trépan* (Figs. 40–42) weighs about 16 tons and has a bar of wrought iron, to a portion of which teeth are attached as in the smaller *trépan*. The teeth are fixed on that portion of the bar which exceeds the diameter of the hole cut at the first operation. The large *trépan* is guided below by a cradle of iron bars fitting loosely within the smaller diameter.

The arrangement of teeth is such as to cause them to cut a sloping surface at the bottom of the shaft so as to ensure the cut material to roll into the smaller pit, where they drop into a sheet-iron bucket previously lowered into it. The rate of progress varies from 3 feet per day in ordinary ground to 1 foot per day in hard rock.

To obviate the excessive vibration which would otherwise be imparted to the rods by tools of such a weight, a special joint, called a slide piece, of great strength, is applied.

To maintain the boring rods in a vertical position throughout their stroke, guides are attached to the upper part of the implement. In the smaller cutter these consist of two strong iron bars set at right angles (G, Fig. 43), having teeth fixed at their extremity, which slightly enlarge and, at the same time, smooth down the sides of the hole. For the large *trépan*, one of the cross pieces is rigid and the other, at right angles, is hinged on both sides of the main rod in such manner that it can be lowered or raised by ropes during the shifting of tools, through a small opening in the working floor. The guide, when in position, forms a fixed cross, through the central opening of which the cutter rod slides freely up and down.

The pit having been sunk through the water-bearing strata in this way requires tubbing. In this process the lowermost ring, like all the upper portion, is cast in one piece. The lower flange G, Fig. 39, is turned outwards, its upper flange C inwards, and it rests on a bed in water-tight ground below the water-bearing rocks. Upon the lower flange and all round the ring a wall of well-selected moss, F, is packed tightly against it and secured in its position by a net placed at the back of it. To assist in forcing the moss against the side of the shaft, small sheet-iron springs, E E, are placed above and below, the effect of which is to give the pressure a definite direction. On the moss cushion rests the next ring, D. Its bottom flange is turned outwards, the top one inwards, and it is of such a size as to slide down outside the bottom ring, when the moss is sufficiently pressed down by the weight, and upon this sliding ring the ordinary tubbing rings are built. Each flange is truly planed, and between the flanges a ring of sheet-lead, $\frac{1}{8}$th of an inch thick, is laid. After screwing up the bolts the lead is beaten in on both sides with hammer and chisel. Each ring of tubbing is from $4\frac{1}{2}$ to 5 feet high, is of extra thickness, and tested on the surface by hydraulic pressure. The bottom simple ring is $2\frac{5}{8}$ inches thick, and weighs $11\frac{3}{4}$ tons for a 14-foot pit. The upper rings are gradually lighter.

To facilitate the gradual lowering of the enormous weight of the tubbing, by means of six rods and screws used for this purpose, a diaphragm or false bottom, L, is attached by screw-bolts at a point near the bottom of the tubbing, and this causes it to float on the water. A central equilibrium tube, A, passes up the shaft from the false bottom, and through cocks, placed at intervals, allows of water being poured into the middle of the tubbing in sufficient quantity as may be required to help its descent. By this means no greater weight than 40 tons rests on the suspension rods.

The bottom ring or moss-box is suspended by light rods, B, to the flange of an upper ring, and is lowered into its position on its seat or bed. The weight of tubbing then bears on the moss and squeezes it down and against the sides of the shaft so as to form a thoroughly water-tight joint.

The annular space between the rings and the shaft is filled in with concrete and allowed to consolidate before the water is drawn out of the pit. The success of the undertaking to a large extent depends on the perfection with which the seat of the moss box is cut and smoothed, and to ensure its suitable condition, a gigantic pair of pincers, H, with arms on the principle of a lazy-tongs, is lowered with and underneath the whole of the tubbing by means of a rod passing up through the central tube. By working the rod up and down, the ends of this tool may be made either to expand to the full size of the shaft, or brought closely together and thus pick up small pieces of stone or other material which may be lying on the bed of the shaft; when in its contracted form it may be passed into the central shaft to be out of the way. After the concrete has set, the water is pumped out, the false bottom taken off by unscrewing the bolts which attached it to a flange, and the moss box is examined. For safety, a lower seating is cut and prepared in the rock a few feet deeper, a wedging curb put in by hand in segments, and the tubbing built up on it to the moss box, against which it is securely wedged. The shaft being now free from water may be sunk deeper by ordinary methods.

The explosives used for blasting in sinking are:—(1) gunpowder, which is useful for rending operations: when the rock is not hard and is also free from water it may be used to much advantage; (2) dynamite, which as a "shattering" agent is a very useful explosive in a wet pit and in a hard rock, as the water does not injure it. Its use, however, has proved dangerous in sinking shafts, owing to missed shots remaining undiscovered, and being accidentally exploded in clearing away the *débris*, or in boring fresh holes. (See also Chapter XVI.)

The following list of shafts having a depth of 1,500 feet and upwards is here given, but such list is by no means exhaustive:—

Country.	Colliery or Mine.	Depth in Feet.
Africa, South	Nourse Deep	1,578
	Robinson Deep, S.A.R.	1,991
America, United States.	Potsville shaft (disused), Philadelphia and Reading Coal and Iron Company	2,000
	Kennedy Mine, Jackson, California	2,150
	Grass Valley, Idaho	2,182
	California Mine, Colorado	2,260
	Belcher Mine and Crown Point Mine (silver), Comstock, Nevada	3,033
	Yellow Jacket (silver), Comstock, Nevada	3,123
	Tamarack (copper), Lake Superior	(1) 4,450
	Red Jacket, Calumet and Hecla, Lake Superior	(2) 4,900
Australia, Victoria.	Victory and Pandora (gold)	(3) 1,872
	Newington (gold)	1,941

Country.	Colliery or Mine.	Depth in Feet.
Australia, Victoria (*continued*).	Magdala, Stawell	2,409
	Lazarus, Bendigo	3,024
	Lansell's (gold), Bendigo	3,302
Austria-Hungary.	Amalia, Schemnitz, Hungary	1,750
	Einigkeit, Joachimsthal, Bohemia	1,750
	Procopi, Przibram, Bohemia	2,900
	Franz Joseph, Przibram	2,900
	Anna, Przibram	3,100
	Maria, Przibram	3,281
	Adalbert (lead, etc.), Przibram	(4) 3,672
Belgium.	Cécile Pit, Seraing	(5) 1,710
	Marihaye Colliery, Liége	2,100
	Houssu Colliery, Centre	2,300
	Sacre Madame Colliery, Charleroi	2,499
	St. Joseph Réunion Colliery, Charleroi	2,763
	Cipley Colliery, Mons	2,950
	St. André shaft, Poirier Colliery, Charleroi	3,100
	Marchienne Colliery	(6) 3,117
	Viernoy shaft, Anderlues	3,300
	Simon Lambert (disused)	3,489
	Viviers shaft, Gilly, Charleroi	3,750
	Produits Colliery, Mons	(7) 3,937
France.	Ronchamp Colliery, Haut-Saône	1,870
	Hottinguer shaft, Epinac	(8) 2,000
	Treuil Colliery, St. Etienne	2,034
	Montchanin Colliery, Le Creuzot	2,300
Germany.	Freiberg, Saxony (maximum depth)	2,060
	Camphausen Colliery, Saarbrücken	2,296
	Maria Colliery, Hongen, Westphalia	2,300
	Hansa Colliery, Huckarde, Westphalia	2,330
	Concordia Colliery, Olsnitz, Saxony	2,420
	Frieden Colliery, Olsnitz, Saxony	2,515
	St. André (silver), Prussia	2,532
	Samson, St. Andreasberg, Harz	2,560
	Einigkeit, Lugau, Saxony	2,620
	Zwickau (lead, etc.), Saxony	2,637
	Kaiser Wilhelm II., Clausthal, Harz	2,960
Great Britain.	Wollaton Colliery, near Nottingham	(9) 1,500
	Mardy Colliery, No. 3 shaft, Glamorganshire	(10) 1,500
	Tankerville Lead Mine, Shropshire	1,500
	Wheal Sisters, Cornwall	1,500
	Avon Colliery, Glamorganshire	(11) 1,504
	Downside Colliery, Strap Pit, near Bath (disused)	(12) 1,520
	Kingswood Colliery, near Bristol	1,524
	Norley Collieries, Wigan, Lancashire	(13) 1,525
	Hoyland Silkstone Colliery, Barnsley, Yorkshire	(14) 1,530
	Barrow Colliery, Barnsley, Yorkshire	1,530
	New Sharlstone Colliery, Barnsley, Yorkshire	(15) 1,533
	Allen's Green Pit, near Radcliffe, Lancashire	(16) 1,539
	Parr Collieries, St. Helen's, Lancashire	(17) 1,548
	Malago Vale Colliery, Bedminster, Bristol	(18) 1,554
	Pleasley Colliery, Mansfield, Nottinghamshire	(19) 1,560
	Penrhiwceiber Colliery, Mountain Ash, Glamorganshire	1,581
	Boldon Colliery, near Sunderland, Durham	(20) 1,590
	Hickleton Main Colliery, between Barnsley and Doncaster	(21) 1,612
	Clifton Hall Colliery, near Manchester	(22) 1,620
	Langwith Colliery, Nottinghamshire	1,620
	White Moss Colliery, near Ormskirk, Lancashire	1,629
	Ryhope Colliery, West Pit, near Sunderland, Durham	(23) 1,630
	Albion Colliery, Cilfynydd, near Pontypridd, Glamorganshire	(24) 1,635

(1) The average winding speed in the shaft is 3,200 feet per minute.

(2) A pair of quadruple engines is used for winding, each with its four cylinders, 18 inches, 27¾ inches, 48 inches, and 90 inches, in diameter, and stroke of 60 inches, driving, through gearing, conical winding drums, 14 to 24 feet in diameter and 12 feet wide, which lift cages carrying 6 tons of ore at the rate of a mile in one and a half minutes.

(3) 25 gold mines in Victoria exceed 2,000 feet in depth.

(4) At the deep shafts of Przibram tapering wire ropes made of special crucible cast steel, having a tensile strength of 114 to 120 tons per square inch, are used for winding.

(5) Cécile Pit 14¾ feet diameter. There are nine shafts in Belgium exceeding 2,610 feet in depth. The average depth of all Belgian collieries is 1,420 feet.

(6) At the Providence shaft, Marchienne, flat tapered ropes made of crucible cast steel are used for winding. The breadth of the rope varies from 7·87 inches at the thick to 6·69 inches at the thin end, and the average weight is 8·2 lbs. per foot, and last about 12 months. The winding engines, of 2,000 horse-power, have cylinders of 43 inches in diameter and 78 inches stroke, and raise a load of 12¼ tons from a depth of 3,117 feet. The load consists of 6¼ tons, the weight of the cage and tubs and 6 tons of coal.

(7) At the Sainte Henriette shaft, flat aloe ropes are used to lift a load of 6¼ tons from a depth of 3,937 feet. The ropes taper in breadth from 16·5 inches to 8·6, and in thickness from 1·93 inch to 1·14. The average weight per foot is 7·4 lbs. The ropes last about 24 months.

(8) Pneumatic hoisting apparatus successfully applied for ten years, thus altogether dispensing with the use of winding ropes. A tube, 5 feet 3 inches in diameter, fitted with a piston, was fixed in the shaft. The piston was attached to a nine-deck cage, carrying 9 tubs, each holding 10 cwt. of coal. To raise the loaded cage the air above the piston was exhausted, the atmospheric pressure beneath then gradually forcing up the load. For the descent, exhaustion was stopped and air allowed to pass upon the top of the piston. The tubs were removed from the cage three at a time by means of three double doors provided in the tube both at the top and the bottom.

(9) The west shaft is an upcast 12 feet in diameter, sunk 1,500 feet and bored about 300 feet further to prove the Kilburn and other seams of the Lower Coal Measures.

(10) No. 3 is an upcast shaft 16 feet in diameter and 1,500 feet in depth to the Five-feet Seam. It was sunk in 1894 and 1895. The shaft lining consists of 9-inch brickwork set in Aberthaw lias lime. In places the brickwork is from 18 to 24 inches in thickness.

(11) 1,449 feet to Six-feet Seam loading stage, there being 55 feet of sump. There are two shafts sunk, each being 15 feet in diameter inside the walling.

(12) The sinking of this shaft was continued from the depth of 660 feet to that of 1,520 feet, the deepening having been completed December 9, 1874. The strata dip north at an angle of 40°.

(13) To the Arley Mine or Orrell four-feet Seam. No. 4, an upcast and winding shaft, is 12 feet in diameter; No. 5, a downcast and winding shaft, is 16 feet in diameter, and 292 yards south-east of No. 4. The shafts are lined with ordinary bricks and mortar without the insertion of any cast-iron tubbing. Considerable feeders of water were encountered in sinking the No. 5 shaft, which are still being raised to the surface.

(14) 20 feet diameter.

(15) To Haigh Moor Seam.

(16) To the Cannel Mine, which is 216 feet below the Trencherbone Seam. The shaft is a downcast 11½ feet in diameter, lined throughout, there being 1,013 feet of 9-inch brick walling from the surface, then 67 feet of cast-iron tubbing and 459 feet of brick walling to the bottom.

(17) Two shafts, Nos. 4 and 5, each 18 feet in diameter, are sunk 1,548 feet to the Arley Mine.

(18) To the Bedminster Little Seam. There are two shafts, situated 66 yards apart. The downcast, called Malago, or Old Pit, is 14 feet in diameter, and 1,500 feet in depth to the shaft inset from which a cross-measure drift or branch is driven to cut the Bedminster Great Vein. The upcast, called Argus, or New Pit, is a furnace shaft 15 feet in diameter, and 1,740 feet in depth. From the bottom of this shaft a level cross-measure drift is driven. Both shafts are walled throughout, and fitted up for the daily raising and lowering of persons and the raising of minerals.

(19) Two shafts, 80 yards apart from their centres, each 14½ feet in diameter, are sunk 1,542 feet to the Top Hard Seam. The sinkings pierced 250 feet of Magnesian Limestone or Permian strata, through which and water-bearing strata below, considerable feeders of water issued, the maximum amounting to nearly 1,000 gallons a minute. The lining of the shafts consists of 54 feet of 18-inch brickwork from the surface, followed by 351 feet of cast-iron tubbing, and then 1,137 feet of 9-inch red brick walling to the floor of the Top Hard Seam.

(20) To the Hutton Seam; 1,458 feet to the Bensham Seam.

(21) Two shafts, 65 yards apart, are sunk 1,612 feet to the Barnsley Seam. Both shafts are 18 feet in clear diameter and lined as follows:—9 feet of 9-inch red brickwork set in cement from the surface, then 150 feet of cast-iron tubbing, followed by 1,453 feet of ordinary red brickwork, the thickness of which varies from 9 to 14 inches. The sinking was rapidly performed. Begun in December, 1892, it was completed in June,

1894, this period of nineteen months including the placing of pumps and the insertion of one wedging crib, tubbing and walling in each shaft. The shaft at which the pumping engine was placed was continuously kept deeper than the other, which was consequently more or less drained by the same engine.

([22]) 1,605 feet to the Trencherbone Seam. The upcast shaft is 9 feet in diameter and the downcast 10 feet. Both shafts are lined throughout, there being about 300 feet of cast-iron tubbing in each, the remainder consisting of ordinary 9-inch brickwork.

([23]) To the Hutton Seam. A boring was made from the Hutton Seam in the shaft to a depth of 114 feet, the shaft and boring together thus proving 1,744 feet of strata from the surface.

([24]) There are two shafts, each being 19 feet in clear diameter, walled throughout. The sinking commenced in January, 1885, and was finished in February, 1887. On Dec. 21, 1887, 1,000 tons coal were raised.

([25]) To the Arley Mine. Shaft 17 feet in clear diameter, belled out to 20 feet near the bottom.

([26]) The east shaft is a downcast 18 feet in diameter and 1,590 in depth to the Lower four-feet Seam, there being 54 feet of sump. The sinking was rapidly performed. Commenced in 1888, it was completed to the Lower four-feet Seam in 15 months, including the lining of 9-inch brickwork set in Aberthaw lias lime. There was no trouble with the water, because it drained to the west shaft either naturally or through drifts. The west shaft is an upcast, also 18 feet in diameter, 43 yards distant from the east shaft. Begun in 1883, the sinking reached the Red Seam, 1,320 feet from the surface, 23 months afterwards. Later a shaft was sunk 270 feet further to give a connection between the Lower four-feet Seam and the upcast. Various beds of sandstone were penetrated in sinking, and these yielded water, but no attempt was made to keep such water back by cast-iron tubbing or water-tight walling. The west shaft was lined nearly throughout with firebrick, 9 inches long, tapered from 5¼ to 6 inches in breadth at the back and 4 inches deep, set in Aberthaw lias lime. The space between the back of the 9-inch brickwork and the sides of the shaft was filled up with rubble stones without mortar. Open spaces were left in the rubble 3 inches square at intervals around the shaft, which allowed the water to drain into garlands of cast iron placed about every 15 yards in depth, the shaft being gradually enlarged for their reception.

([27]) To the Yard Seam. Two shafts, each 15 feet in clear diameter, are sunk at a distance of 123 yards apart. The No. 1, or north shaft, is a downcast and has three sections of cast-iron tubbing inserted, the entire length of which is 360 feet. The No. 2 or south shaft is an upcast and winding shaft; it has two sections of cast-iron tubbing, making together 240 feet in length. The remaining portions of the shafts are lined with ordinary brickwork.

([28]) To a horizontal drift to cut the Jewel Seam, which was intersected in the shaft 1,548 feet from the surface. Inclination of the measures 1 in 2·3. The sinking of the shaft, 20 feet in clear diameter, was begun in March, 1890, and completed in December, 1893. Some feeders of water were kept back from the shaft by means of concrete walling faced by 9 inches of red brickwork. By means of a movable steel scaffold suspended on wire ropes, the masons' work of lining the shaft with bricks proceeded simultaneously with the excavations in the pit bottom.

([29]) 1,692 feet to the Lower Florida Seam; 33 feet of sump. No. 1 is a downcast shaft 16 feet in diameter, and No. 2 is an upcast shaft 14 feet in diameter, the two shafts being of the same depth.

([30]) Two shafts are sunk, 22 yards apart from centres, each 9½ feet in diameter, and 1,740 feet in depth to the Doe Seam. The sinking of the shafts occupied a period from 1852 to 1856. The lining consists of 50 feet of brick walling set in cement from the surface, then 450 feet of cast-iron tubbing, following which are 1,240 feet of ordinary 9-inch brick walling. In 1896, two new shafts were being sunk; No. 3, a downcast 15 feet in clear diameter, and No. 4, an upcast 14 feet in clear diameter, 30 yards north of No. 3 shaft. The estimated depth to the Trencherbone Seam at these new shafts was 2,250 feet. In the No. 3 shaft a stratum of quicksand 4 feet in thickness, 8 feet below the surface, was piled through, a double row of piles having been used.

([31]) 1,701 feet to the Hutton Seam, below which are 44 feet of sump. A boring was made from the bottom of the sump and proved the Harvey Seam 117 feet below or 1,862 feet from the surface. The Maudlin Seam was cut in the shaft 1,589 feet from the surface. The Monkwearmouth pits are notable as being amongst the foremost which pierced the Magnesian Limestone before reaching the Coal Measures. The sinking, which was commenced in May, 1826, did not reach the coal measures until August, 1831. The thickness of strata over the coal measures is 330 feet, of which 117 feet are alluvial, and 213 feet Magnesian Limestone. At the bottom of the limestone, the feeders of water amounted to 3,000 gallons per minute. These were successfully dealt with during the sinking, and effectually kept back by means of cast-iron tubbing. For many years, this colliery was regarded with great interest, both from a scientific and a commercial point of view, because of the speculative nature of the enterprise, and the indomitable perseverance shown in sinking a to the Black Mine 2,852 feet, while the total depth of the shafts is 2,880 feet. The seams dip 1 in 4. Barometer readings, March 1, 1887, were:—

At the surface	30·10 inches.
At bottom of the shaft	33·30 ,,

Time occupied in drawing loaded cage up the shaft 1 min. 25 sec.; time occupied in changing 35 seconds. Boiler fires at the bottom of the upcast shaft and a furnace placed midway between top and bottom produce the ventilation, amounting to about 100,000 cubic feet per minute. Very little firedamp is given off in the workings.

(69) Depth of shaft, 1,575 feet to the Ram's Mine. From the bottom of the shaft an incline has been driven a distance of 4,700 feet, dipping about 1 in 3. The downcast shaft is 8 feet in diameter, the upcast 7½ feet. A furnace is placed at the bottom of the latter.

At the commencement of sinking the Ashton Moss shafts, which are the deepest in the United Kingdom, difficulty in the form of an alluvial deposit presented itself. This consists of different layers of clay, clay-marl, gravel, and quicksand, and extends from the soil immediately under the surface to coal-measure rock, the distance from the surface being 49 yards, 2 feet, 7 inches. The layers of gravel and quicksand were copiously watered, and the whole covering over the coal-measures was of the loose character peculiar to such deposits. Shafts in other parts of the country have encountered similar alluvial deposits caused by the gradual settlement of sediment from flowing waters. These deposits vary in character, but are mostly of clay, silt, gravel, or sand. If the sand contains much water it becomes a quicksand, and on being tapped, runs into the pit from all sides and gives much trouble. Sometimes large boulders or pebbles are embedded in the clay, and sometimes sand and gravel are intermixed. At the Norwood Colliery, near Gateshead, in the county of Durham, beneath a few inches of soil, about 100 feet of alluvial deposit was pierced, the top layer being of dark mud with vegetable matter, closely resembling peat.

Quicksands are not only troublesome in the sinking of a pit, but also a source of danger to machinery and buildings on the surface, because of the insecurity of foundation. Any extensive rush of quicksand into the sinking shaft at no great depth below the surface, or any gradual up-rising of the pit bottom, is almost certain to be followed by a subsidence of the surface and possibly serious damage to costly machinery. Poetsch's freezing system already described has been most successfully applied under these circumstances.

The New Red Sandstone contains soft beds of sandstone, which have presented great difficulties to those sinking shafts through them in order to win the coal measures. The soft sandstones are heavily watered, and the force of flow into the sinking shaft carries down disintegrated portions which settle at the bottom of the shaft. Here they impede the working of the pumps, by wearing the bucket leathers, and otherwise add to the labour of sinking, inasmuch as the running sand has to be removed.

The method of conducting sinking operations through the drift or alluvial accumulation at the Ashton Moss Colliery is shown in Figs. 47, 48, and 49.

These, however, do not show the alternations of clay, gravel, and quicksand in the drift. The shaft was carried down through the overlying material 24 feet in diameter inside the 9-inch brick lining shown in the Figs. until it reached the top bed of the coal-measures. During the sinking this 9-inch ring of brickwork was continued from the surface downward in sections, as it was required. In order that it might be upheld securely, the walling was entirely suspended from the surface, four massive wood beams being placed across the top of the shaft to carry the weight, which was distributed by means of 12 wrought-iron hanging rods 2½ inches in diameter. From this point the sinking was resumed at a reduced diameter of 17 feet 6 inches, and continued until a sound bed was reached which would carry the permanent walling crib. This was then carefully laid and a 9-inch ring of brickwork built upon it. When this reached the bottom of the alluvial deposits where the shaft widened, a 4½-inch brick ring was started off the coal-

CHAPTER V.

FITTING UP THE SHAFT AND SURFACE ARRANGEMENTS.

Arrangement of Pit Bottom for Small and Large Trams—Shaft Gates—Conductors—Buntons—Keeps—Pit Cages—Safety Cages—Detaching Hooks—Pit Head-gear—Pulleys—Ropes—Capping Round and Flat Ropes—Observations for Users of Ropes—Tables of different qualities of Round and Flat Ropes and of Chains—Method of Splicing Ropes—Shaft Signals—Pit Stage—Tipplers—Screens and under Railways—Winding Engines—Conical and Spiral Drums—Steam-brake to prevent over-winding—Counterbalancing the Load in Shaft—Rules for Winding Engines—Calculations of Sizes required under given Conditions—Questions and Answers on Steam and Steam-engines—Systems of Winding Coal up Shafts without using Drums.

THE shaft bottom and roadways, for some distance, leading from the pit bottom are generally arched. Where small trams are to be used the space round about the shaft bottom is usually laid with flat sheet-iron for facilitating the operations.

Where large trams are used rails are laid leading to each cage from opposite directions. This allows of the empty trams being propelled from the cage on one side as the loaded ones enter it at the other.

Where flat sheets are used, they allow of light full tubs, or the lighter empties, being quickly turned in any direction without having to follow a particular course.

The pit is sunk a few feet below the level of the flat sheets to form a sump, and into this the water (if any) drains; from thence it is raised direct by the pumps placed in the shaft or conveyed elsewhere to be dealt with. The pit bottom is arranged so that the loaded tubs are pushed towards the cage down a slightly falling road, and the empty tubs pass out of the cage on the opposite side of the shaft.

The top of the pit and any intermediate loading places between the top and bottom, are provided with gates for the protection of those moving about.

Shafts are fitted with CONDUCTORS or GUIDES, which, if of wood or iron, are attached to buntons or crosspieces fixed across the pit and which have either been built into the walling or are afterwards let into it. The strength of the buntons must be proportioned to the size of shaft and the weight of the load; for a shaft 10 feet in diameter with single cages carrying one tram of 12 or 15 cwts., Memel or red pine, 9 inches by 3 inches, placed at intervals of six feet in the shaft, would be sufficient. The guides (if of wood) should also be of Memel pine, not less than 4 inches by 3 inches in section, and properly bolted to the buntons. Bolts and nuts are preferable to wood screws which are often used for this purpose. There is usually only one guide on each side of the cage, but the arrangements respecting them are various, according to the requirements of the case. Frequently, instead of wood, bridge or single headed rails are used for guides, and in some cases angle iron, they being kept in line by suitable fish-plates and bolts, and securely fastened by bolts to the buntons. In Lancashire and Yorkshire some pits have guides consisting of round bars of iron fixed at the pit bottom and screwed up to the head frame. There are two rods for each cage, the cross bar of which, having a ring at each end, runs upon the rods.

In most of the large collieries in South Wales wire-ropes are used as guides, fixed to wooden balks at the shaft bottom and to the head frame, where they are tightened by screws; another means of keeping them tight is to suspend heavy weights from their lower extremities beneath the balks, or by weights hanging over pulleys on the surface.

Where the depth and consequently the cage-speed is great, three and sometimes four of these guides are required to prevent excessive vibration. In some instances two additional ropes are suspended between the cages to prevent one cage from catching the other in passing. Rigid guides are so fixed that the cages shall have not less than 9 inches of clearance as they pass each other, and if iron wire guides are used and the pit a deep one there should be from 12 inches to 18 inches of clearance, according to the depth of the shaft, the number of guides used, and the speed of the cages in the pit.

"KEEPS," "FANS," or "SHUTS" are supports for the cage on its arriving at the surface or shaft bottom, and at intermediate loading places, if there be any. They are arrangements of counterbalanced levers, and those placed on the pit top offer no obstacle to the ascent of the cage, which after passing by the "keeps" is lowered by the engine-man on to the supports. With double-decked cages, when the tub on the bottom deck has been changed and a signal received that the tub in the top deck (which it must be remembered stands on the shaft bottom "keeps," when the bottom deck of the other cage is on the "keeps" at the surface) is also changed, the engine-man lifts the cage from the supports, and the attendant, by means of a lever, pulls them back clear of the cage until the bottom deck is lowered below them, when the attendant lets go his hold of the handle and they form a support to the top deck. During the change here, the tub in the bottom deck is changed at the pit bottom, and this being effected, the cage is lifted by the engine-man, the attendant pulls back the "keeps," the cage is lowered, and when it has descended clear of the "keeps" they are allowed to spring back ready for use again. The "keeps" at the shaft bottom are necessarily handled differently. As the loaded cage leaves the shaft bottom the attendant there pulls the handle of the "keeps" back and secures it there, by this means preventing the "keeps" from protruding in the pit. As the cage in its downward course approaches him, he takes the handle of the lever which works the "keeps" in his hand, and having allowed the bottom deck of the cage to pass below the level of the "keeps" they are allowed to spring out and support the top deck of the cage. The tub is changed here whilst the bottom deck of the other cage is changed at the surface. The "keep" handle will not require further attention from the attendant below until the cage has left for the surface when he secures the handle back in its place.

Stauss's *patent keeps*, shown in Figs. 50 to 55, have been designed to dispense with the lifting of cages before they are lowered into the pit.

Figs. 50 and 51 show the arrangement of these keeps, fixed on wooden spring cantilevers, to take the shock of the cage when lowered on to the keeps, and Figs. 52 to 55 show the details.

The wear and tear of the winding ropes and engines are reduced, because owing to there being no lifting of the cage before descending, the accompanying jerks are avoided; and jerks are the main cause of deterioration of ropes and engines. When these keeps are used, the winding rope must of course be adjusted in length, so that the cage does not fall after drawing back the keeps. The sinking of the cage should not amount to more than the slack of the rope when unloaded.

The length of the rope can easily be adjusted by means of Freudenberg's *cage-adjusting hangers*, as shown in Figs. 56 and 57, or similar appliances. This is only necessary for the first few days with new ropes; later on very seldom.

The arrangement works as follows: If the cage is to be held and prevented from going down the pit, it is lowered on to the keeps *c*; these are supported by the surfaces *x* and the pin *b*, the latter being prevented from moving vertically or horizontally. It is prevented from moving vertically by the hanging links *e*, which press upwards against the shaft *d*, and horizontally by the toggle *f*, which thrusts against the pin *i* of the lever *k*, and through the latter against the rocking-shaft *l*. The whole arrangement is thus locked and any movement prevented, as the weight of the hand-lever *h* presses the lever *k* downwards upon the block *m*, which is fixed to the bed-plate *a*; and prevents any further downward movement. When *h* is thrown over into the dotted position shown in Fig. 52, through an arc of 60°, *i* comes into position *i'* and *b* to *b'*, thus drawing the keeps *c* away from under the cage and downwards, so that the cage is free to descend. Fig. 55 shows the relative positions of the different parts when the keeps are drawn back.

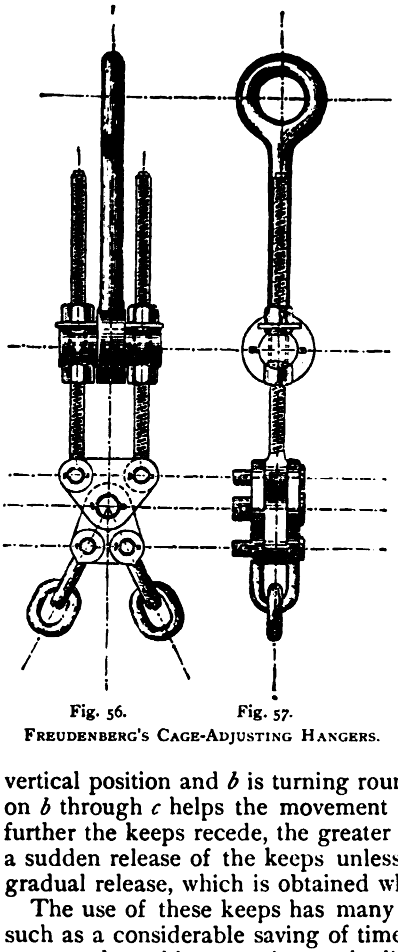


Fig. 56. Fig. 57.
FREUDENBERG'S CAGE-ADJUSTING HANGERS.

When the cage has again been lifted to the pithead, *h* is pushed over into its first position, this action carrying the keeps forward and the cage can again be lowered on to them.

Owing to the keeps *c* being free to turn round the pin *b*, the cage cannot catch fast in them, if they should be pushed out too soon. No injury can result from such action, because the cage would then only lift the keeps up into the dotted position in Fig. 52, and after it had passed above them they would fall back again into their proper position by gravity.

The friction which takes place between the surfaces *x* and *y* and in the joints when the keeps are withdrawn is easily overcome by the hand-lever, as the weight of the cage itself helps and endeavours to push the keeps backwards down the incline of 9° marked *x*. Further, as soon as the hanging links *e* have left the vertical position and *b* is turning round *d* as centre, the weight of the cage acting on *b* through *c* helps the movement of rotation and the force acting on *i*. The further the keeps recede, the greater this force becomes, until it may finally cause a sudden release of the keeps unless the angle of the incline *x* is suitable for gradual release, which is obtained when *x* is inclined at an angle of 9°.

The use of these keeps has many advantages as compared with other makes, such as a considerable saving of time and of steam, and less wear and tear on the ropes and machinery, owing to the lifting of the cage to release the keeps being unnecessary. This saving of time amounts to from 3 to 6 seconds, according to the skill of the engine-man, for each lift, with single-deck cages; and for cages with several decks the saving is more in proportion.

The CAGE is a receptacle for the tubs traversing the pit either empty or full. It is also the usual means of transport for the workmen and all others between the surface and the different loading stages in the shaft. The pit timber, workmen's tools, horse food, and water, and frequently the horses themselves, are lowered by means of the cage.

When men are riding in one cage no loaded or empty tubs are placed in the other, or in an under or over deck of that holding them. If men are not in both cages one is allowed to run empty.

The cage is usually made of wrought iron but sometimes of steel. As to its form, it is of course governed by the shape of the division of the shaft it has

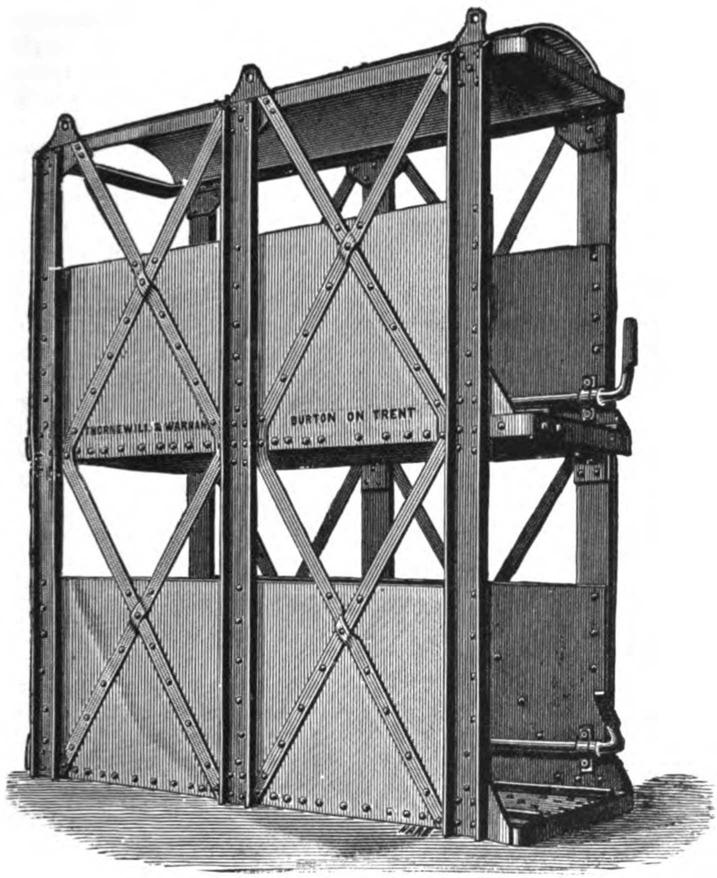


Fig. 58.—Double-decked Cage.

to run in, and it may be single or double decked, or have more decks than two if desirable. Again each deck may have one or more tubs placed in it as may be desired and arranged. Each deck floor is laid with rails to allow of the tubs being pushed in whilst the loaded ones are pulled out on opposite sides. There are various modes of keeping the tub secure in the cage during its ascent or descent. One of these is by having "false bottoms" in the cage, which is an arrangement whereby the floor of that part of the deck on which the tubs are placed falls or sinks an inch or two below the other and outer portions of the deck, but when the cage rests on the keeps all the deck floor is on one level, allowing the tubs to be changed.

Another mode of securing tubs in the cage is a bar running through the cage, and at either end is placed a short lever which turns down or up on being pushed;

when down it covers the ends of the tubs and prevents their moving, when up it allows them to be changed.

The best form of catch is that which grips the axles on the tub being pushed in, without the necessity for the attendant to have to put his foot on the cage to work the catch. Sometimes a catch in the floor secures the tubs in place.

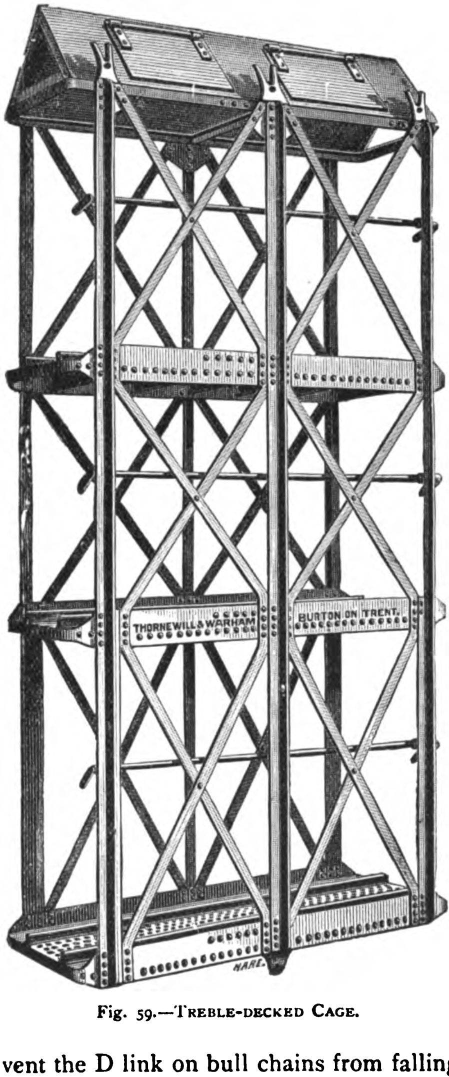


Fig. 59.—Treble-decked Cage.

The slides of the cage fit loosely to three sides of the wooden conductors, and are slightly bell-mouthed. They are applied at the upper and lower bars of the framing.

A two-decked cage would have 3 such slides on either side of it, in the usual arrangement adopted. The top of the cage is provided with an iron bonnet or cover for the protection of persons whilst descending or ascending. The cage is suspended from the rope by four short chains called "bridle" or "bull" chains, one being at each of the upper corners, and in the case of heavy cages from the middle of the longer sides as well, so that in the latter case there would be six bridle chains.

Fig. 58 shows a double-decked cage, and Fig. 59 a treble-decked cage, as made by Messrs. Thornewill & Warham, Engineers, Burton-on-Trent. The former is steel throughout, the deck frames being angle steel, the uprights of channel steel, and the bracings of flat steel. The deck frames have cross-bearers of angle steel, with "knee" ends riveted to the frames. The floor of each deck consists of perforated steel sheets.

The cage-hangers are of forged steel and sufficiently large to form a gusset to receive the uprights, cross-bracing, and top frame; they have horns forged on them to prevent the D link on bull chains from falling over when the chains are slack.

The treble-decked cage is of similar construction, the uprights being of angle instead of channel steel.

The tub-catches are plain bars with bent ends, working in suitable chocks fixed to the uprights. The roof of each cage is provided with doors, so that long pit wood and other articles may be carried on the upper deck. All rivet holes are drilled and rivets where possible put in by machine. The cages are provided with guide-cheeks for square, or loops for wire rope conductors, and each deck is fitted with rails. The finished weight of the double-decked cage is 2 tons, 3 cwts., 2 qrs., without the bull-chains. They are in daily use at the Walsall Wood

Detaching Hooks.—Closely associated with safety cages are contrivances of a similar character for preventing the cage from falling when severed from the rope through overwinding. The object sought to be accomplished in these overwinding safety appliances is to cause the link by which the cage is suspended from the rope to release its hold of the rope and take hold of a portion of the framework of the headgear.

The Mines Act, 1887, does not make the use of any overwinding appliance compulsory, but a limit of speed is fixed, when men are being raised, if the mine is not provided with an automatic contrivance to prevent overwinding.

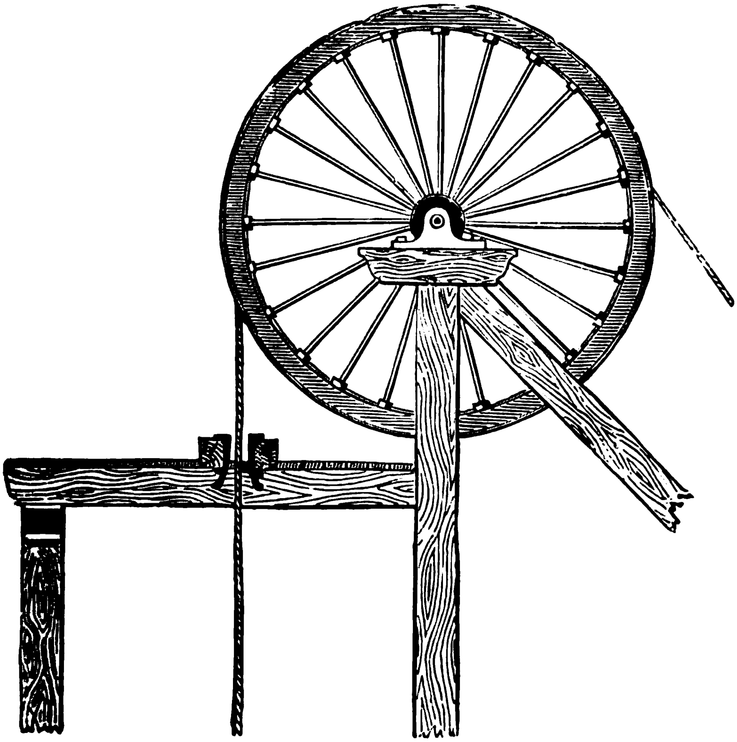


Fig. 64.—Arrangement of Catch-plate in Headgear for Ormerod's Safety Link.

Ormerod's is one of the best of these safety links, and is the invention of Mr. Edward Ormerod, of Atherton, near Manchester.

Fig. 63 (A) is a cross view of the link.

Fig. 63 (B) is a side view of the same.

Fig. 63 (C) represents the position it assumes when wound up into the cylinder, the rope shackle being disconnected, and the link firmly locked in its position.

Fig. 63 (D) shows the rope shackle re-connected for lowering the link through the cylinder.

Fig. 64 shows a section of the cylinder as fixed in the headgear or pit frame, also the platform for convenience in re-connecting the rope shackle.

It will be seen on reference to the engravings that the apparatus when in ordinary use, as in Fig. 63 (B), is wider at the bottom than the top; but in the event of overwinding, the link is drawn into the bell-mouthed cylinder F F in Fig. 63 (C), the wide part of the link at H H coming in contact with the cylinder at F F, thereby closing the bottom part of the link, also causing the top part to expand and the projections to catch over the top of the cylinder, while at the same time the rope shackle A is forced out of its seat, thus being allowed to go free; the bottom shackle B drops into the slot D and locks the link firmly in its position. The cage being suspended from the chain cannot fall back. To prevent the possibility of the link becoming disarranged in ordinary work, a small pin, P, is inserted through the plates, which pin is sheared off as the apparatus passes into the cylinder.

For lowering the cage the shackle is attached to the ear on the middle plate as shown in Fig. 63 (D). On removing the pin C, and slightly winding the rope, the middle plate (having a slotted hole in it) is elevated into the position shown, and allows the apparatus to pass down through the cylinder, and safely lower the cage.

The following advantages are claimed for it :—

1. It is self-contained. The load in ordinary work being carried from the outside plates only, thereby avoiding appreciable wear to its working parts.
2. The clasp stud E, which lips over the top part of the outside plates, considerably strengthens the hook in case of any excessive strain or jerk, and also assists the hook in taking the first shock in case of overwinding at a very high speed.
3. When detached the middle plate constitutes additional metal thrown out, and the hook is therefore actually considerably stronger than when in its working position.
4. The bell-mouthed cylinder for detaching the hook is a substantial and exceedingly strong fixing for the headgear, and is never liable to be torn away, neither does it collapse or injure the ropes through vibration.
5. The cylinder also affords a much more effective entrance for the hook than any other appliance whatever.

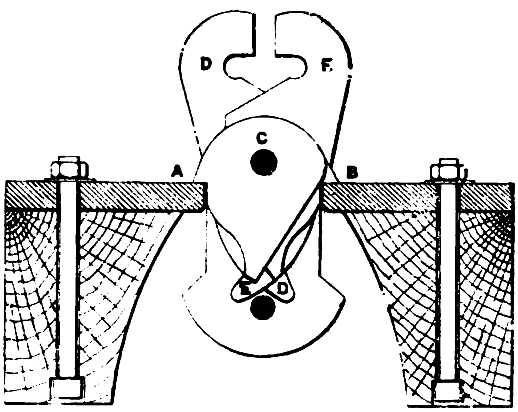


Fig. 65.—Forster and Brindle's Detaching Hook.

Forster & Brindle's is another of these detaching hooks, and is shown in Fig. 65. Two similar plates, A and B, are suspended upon a pin, C, and are made so that in passing through the detaching plate they are pressed in. On emerging from the upper end of the detaching plate they open out again so as to prevent the hook from returning. In case of a partial overwind the hook enters the detaching plate, but perhaps not sufficiently to cause shearing of the rivet which must precede detachment; the plates then clutch the point of sustainment and prevent the cage from falling back. Where the overwind is complete, the plates A and B are pressed in as they pass through the sustaining plate, and immediately after being through, are pressed out by plates D and E, which also cause detachment. The rope is expeditiously re-attached, to effect which the connection with the rope is made, and the weight tightened on it, the two plates, A and B, are pressed in by hand, and kept in whilst being lowered through the detaching plate, after which work may be resumed.

There are many other patent detaching hooks, but the difference between them and those described is not sufficient to justify a description of each.

To some extent the same remarks apply to the use of disconnecting or a separate ladder provided, so as to admit of an attendant going up to examine the head-gear and oil the pulleys.

Each upright should rest in an iron footing placed upon a specially prepared pillar to take the vertical pressure. The backstays should each rest on ashlar

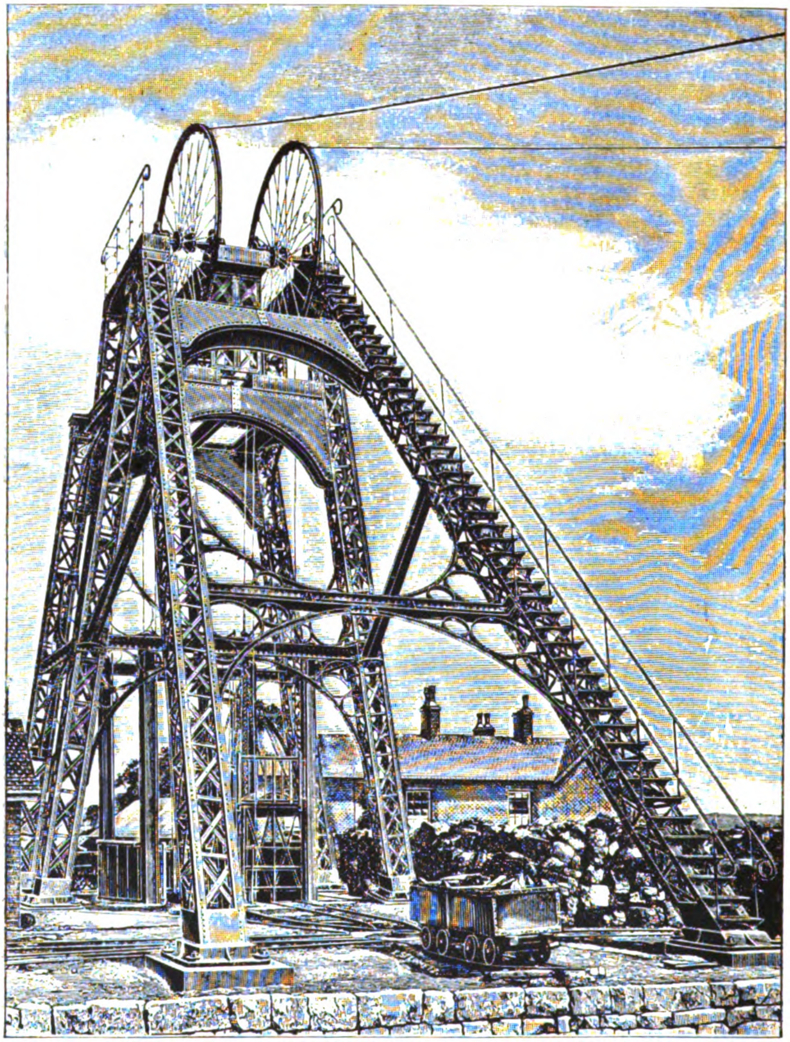


Fig. 67.—Pit-head Frame erected by Messrs. Thornewill & Warham at the New Hall Park Collieries.

or concrete foundation, and not against the engine house wall. Where space is limited and the engine house is necessarily erected rather near the shaft, there is no objection to the backstays being taken through the engine house wall so as to rest against the engine pillars. They should always be taken up to the centre of the pulleys, and not as sometimes seen to a point below this which gives the backstays less resisting power to the strain on them. The whole framework should be frequently painted for the preservation of the material composing it.

The pulleys, usually placed side by side, have wrought-iron arms, the rim and central boss being generally of cast iron. They should be placed with proper regard to the lead of the ropes off the drum to the pulleys, so that the angle of the rope to each is equal. The rim is made to suit the kind of rope to be used, and is grooved accordingly. Pulleys are usually from 10 to 20 feet in diameter. In deciding on a suitable size, it must be borne in mind that ropes receive more injury from working over small pulleys than over large ones, and consequently wear out quicker. A good plan is to have the pulleys the same size as the drum of the winding engine, a rule for ascertaining which is given later in this Chapter. Provision must always be made for their adjustment and oiling.

Fig. 67 shows a Pit-head frame which is elegant in appearance and of great strength, as made by Messrs. Thornewill & Warham, Engineers, of Burton-on-Trent, and erected at the Earl of Carnarvon's New Hall Park Collieries, near Burton-on-Trent. The general arrangement of the pit bank is also shown in the Figure.

The height from pulley centres to pit bank is 40 feet, the pulleys being 15 feet in diameter. The legs are of open lattice work with $4'' \times 4'' \times \frac{1}{2}''$ angle iron bars connected by flat bars $2\frac{1}{2}'' \times \frac{1}{2}''$, and are suitably cross-braced and stiffened by plate girders and spandrils of various sections.

The platform around the pulleys is fenced and access thereto is obtained by a stairway on one of the back-legs.

The pulleys are of the usual type, having cast-iron rims and bosses with wrought-iron arms, and are fitted with steel spindles having journals running in pedestals with adjusting screws and ample lubricating boxes attached.

The legs are provided with cast-iron plates at their feet, bedding on stone blocks mounted on brick pillars, and secured by large foundation bolts.

For lighter loads and small plants the frames are constructed of I and ⊔ iron sections and are very neat, and strong, and preferable to wood.

When required for shipment the frames are erected, and marked before being taken to pieces, and the necessary bolts provided for locking together in re-erection.

Ropes are now usually made of steel wire of different qualities, all being stronger than iron for the same size; and they may be round or flat. The round are preferable and certainly the most popular. Ropes of hemp are also used, but only to a limited extent, and are gradually falling into disuse for winding.

Wire ropes should be carefully protected with best water-proof grease. The safe working load of ropes may be taken from $\frac{1}{5}$th to $\frac{1}{8}$th of the breaking strain. A close approximation to the safe working load of ordinarily-made wire ropes moving at high speeds is found by multiplying the weight of the rope per fathom in pounds by 5 for iron wire, and by 8 for steel wire, and consider the product as hundred-weights. Thus, an iron wire rope, weighing 16 lbs. a fathom, has a safe working load of $16 \times 5 = 80$ cwt.; a steel wire rope of the same weight, $16 \times 8 = 128$ cwt. These are only rough approximations—rules that can be easily carried in the memory. André, in his *Treatise on Coal Mining*, gives the following rules to find the safe working load:—$C = \sqrt{4\ L}$ for iron wire and $C = \sqrt{2{\cdot}4\ L}$ for steel wire, where $C =$ the circumference of the rope in inches and $L =$ the safe working load, and therefore $L = \frac{C^2}{4}$ for iron wire and $\frac{C^2}{2{\cdot}4}$ for steel wire.

The following rules were at one time used by some rope manufacturers, but the difference in quality of the materials used renders the rule of little practical value. Owing to this difference probably, there is a want of uniformity in the strength of similarly-sized ropes quoted by manufacturers. Let $B =$ breaking weight in tons, and $W =$ weight per fathom in pounds. Then for hemp ropes $W = B$; for iron wire ropes $W = {\cdot}55\ B$; and for steel wire ropes $W = {\cdot}33\ B$. Or, examined, whereas in the other methods the rope at the cap is quite hidden from view.

At some collieries it is customary to fix the cappings without the use of rivets. Where this practice is followed, three or four rings or hoops are used, according to the size of the rope, the weight to be lifted, and the length of the cap. After slipping the hoops up the rope in their proper order according to size, the rope may be prepared in one or two ways. Small wire is tightly bound round the rope

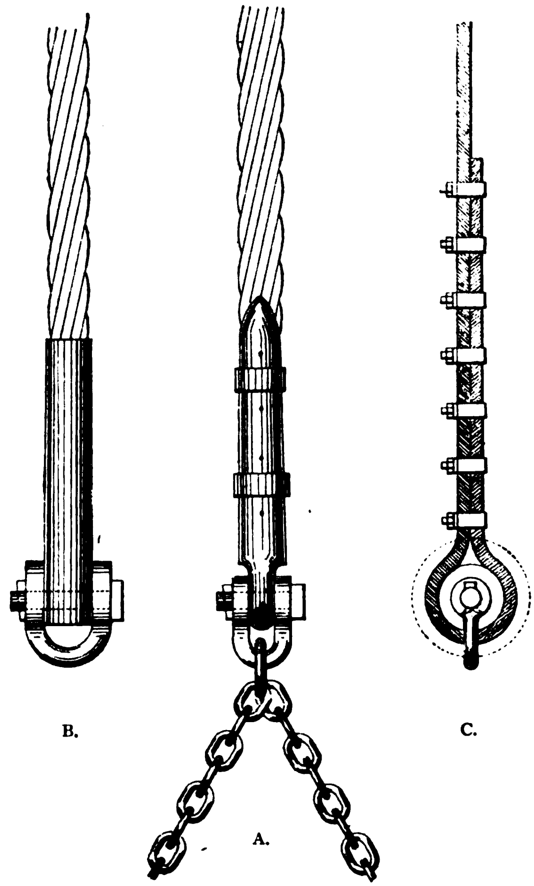


Fig. 68.—Wire-rope Capping.

two feet from the end, the strands are untwisted and the core of the rope removed. The wires are then bent back the opposite way to the strand, care being taken to tuck them under every other strand, and also to cut off some wires between each tuck so as to make the rope taper for receiving the cap; or it may be done after cutting away the core by bending back the whole of the wires around the rope, thinning them out to the required taper. In each case after the tucking or turning back is completed, small wire is wound round outside so as to make the rope as solid and uniformly taper as possible. The cap, having been made hot at the bow, is then laid to the rope and by means of clamps pressed down tightly to it; the hoops are then slipped down over the cap and driven home.

Self-acting incline ropes, and hauling ropes frequently require splicing as a result of breakage. Pit ropes, are, for obvious reasons not spliced.

A common method of repairing a broken rope is by a shackle joint. The method of capping a rope, shown in Fig. 68 (A), is to a large extent followed, but the rings over the joint would be greatly in the way and are therefore not used. A socket with bow having been riveted to one end of the broken rope, the other

Fig. 69.—Repairing Broken Wire-rope.

end is similarly treated and the two ends are joined by means of a link passed through the two loops and carefully closed, see Fig. 69. A rope joined in this way may last a long time, but there is an increased amount of friction caused by the joint passing over the rollers.

The following method of splicing ropes is advised by Mr. Frederick W. Scott, of Reddish, near Stockport:—

"In splicing a wire rope the greatest care should be taken to leave no projecting ends or thick parts in the rope. Heave the two ends taut, with block and

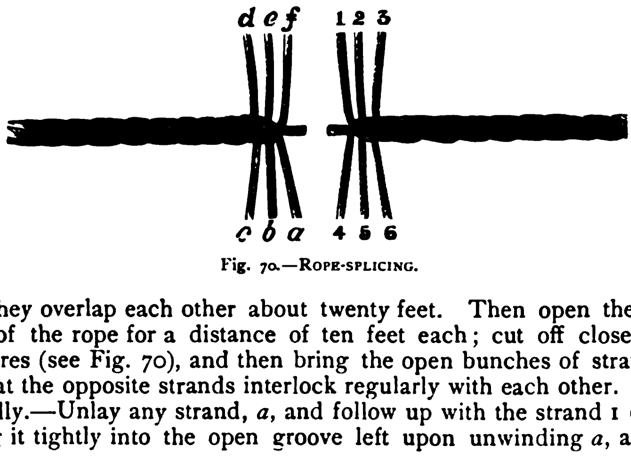


Fig. 70.—Rope-splicing.

fall, until they overlap each other about twenty feet. Then open the strands of both ends of the rope for a distance of ten feet each; cut off closely the main heart or cores (see Fig. 70), and then bring the open bunches of strands face to face, so that the opposite strands interlock regularly with each other.

"Secondly.—Unlay any strand, *a*, and follow up with the strand 1 of the other end, laying it tightly into the open groove left upon unwinding *a*, and making

Fig. 71. Rope-splicing. Fig. 72.

the twist of the strand agree exactly with the lay of the open groove, until all but about six inches of 1 are laid in, and *a* has become twenty feet long. Next cut off *a* within six inches of the rope (see Fig. 71), leaving two short ends, which should be tied temporarily.

"Thirdly.—Unlay a strand, 4, of the opposite end, and follow up with the strand *f*, laying it into the open groove, as before, and treating it precisely as in the first case (see Fig. 72). Next pursue the same course with *b* and 2, stopping, however, within four feet of the first set; next with *e* and 5; also with *c*, 3, and *d*, 6. We now have the strands laid into each other's places, with the respective ends passing each other at points four feet apart, as shown in Fig. 73.

current passes to an ordinary stroke bell. Telephones are used and conversation carried on between the occupants of the cage and the engineman at any portion of the winding.

THE STAGE AND ITS FITTINGS.—The stage should be arranged to suit the particular circumstances of the colliery, but a good plan is to raise it on cast-iron columns 20 to 24 feet above the railway level to allow for proper screening arrangements and for the largest trucks to be loaded under the screens.

Besides screens the staging should carry the weighing machine, tables and cabins. There may also be a small smithy for sharpening picks, &c., and a workshop for repairing tubs if space is available; but none of these erections should in any way interfere with the view of the winding engineman, which must be clear and uninterrupted, so that he can watch the cages as they come up the pit.

The "tippler," "tumbler," or "kick-up" is a contrivance for facilitating the discharge of the coal out of the tub on to the screen; it is placed close to the top of the screen and the tub on being pushed into it, in some cases turns right over and in others sufficiently so to allow the coals to pass gently into the screen. The attendant afterwards easily puts it into its original position and returns to the shaft with the empty tub.

A recently improved patent kick-up works automatically, and may be associated with a self-indicating weighing machine. The tub on entering the tumbler causes the weighing machine to register its weight and directly afterwards turns over, shoots out the coals, and returns to its usual position. The weight of empty tub is then registered and deducted from the previously taken gross weight. The self-righting tippler is so made as to allow the full tub to follow in and push the empty forward as it enters. A vessel is attached to the bottom of the kick-up in which is placed a liquid, the weight of which is sufficient to cause the tub to right itself. It thus gives the kick-up its automatic action. The tub turns over side-ways, not end-ways as is usual.

In placing the upright columns which carry the stage, care should be taken that they rest on a solid foundation of stone and that they are well arranged, and with at least $2\frac{1}{2}$ feet clear space between them and the side of the waggons while being loaded under the screens, so as to avoid accidents. The floor about the railway under the screens may be paved to enable the coal to be swept up unless the pipes, &c., laid, render it inadvisable. If more than one screen is erected to load on the same line there should be sufficient space between them to allow for the largest railway wagons. The length and pitch of the screen will depend on circumstances—the size, quality and freedom from impurities in the coal raised. A very usual pitch is 1 in 2. The sides are usually of wood or iron, the bottoms of cast-iron plates. There should be a slight fall from the shaft to the screens. The screen bars may be of wrought-iron or steel and are usually placed to have from $\frac{1}{4}$ to $1\frac{1}{2}$ inch space between them, according to the circumstances of the colliery. The railways about the screens should be laid at such gradients that empty wagons will, by gravity, quietly move under, and loaded wagons move away from the screens on being started. The greatest difficulty in the matter is, that if the gradients are suitable for good weather, they are certain to be too flat for winter during frost and snow; if they are made to suit winter weather, they are too steep in good weather or during rain, and entail much labour in either pushing, or braking and spragging in these extreme seasons. Some wagons run much better than others, so that an inclination suitable to one may not answer so well for another. It is usual to adopt an inclination of from $\frac{3}{8}$ths of an inch to $\frac{1}{2}$ an inch per yard.

Where greater facilities are required for cleaning and preparing different kinds of coal and of various sizes, such as nuts, beans, peas, small, duff, &c., more elaborate arrangements are provided, which are worked by machinery, such as revolving riddles, moving bands, vibrating screens, and cleaning tables. There should be 6 feet of space between each line of railway, and sufficient siding accommodation for empties and also for loaded wagons for one day's work.

WINDING-ENGINE.—One of the most important of the surface arrangements is the winding-engine, which is vertical, horizontal, single, or double; it may be a beam engine or geared. The horizontal, direct-acting coupled engines are unquestionably the best. A single engine causes delay and is an annoyance when it gets on "centre" and every part of the horizontal engine is more open to inspection by the engineman, than the vertical; is easier cleaned, oiled, and repaired. With 60-foot high pulley frames there should be not less than 20 yards between the centre of the drum and the centre of pit, but with higher pulley-frames the distance must be proportionately greater. The under rope of the drum is subject to more strain than the other because it is bent one way in coiling on the drum and another in passing round the pulley.

A few winding-engines are condensing, but this arrangement is not easy of application, owing to the rapidity of winding and the frequent stoppages and startings. Occasionally winding-engines work expansively but the intermittent working of the engines prevents a more general adoption of this plan, though condensing and expansion both help to economise fuel. Compound engines have in a few instances been applied to wind coal. The Great Western Colliery Co. have erected compound engines to wind coal at one of their shafts near Pontypridd, the steam for which is supplied by Lancashire boilers working at 120 lbs. pressure—a higher steam pressure than formerly prevails, 60 lbs. being a very common steam pressure; and modern Lancashire boilers are made to work at or above 100 lbs. The length of the cylinder is about double its diameter, and is then considered to be well proportioned. The best position for the winding-engine is on a level with the stage top where the tubs are pulled out of the cages, or but slightly above that level as it thus affords the engineman on duty a clear and uninterrupted view of the pit top. It should never be placed below the stage level. The cylinders of the winding-engine should in all cases be behind the drum, so that when the engineman is at the handles, all parts of the machinery are before him as he looks towards the pit top. The drum for a round rope may be either plain, conical or spiral. André gives the following rule for plain cylindrical winding drums. Assuming 10 feet to be the minimum diameter for a wire rope 1 inch in circumference, add 6 inches to the diameter of the drum for every increase of $\frac{1}{4}$ of an inch in the circumference of the rope. Thus a $4\frac{1}{4}$ inch circumference rope will require a drum 10 feet + 6 feet 6 inches = 16 feet 6 inches in diameter. At very great depths there is a disadvantage in having the drum excessively large, for owing to the inertia at the lift the power required is much in excess of that during the latter part of the ascent of the load. This fact has led to the introduction of conical and spiral drums, so as to equalise, at least to some extent, the strain on the engine.

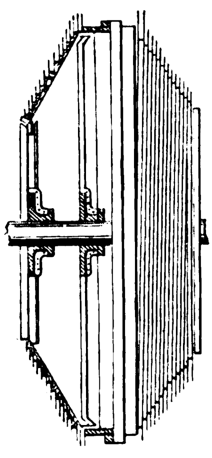


Fig. 75.—SCROLL OR SPIRAL DRUM.

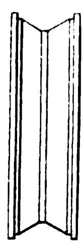


Fig. 74.—CONE DRUM.

In conical and spiral or scroll drums shown in Figs. 74 and 75 respectively, the diameter of the drum is least when the engine lifts the load and increases as the cage advances up the shaft. There is an objection to the cone drum, because the rope is liable to slip on it. A spiral drum should have considerable difference in its diameters at the beginning and end of the winding, and to avoid liability of slipping, the last few coils of the rope in the winding should lap on a plain part of the drum as shown on the sketch given. The spiral drum is the most perfect form of counterbalancing known, but is enormously heavy and costly, and unless for very deep shafts, these disadvantages will outweigh the advantage gained by its counterbalancing effect.

In winding-engines having cylinders of 25 inches in diameter and upwards, Cornish double beat valves, being more easily worked, are preferable to the slide valve, with which, unless it is balanced by a special arrangement, there is much friction, consequently the engine becomes unwieldy, and there is a difficulty in reversing it with the steam on. The Cornish valve may be opened and closed and put into any position with ease without special appliance.

Connected with the crank shaft either directly or by means of gearing is the drum for winding, the eccentrics for working the valves in the steam chest placed alongside the cylinder and, usually, the indicator which shows the engineman the position of the cages in the shaft is worked off the crank shaft itself. The throttle valve is usually placed immediately under the engineman and by means of a handle communicating with the valve, he is able to turn the steam into the engine or to stop its entry at pleasure. Another handle near him communicates with the reversing gear, and a foot brake is generally attached to the drum, but sometimes a steam brake is applied in preference. A stuffing box with gland in the end of the cylinder, prevents the escape of steam there, as the piston rod works through the gland.

A simple form of automatic brake has been devised to prevent overwinding. At a point, which the cage, in its ascent, ought not to reach when under proper control, are placed levers overhanging the shaft. The cage, on striking these levers, puts into action a steam brake which acts directly on the drum.

Fig. 76 represents a pair of high-pressure winding-engines as made by Messrs. Thornewill & Warham, Engineers, of Burton-on-Trent. The cylinders are 26 inches in diameter with a 5-foot stroke, the winding-drum being of cast iron.

The cylinders are bolted and keyed to planed facings on the bed-plates, the covers being fitted with glands and stuffing-boxes, brass-bushed and provided with square-threaded gland-bolts and nuts. The covers are well stiffened and have an ample number of joint-bolts and studs.

Branches are cast on the cylinder to receive the nozzle boxes, each having fitted one steam- and one exhaust-valve, both being Cornish double-beat valves made of gun-metal, with stems of steel or phosphor-bronze, and are fitted with bridles to receive the lifting-cams on the rock-shafts which are operated by the link motion. The reversing motion shown is of the shifting-link type, but the Gooch and Allan link motions are fitted when preferred. The rods and shafts are of best iron, and the pins, links, and dies are of steel; the eccentrics and straps are of best cylinder metal.

The pistons are of cast-iron of box section, and packed with Oldham's or other makers' special rings if required; the piston-rods are of mild steel, the cross-heads of hammered iron with steel gudgeons; the connecting-rods of hammered iron fitted with brasses, straps, gibs and cottars; the cranks of hammered iron with steel pins, and are shrunk when hot and afterwards keyed on the crank-shaft which is preferably of best hammered scrap-iron.

The guide-bars and blocks are of cast-iron, planed and having means for adjustment when worn.

The bed-plate is of hollow box section, well ribbed, and having facings to receive the various mountings.

The main bearings are fitted with heavy gun-metal steps secured by iron caps and bolts.

The winding-indicator consists of a cast-iron open-fronted column containing from the end of its journey. This is a positive indicator, and is superior to any type depending on cords, chains, or clock-faced dials.

The drum has three "centres" to support the wood-lagging, and all are firmly keyed to the crank-shaft.

A brake, having two straps with the necessary connections, is fitted, and the reversing-handle, throttle-valve-handle, and brake-treadle, are arranged conveniently for the engineman.

All the parts are machined, and the necessary portions of the engine polished. The cylinders are clothed with non-conducting composition and mahogany lagging strips secured by polished brass bands.

All holding-down bolts are supplied, and also steam and exhaust-pipe connections to the outside of the engine-house.

The timber and labour for drum and brake-laggings are usually provided by the purchaser, but the contractor turns up the laggings after they have been fitted.

Engines are now being fitted with wrought-iron built drums; being much lighter than cast-iron they enable the engines to start more quickly and work more rapidly.

When automatic expansion is desirable it is usual to fit a tumbler to each steam-valve spindle swinging on a pin kept up to its position by a spring. The tumbler has an arm which comes into contact with a cam, the position of which is regulated by the governor that disengages the tumbler-toe from the steam-lifter on the rock-shaft; the steam-valve and spindle are then free to fall, the rate of descent being controlled by a dash-pot fitted with an adjustable valve.

COUNTERBALANCING.—In all cases of winding coal up vertical shafts with drums of equal diameters, a great disadvantage arises from the fact that the working strain is much greater on the engine at the commencement than at the end of winding.

Frequently, where the shaft is deep, the load at the "lift" is doubled, because the weight of the rope exceeds that of the coals, tubs, and cage, to be drawn up.

Let us suppose a case of a pit 1,530 feet deep in which the weight of the double-decked cage used is 36 cwt., there being one tub carried in each deck, the weight of the bridle chains is 4 cwt., that of each tub 7 cwt., with a carrying capacity of 1 ton.

The total weight for the engine to lift from the bottom would be $36 + 4 + 20 + 20 + 7 + 7 = 94$ cwt. $= 10{,}528$ lbs., + the weight of the rope, which, say, is a round steel wire of 4 inches circumference weighing 8 lbs. per yard or $510 \times 8 = 4{,}080$ lbs. With a 51-foot circumference or 16·2338-foot diameter plain cylindrical drum the number of strokes in making a winding will be $\frac{1{,}530}{51} = 30$.

We may proceed to consider the load on the engine in this case from a statical point of view, omitting all consideration of momentum and inertia which very much complicate the question of counterbalancing.

It will be clearer with the help of a diagram. In Fig. 77, from the zero line OO′ set off horizontally by means of the scale, the load in foot-pounds at the end of each stroke, to ascertain which multiply the load by the leverage of the winding drum, and also set off vertically the number of strokes in the winding.

The full weight at the lift is $(10{,}528 + 4{,}080) \times 8{\cdot}12$ the leverage $= 118{,}617$ foot-pounds. At the end of the first stroke it will be $\{10{,}528 + (493 \times 8)\} \times 8{\cdot}12 = 117{,}512{\cdot}7$ foot-pounds. At the end of the second stroke it will be $\{10{,}528 + (476 \times 8)\} \times 8{\cdot}12 = 116{,}408{\cdot}4$ foot-pounds, and so on, the number of foot-pounds diminishing regularly 1,104·3 at each stroke, so that when the full tub reaches the surface the foot-pounds will be $10{,}528 \times 8{\cdot}12$ the leverage $= 85{,}488$. The line AA′ in Fig. 77 represents this, the full cage-load throughout the winding.

Similarly the weight in foot-pounds of the empty cage rendering assistance in its descent is shown on the diagram Fig. 77 by the line BB′, it being plotted on the opposite side of the zero line to that showing the weight of the full cage. It may be estimated thus:—

requires a separate pit or well about 50 yards deep for the chain to work in. A rope is fixed to the drum-shaft of the engine and to the balance-chain in the small pit. The balance-chain would be 50 yards long, and is so arranged that with one cage at the surface and the other at the shaft bottom the whole length of chain is hanging in the small pit. The rope by which it is wound up allows the whole of the balance-chain to rest upon the small pit bottom when the ascending and descending cages meet in the shaft. The rope passes over the drum-shaft in a contrary direction to the drawing-rope.

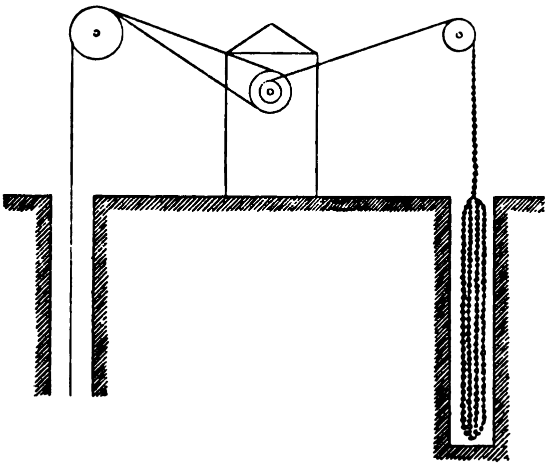


Fig. 79.—Chain Counterbalance.

A fourth method is that of the *inclined plane*, Fig. 80. Usually this is only applicable to shallow shafts, but where the engine-house is situated on high ground, and a gradual slope from it can be obtained, it admits of a long travel for the tub or truck, and so becomes applicable to greater depths of winding. Instead of the bunch of chain used in the last described method, a weighted tub or truck is attached, and this travels over an inclined plane the gradient of which must vary throughout its course, and be steepest at the commencement of the plane next the engine-house. Any attempt to use this kind of counterbalance where the gradient is uniform for the tub or truck to move on is quite useless. The rails laid should form a curve, and not break abruptly from one uniformly inclined short portion to another. At half winding the tub or truck reaches the end of its run, and is wound upwards towards the engine-house as the winding proceeds; on completion of the winding it returns to its starting point.

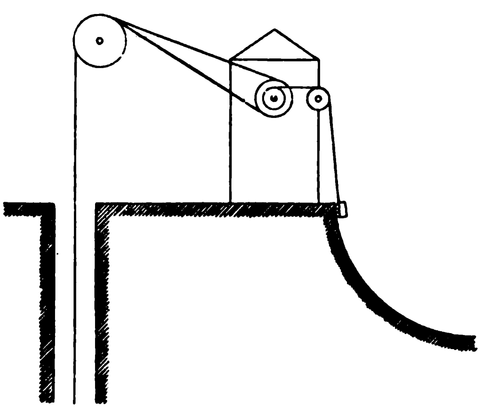


Fig. 80.—Inclined Plane Counterbalance.

Other methods of counterbalancing are by using the cone and scroll drums in preference to plain or cylindrical drums, and these have already been alluded to.

A plain conical drum, to be effective as a counterbalance, requires an angle so great as to be dangerous on account of the liability of the rope to slip. Any advantage to be derived from having a safe angle being trifling, it is really not worth the risk.

Rule 29 of the Mines Act, 1887, renders the use of flanges or horns to the drum compulsory, and if the drum is conical, there must be appliances sufficient to prevent the rope from slipping.

Supposing, in the instance we have been considering, a rope-balance be applied, and that it is of precisely the same weight per yard as the round winding rope, then it is obvious that no calculations are required, for the ropes, cages and tubs must balance each other throughout the wind; the line EF, the actual weight of the coal will be the load on the engine which will be uniform during the ascent of the cage from the shaft bottom to the surface.

The pendulum counterbalance is not applicable to a pit of this depth, we therefore proceed to consider the chain counterbalance in the instance given. Assuming the bunch of chain has to be hung in a 51-yard pit, then as the engine makes 30 revolutions, the rope roll for the counterbalance must have a circumference of $\frac{51 \times 3 \times 2}{30} = 10{\cdot}2$ feet, or a diameter of 3·24675 feet. The leverage then is 1·62338 foot, so that the weight of the bunch must be $\frac{33,129{\cdot}2}{1{\cdot}62338} = 20,407{\cdot}5$ lbs., or 9 tons, 2 cwt., 23 lbs.

When the cages are at meetings, and the whole of the bunch is lying coiled at the bottom of the counterbalance pit, there would not be such relief as would arise from a perfect counterbalancing effect, because there would be the rope in the counterbalance pit, 51 yards and possibly 19 above the pit = 70 yards at, say, 10 lbs. per yard = 700 lbs. × 1·62338 = 1,136 foot-pounds. The effect of this would cause the line DD′ to deviate from a straight one slightly at E, but a very good attempt would be made to produce a counterbalance.

Next: to design an inclined plane counterbalance in the example we are considering, which is, say, 51 yards long and the diameter of counterbalance rope roll 1·62338 foot as before:—Let a weighted truck of, say, 30 tons be used. Then for its effective weight to be similar to that of the chain bunch or 9 tons, 2 cwt. 23 lbs., the inclination for the first stroke or 10·2 feet = $\frac{30 \text{ tons}}{9 \text{ tons, } 2 \text{ cwt., } 23 \text{ lbs.,}}$ = 1 in 3·293. The foot-pounds at the next stroke must be reduced $\frac{1}{15}$th or $\frac{33,129{\cdot}2}{15}$ = 2,208·6. 33,129·2 − 2,208·6 = 30,920·6. $\frac{30,920{\cdot}6}{1{\cdot}62338}$ = 19,047 lbs. The inclination for the next stroke then must be $\frac{30 \times 2,240}{19,047}$ = 1 in 3·528. The foot-pounds must be reduced at the next or 3rd stroke $\frac{2}{15}$ths of 33,129·2 or 4,417·2. 33,129·2 − 4,417·2 = 28,712. $\frac{28,712}{1{\cdot}62338}$ = 17,686·5. The inclination for the remaining strokes will be:—

At the commencement of the 3rd stroke $\frac{30 \times 2,240}{17,686{\cdot}5}$ = 1 in 3·799.

„ „ „ „ 4th „	$\frac{30 \times 2,240}{16,326}$	= 1 in	4·116.
„ „ „ „ 5th „	$\frac{30 \times 2,240}{14,965{\cdot}5}$	= 1 in	4·49.
„ „ „ „ 6th „	$\frac{30 \times 2,240}{13,605}$	= 1 in	4·939.
„ „ „ „ 7th „	$\frac{30 \times 2,240}{12,244{\cdot}5}$	= 1 in	5·488.
„ „ „ „ 8th „	$\frac{30 \times 2,240}{10,884}$	= 1 in	6·174.
„ „ „ „ 9th „	$\frac{30 \times 2,240}{9,523{\cdot}5}$	= 1 in	7·056.
„ „ „ „ 10th „	$\frac{30 \times 2,240}{8,163}$	= 1 in	8·232.
„ „ „ „ 11th „	$\frac{30 \times 2,240}{6,802{\cdot}5}$	= 1 in	9·878.
„ „ „ „ 12th „	$\frac{30 \times 2,240}{5,442}$	= 1 in	12·349.

At the commencement of the 13th stroke $\frac{30 \times 2{,}240}{4{,}081 \cdot 5} = 1$ in 16·464.

" " " " 14th " $\frac{30 \times 2{,}240}{2{,}721} = 1$ in 24·697.

" " " " 15th " $\frac{30 \times 2{,}240}{1{,}360 \cdot 5} = 1$ in 49·393.

" " end " 15th " becoming level.

As the winding proceeds toward completion, the loaded tub changes its direction after the 15th stroke and returns over the inclined plane whose gradients have just been estimated.

Now, to consider the last system of counterbalancing, viz., the spiral drum. If the initial diameter be 16 feet, the weight of the full cage is represented by $(10{,}528 + 4{,}080) \times 8$ the leverage $= 116{,}864$ foot-pounds. The weight of the coal must always remain to be overcome by the engine even in a perfect counterbalance. This would be $20 + 20 = 40$ cwt. of coal $\times 112 \times 8 = 35{,}840$ foot-pounds, and as the empty cage and tubs at the surface weigh 6,048 lbs., the final radius would be $\frac{116{,}864 - 35{,}840}{6{,}048} = 13 \cdot 39$ or a diameter of 26·78, changing regularly for each revolution. The number of revolutions such drum would make $= \frac{1{,}530}{\left(\frac{16 + 26 \cdot 78}{2}\right) \times 3 \cdot 14159} = 22 \cdot 76$ and therefore the diameter must increase regularly $\frac{26 \cdot 78 - 16}{22 \cdot 76} = \cdot 47$ foot at every revolution.

Where a flat rope is used for winding, lapped one coil over the other, it has to a very slight extent, a counterbalancing effect. The difference between the extreme diameters is, however, a negligible quantity.

Miscellaneous.—In laying out the surface arrangements, a considerable amount of thought is necessary to ensure a thoroughly satisfactory and economical working after completion. The particular circumstances and requirements of each colliery must be thoroughly mastered in order to design an effective scheme. The engines and buildings must be adapted to their work, and after the coal is brought to the surface every operation connected with its weighing, screening, cleaning, and after-disposal, should be calculated to give as little manual labour as possible in working, and at the same time yield the various sizes of coal in a good, clean, and marketable condition.

A carelessly laid out bank top renders it necessary to keep employed a large number of workmen to deal with the daily out-put, and thus add to the cost of production. In times of keen competition even a penny extra cost on the tonnage price in raising the coal may place a colliery at a disadvantage as compared with a neighbouring one, and result in loss of contracts; with greatly increased cost prices, it may be impossible to work the colliery at all.

Mr. C. M. Percy, in his excellent work on *The Mechanical Engineering of Collieries*, gives the following rules for winding-engines. (1) To find the load which a given pair of engines will start. Multiply the area of one cylinder by the pressure of steam and twice the length of stroke. Divide this by circumference of drum and deduct $\frac{1}{3}$ for friction, &c. The result is the load the engines can start. For instance, 2 — 20-inch diameter cylinders by 40-inch stroke, with a 12-foot diameter drum and the steam pressure at boilers 50 lbs. $\frac{314 \times 50 \times 80}{454 \text{ in.}} = 2{,}766 - \frac{1}{3}$ or $922 = 1{,}844$ lbs. the load. N.B.—The load referred to in this and the next rule comprises rope and coal, because cages and tubs balance each other. (2) Knowing the load and the diameter of drum and the length of stroke and the pressure of steam, to find the area and diameter of cylinders. Multiply the load by the circumference of the drum and add $\frac{1}{2}$ for friction, &c. Divide this by the steam pressure multiplied by twice the length of stroke, and the result is the area of the cylinder. Example, a drum 15 feet, stroke 6 feet, the steam pressure 60 lbs., and the load 7 tons = 15,680 lbs. $15{,}680 \times 47 = 736{,}960 + \frac{1}{2}$ or $368{,}480 = 1{,}105{,}440$. $\frac{1{,}105{,}440}{60 \times 6 \times 2} = 1{,}535$ square inches, area of piston, and to find the diameter we must divide the area by ·7854 and take the square root of the result, thus $\sqrt{\frac{1{,}535}{\cdot 7854}} = 44\frac{1}{4}$ inches diameter of cylinder. (3) To find the period of a winding approximately. Reckon the piston to travel at an average velocity of 400 feet per minute, and divide this by twice the length of the stroke, and multiply by circumference of drum. This gives speed of cage in feet per minute, and divide depth of pit by this, and the result gives period of a winding. Example, drum 47 feet circumference, stroke 6 feet, depth of pit 600 yards $= \frac{400}{6 \times 2} \times 47 = 1{,}565{\cdot}1$, speed of cages in shaft per minute $\frac{600 \times 3}{1{,}565} = 1{\cdot}15$ minute or about $1\frac{1}{7}$ minutes. (4) To find the *useful* horse-power during a winding. Multiply the depth of pit by weight of coal raised, and divide this by period in minutes occupied in winding, and divide again by 33,000. Example, coal 2 tons = 4,480 lbs., depth 1,500 feet, period of winding $\frac{5}{6}$ths of a minute; $\frac{1{,}500 \times 4{,}480}{\frac{5}{6} \times 33{,}000} = 244$ horse-power. (5) To find the approximate horse-power exerted by a pair of winding-engines during a winding. Multiply area of both cylinders by $\frac{2}{3}$rds of the boiler pressure of steam and by 400, and divide by 33,000. Example, cylinders 30 inches diameter = 706 square inches, boiler pressure 60 lbs., $\frac{2}{3}$rds of which is 40. $\frac{706 \times 2 \times 40 \times 400}{33{,}000} = 685$ horse-power. (6) To ascertain the piston-speed of a winding-engine. Divide the windings into periods of 3 revolutions each. Let two persons take alternately the time in seconds which each 3 revolutions occupy. Take the average from these as follows:—Suppose there are thirty revolutions divided into 10 periods, three revolutions each, the time occupied in seconds comes out 9, 7, 5, 3, 2, 2, 2, 3, 6, 9 = 48 seconds for the whole winding. Suppose the stroke 5 feet × 2 = 10 × 30 = 300 piston feet in 48 seconds and 300 × 60 = 18,000 ÷ 48 = 375 piston feet per minute.

If only the average piston speed throughout the wind is required, there is no useful object attained by dividing the winding into periods of a certain number of revolutions. The whole winding may in that case be timed, and the number of seconds occupied in the wind be divided into twice the length of stroke multiplied by the number of revolutions. Where a comparison of different rates of the varying piston speed throughout a wind is desired, it would, of course, be necessary to sub-divide the total revolutions.

It will be well to bear in mind the following rules for winding-engines:—

$$\text{Horse-power} = \frac{\text{area of cylinder} \times \text{effective pressure per square inch} \times \text{piston speed feet per min.}}{33{,}000}$$

$$\text{Area of cylinder} = \frac{\text{Horse-power} \times 33{,}000}{\text{Effective pressure per square inch} \times \text{piston speed in feet per minute.}}$$

24 cwt., assuming that iron tubs, an iron cage, and an iron wire rope be used. The weight the engine would have to lift would be—

Cage and chains, say	1 ton 4 cwt.
Coal in 4 tubs	1 ,, 10 ,,
4 empty tubs = 4 × 4 cwt.	0 ,, 16 ,,
	3 ,, 10 ,, besides the weight

of rope. The weight of 150 fathoms of iron wire rope at say roughly 14 lbs. per fathom, would be nearly 1 ton, and 1 ton + 3 tons 10 cwt. = $4\frac{1}{2}$ tons as a safe working load for the rope. Circumference = $\sqrt{4 \times 4\frac{1}{2}} = 4\frac{1}{4}$ inches as the circumference of iron wire rope necessary.

Assume a 16-foot diameter drum (Mr. André's rule would give 16 feet 6 inches), and also that the engine has a 5-foot stroke, with a boiler pressure of steam of 60 lbs., the piston speed being 400 feet per minute, $\frac{400}{5 \times 2} = 40$ strokes per minute. $40 \times 16 \times 3{\cdot}14159 = 2{,}010$ feet per minute as the average speed of the cage whilst travelling from bottom to top of pit. $\frac{300 \times 3 \times 60}{2{,}010} =$ nearly 27 seconds. As a trip must be made every $\frac{9}{10}$ths of a minute = 54 seconds, this leaves 54 − 27 = 27 seconds in which to do the changing.

To find the size of cylinder we have the weight of coal and rope at $2\frac{1}{2}$ tons. $\frac{50{\cdot}26 \times 2\frac{1}{2} \times 2{,}240}{5 \times 2 \times 60} = 469{\cdot}09 + 234{\cdot}54$ for friction $= 703{\cdot}63$ and $\sqrt{\frac{703{\cdot}63}{{\cdot}7854}} =$ 29·93, or say 30 inches as the diameter of the cylinders.

Question 4.—What is meant by such expressions in machinery as horse-power, units of work, and back-pressure of the steam?

The English unit of work is the work done in overcoming a resistance of one pound avoirdupois through a space of one foot, and is spoken of as the foot-pound. It is assumed that a horse will raise 33,000 lbs. 1 foot high in 1 minute, therefore we call one horse-power, 33,000 lbs. raised 1 foot high in 1 minute. Back pressure of the steam is the resistance the piston of an engine meets with from the spent steam as it is forced into the exhaust port, and its tendency is to lessen the effect of the engine.

Question 5.—What is meant by the duty of an engine?

The amount of work yielded by that engine, or the number of lbs. raised 1 foot high by the combustion of a given quantity of coals.

Question 6.—Why is a machine put in motion by steam?

Because the pressure of the steam on being applied to the piston of the engine is greater than the combined resistance offered by the weight attached and the friction of the engine itself.

Question 7.—What is steam and how is it obtained? What do we mean by such expressions as high pressure and low pressure steam?

Steam is an invisible elastic fluid, generated from water by the application of heat. It is measured by pressure gauges, and we speak of it as being so many pounds pressure to the square inch. We also speak of steam having a pressure of 15 lbs. to the square inch as one atmospheric pressure; 30 lbs. as two atmospheres. 45 lbs. as three atmospheres, and so on. Steam of two atmospheres and above is called high pressure steam, and below two atmospheres, low pressure steam.

Question 8.—Describe an experiment for ascertaining approximately the relation between the pressure and temperature of steam at a moderate pressure above that of the atmosphere.

This experiment can be made by means of a Marcet's boiler, see Fig. 81. The apparatus consists of a vessel in which is first placed a little mercury, G, and above the mercury a small quantity of water, F; above the surface of the water a space is left for the steam. A piece of glass tube, A, about a yard long, and open at both ends, is passed steam tight through the top of the boiler, and reaches nearly, but not quite, to the bottom, so as to be well into the mercury. It is placed in an upright position. Similarly, a thermometer, C, is passed through the top of the boiler, but its bulb does not quite reach the surface of the water. The glass tube is furnished with a graduated scale, B, placed outside the boiler, by means of which the height of mercury in the tube above that in the boiler can be read off approximately. A cock, D, is also placed outside the boiler, above the surface of the water, so that when it is opened the enclosed air may be driven out through it by the steam. The boiler being placed conveniently for the purpose, a Bunsen flame, K, is applied under it, causing the water to boil and generate steam.

On this steam attaining a greater pressure than the pressure of the external air, the mercury will begin to rise in the tube. By opening the cock for a short time the steam will drive out the enclosed air from the boiler, and the cock may be shut again. As the pressure of the steam above the atmosphere increases, it will be noticed that the mercury rises in the tube, and at the same time that the temperature of the steam also rises above 212° F., as seen on the thermometer.

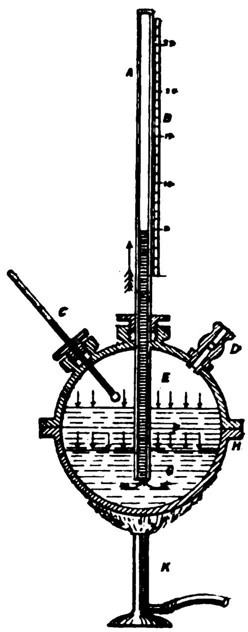


Fig. 81.—Marcet's Boiler.

The cock must now be carefully adjusted so as to allow the exact quantity of steam being generated in a certain time to pass through it to the outer air, in the same time.

The temperature and the pressure corresponding to that temperature will then be constant, and may be read off and recorded from their respective scales.

By closing the cock again, the temperature may be allowed to rise, say, 10°, upon reaching which the cock is again adjusted as before, and the temperature and pressure again noted. This operation may be repeated for any reasonable number of times, and it will be seen that for a given temperature there is a corresponding pressure, or vice versâ.

When the temperature is 233° F. the corresponding pressure will be 7·4 lbs., half an atmosphere, or 22 lbs. absolute, for the mercury will have risen to 15 inches.

At a temperature of 250° F. the mercury will have risen to about 30 inches, corresponding to a pressure of 14·7 lbs. on the square inch, or 1 atmosphere, or 29·4 lbs. absolute.

Question 12.—Does the pressure on the piston increase or decrease during expansion? Explain the effect of using steam expansively? How is the mean pressure of the steam throughout the stroke ascertained?

The pressure of steam decreases on the piston during expansion. Mariotte's law with regard to the expansion of steam is as follows:—If a given weight of steam be made to vary its volume without changing its temperature, the elastic force of the steam will vary in the inverse ratio of the volume it is made to occupy.

Although there is a great economy in steam, and consequently in fuel, when using steam expansively, there is not actually more work done in a steam-engine working expansively, than where the same pressure is allowed to remain throughout the stroke. The advantage gained by expansion is the fact of getting more work out of a given quantity of steam. This may be explained by working out examples. The following is from *The Steam Engine*, by T. Baker. The rule to find the work done by expansion without the use of logarithms is this. Divide that part of the stroke through which expansion takes place into any even number of equal parts, and calculate the pressure per square inch upon the piston at each division of the stroke by Mariotte's law; take the sum of the extreme pressure in pounds per square inch, four times the sum of the even pressures, and twice the sum of the odd pressures; multiply the sum of all these by one-third of the common distance between the positions of the piston and the result will be the work done upon each square inch of the piston after expansion begins. The work done before expansion begins, being evidently equal to the pressure per square inch multiplied by the number of feet moved before expansion; and the whole work done during a single stroke is equal to the sum of the work done before and after expansion.

Example.—The pressure of steam upon the piston is 60 lbs. per square inch, the resistance arising from imperfect condensation is 4 lbs. per square inch, the length of the stroke is 12 feet, and the steam is cut off at $\frac{1}{6}$th of the stroke; it is required to determine the number of units of work done upon each square inch of the piston, the number of units of work gained by working expansively, and the load per square inch.

By dividing the remaining portion of the stroke after the steam is cut off, viz., $12 - 2 = 10$ feet into 10 equal parts, each will be a foot. Let the pressures at these different divisions be represented by p, p_1, p_2, p_3, &c., and then by Mariotte's law already given.

$$\begin{aligned}
\text{As } 3:2::60:p_1 &= 40\\
4:2::60:p_2 &= 30\\
5:2::60:p_3 &= 24\\
6:2::60:p_4 &= 20\\
7:2::60:p_5 &= 17\cdot142\\
8:2::60:p_6 &= 15\\
9:2::60:p_7 &= 13\cdot333\\
10:2::60:p_8 &= 12\\
11:2::60:p_9 &= 10\cdot909\\
12:2::60:p_{10} &= 10
\end{aligned}$$

Then $60 + 10 = 70$, the sum of extreme pressures.

$$\begin{array}{r}
40\\
24\\
17\cdot142\\
13\cdot333\\
10\cdot909\\
\hline
105\cdot384\\
4\\
\hline
421\cdot536
\end{array}
\begin{array}{l}
\\ \\ \\ \\ \\
= \text{the sum of the even pressures.}\\
\\
= \text{4 times the sum of the even pressures.}
\end{array}$$

taining the relationship subsisting between its temperature and absolute pressure. The more simple of these are only applicable, and that approximately, to certain ranges of change.

Tredgold's formula for pressures from 1 to 4 atmospheres is $p = \left(\frac{103 + t}{201 \cdot 18}\right)^6$. Applying this first to the pressure at 212° we shall find that it complies with the 14·7 lbs. already stated, thus $p = \left(\frac{103 + 212}{201 \cdot 18}\right)^6 = 14 \cdot 73$; and for the pressure at 225° we have $p = \left(\frac{103 + 225}{201 \cdot 18}\right)^6 = 18 \cdot 7816$ lbs., showing as stated an increased pressure.

From the formula to find the relative volumes when both temperature and pressure change at the same time $u = 1{,}700 \times \frac{p_1}{p} \times \frac{1 + \cdot 00203\,(t - 32)}{1 + \cdot 00203\,(t_1 - 32)}$ and substituting the known values in the formula $u = 1{,}700 \times \frac{14 \cdot 7}{18 \cdot 7816} \times \frac{1 + \cdot 00203\,(225 - 32)}{1 + \cdot 00203\,(212 - 32)}$, but as already worked out the last item in the above was found to equal 1·019326, it will render the work easier to substitute it, thus $u = 1{,}700 \times \frac{14 \cdot 7}{18 \cdot 7816} \times 1 \cdot 019326$ and therefore $u = 1{,}359$ which is the volume 1,700 cubic inches at a pressure of 14·7 lbs. and at 212° F. will occupy at a pressure of 18·7816 lbs. and at 225° F.

> *Question* 14.—What weight would a pair of 22-inch cylinder horizontal engines with a $4\frac{1}{2}$-foot stroke and an 8-foot cylindrical drum on the first motion raise from a pit 260 yards deep with a round wire-rope, the boiler pressure being 40 lbs. per square inch? The engine works expansively, the steam being cut off at $\frac{3}{4}$ stroke.

With a boiler pressure of 40 lbs. the initial pressure of the steam in the cylinder would be, say $\frac{2}{3}$rds of this 27 lbs

To find the pressure at the end of the stroke or at any point during expansion proceed by the following formula:—

P = Initial pressure of steam in pounds per square inch including the pressure of the atmosphere.

l = Distance travelled by the piston before steam is cut off.

L = Distance travelled by the piston when the pressure of the steam = X.

X = Pressure of steam in the cylinder including the pressure of the atmosphere, when the piston has travelled a distance L.

$$X = \frac{Pl}{L}.$$

$$\therefore X = \frac{(27 + 15) \times 3}{4} = 31\tfrac{1}{2}$$ from which deduct 15, the atmospheric pressure $= 16\frac{1}{2}$.

The average pressure throughout the stroke then is $\frac{27 + 27 + 27 + 16\frac{1}{2}}{4} = 25$ nearly. For an average pressure of 25 lbs. on the piston throughout the stroke take a boiler pressure of $25 \times \frac{3}{2}$ = nearly 38 lbs.

To find the load these engines will lift from the stated depth, and the other particulars as given. The circumference of the drum is $8 \times 3 \cdot 1416 = 25 \cdot 1328$ feet, and the area of a 22-inch cylinder is $22^2 \times \cdot 7854 = 380 \cdot 133$. Therefore $\frac{380 \cdot 133 \times 38 \text{ lbs.} \times 108}{25 \cdot 1328 \times 12} = 5{,}172$ from which must be deducted $\frac{1}{3}$rd for friction 5,172 − 1,724 = 3,448 lbs. and this load which the engines will lift comprises the rope and coal. The cages and tubs balance each other and need not be considered. The round wire rope used must have a safe working load then of 3,448 lbs. plus the weight of cage and tubs, say for an iron wire rope a safe working load of 57 cwt. and a $3\frac{3}{8}$-inch circumference rope will be required weighing $9\frac{1}{2}$ lbs. per fathom for the winding. In a pit 260 yards or 130 fathoms deep the total weight of such rope would be $130 \times 9\frac{1}{2} = 1{,}235$ lbs., and therefore the actual weight of coal these engines will lift is $3{,}448 - 1{,}235 = 2{,}213$ lbs. or 19 cwts. 3 qrs. 1 lb., or rather under a ton.

Question 15.—What is meant by initial, mean, and terminal pressure of the steam?

Initial is the full pressure of steam per square inch acting on the piston over a portion of its stroke previous to the closing of the steam valve. Mean pressure of steam expresses the average pressure of steam throughout the stroke, an example of how to get which is given in the answer to Question 12. The terminal pressure is that acting on the piston at the close of its stroke. Thus if steam enters the cylinder of a non-condensing engine with a 4-foot stroke at a pressure of 60 lbs. to the square inch and the steam is cut off at a quarter stroke, the absolute pressure or the pressure including that of the atmosphere is 60 lbs. + 15 lbs. = 75 lbs. By the formula in Answer 14, the terminal pressure is $\frac{75 \times 1}{4} = 18{\cdot}75$ absolute $\therefore 18{\cdot}75 - 15 = 3{\cdot}75$ lbs. as the terminal pressure shown by the pressure-gauge. As the engine is non-condensing it is necessary to deduct 15 the atmospheric pressure from the 18·75.

Question 16.—What provision should be made for letting the condensed water out of horizontal winding engine cylinders? and is the condensed water liable to accumulate elsewhere?

A small pipe with drain-cock should lead from either end of the cylinder and from the under side of it. There will always be a little condensation after the engine has stood some time on admitting the steam to the cylinders. Sometimes these are covered with wood and sometimes with cement, or they may be steam-jacketed as a means of retaining their heat. The steam pipes leading to the cylinders of the engine, if long, present a large cooling surface causing condensation and therefore should also have a separator and steam trap in the bend at the lowest point to allow for the escape of the condensed vapour.

Question 17.—Where would you place the brake for a winding engine and which kind do you prefer?

I prefer a well designed foot brake for small or ordinary sized engines, because a steam brake requires a good deal of fitting up, and when used comes into action very abruptly and suddenly causing some shock to the machinery, and a good foot-brake meets all the requirements, comes gently into operation, and can be gradually or firmly applied at pleasure. In powerful engines with heavy drums of large size subjected to rapid winding, the momentum acquired by the drums renders a greater brake power necessary, and where these are used, a steam brake should be adopted, in order that the engine-man may have greater control over the engines. When the brake is applied to the drum, a ring is formed there which may have either one or two iron straps lined with wood or hemp rope forming the brake, and care should be taken that when the brake is "off" these straps clear the drum ring. The reason for placing the brake on the drum in preference to the fly wheel is the fact that in case of accident to any part of the machinery the engine-man could at once apply the brake on the drum, thus pre-

Question 24.—Describe any system or systems of winding coal up shafts in which the use of the drum on the engine is dispensed with.

The use of the drum is dispensed with in Craven's improved winding gear, and also in the Kœpe system of winding. The following description of the former method appeared in the *Colliery Guardian* of June 9th, 1882:—

"Important improvements have lately been introduced by Mr. John Craven, of Wakefield, in the form and arrangement of winding gear for mines, for the invention of which letters patent have been granted. The improvements consist mainly in obviating the necessity for coiling ropes round the drums usually employed, and so removing the danger and expense arising from the great wear and frequent injury of the ropes by one coil chafing against the other. In order to effect these objects, the inventor employs a single winding rope, an upper set of grooved headgear pulleys, and a lower set of grooved winding pulleys, each consisting of two pulleys, and an intermediate grooved pulley between the two sets. The rope is attached at one end to one of the cages, passes over one of the headgear pulleys, under one of the winding pulleys (to which the motive power is applied), and back over the intermediate pulley, and then under the other winding pulley, and thence over the other headgear to the other cage, to which the end of the rope is attached. This arrangement is designed to give greater durability to the rope, and to obviate all tendency to slipping of the rope, as in proportion as the weight of the load is increased the adhesion of the rope is augmented. The bearings of the intermediate pulley may be carried in a movable frame, either inclined or otherwise, so as to admit of the pulley being adjusted as required, in order to maintain the rope taut.

"Fig. 89 represents in elevation and Fig. 90 in plan a winding gear constructed and arranged after the method above described. A set of two-grooved headgear pulleys is mounted in bearings in the headgear or framework, and another set of two-grooved winding pulleys is keyed on the crank shaft of the engines in the engine-house. The headgear pulleys are preferably set at an inclination inwards towards the winding pulleys in the engine-house, as shown in the plan. The intermediate grooved pulley is mounted in bearings carried by the framework. The single winding rope is attached at one end to one of the two cages and passes over one of the headgear pulleys, thence under one of the winding pulleys in the engine-house, and then back and over the intermediate pulley, and then under the other winding pulley in the engine-house, and to and over the other headgear pulley to the other cage, to which the end of the rope is attached. The winding engine is represented in the engraving, and its power is applied to the crank shaft of the engines, on which the winding pulleys are mounted, so as to drive the pulleys in the one or the other direction for raising the one cage and lowering the other by the one rope. The bearings of the intermediate pulley are in blocks which are capable of siiding upon guides carried by the framework. To these blocks are attached the rods represented in the elevation, the other ends of which are screwed and pass through lugs, and are provided with screw-nuts. By screwing up or slackening these nuts the bearings can be moved in one or the other direction to maintain the rope taut, or to slacken or tighten it as required. The following specific advantages are claimed for the invention:—No chafing of rope as in the ordinary system of drum, so that the ropes last longer; reduction of work for the engine to do in starting; reduction of strain upon the engine, &c., in stopping; speed, instead of being obtained by a large diameter of drum, is got from the engine running quickly; a smaller engine is required than with ordinary gear, owing to the comparative lightness of the winding pulley; the winding pulleys have only one groove each, and are of very small weight comparatively; saving in first cost, the engine-house being much narrower, and only one rope required instead of two; great adhesion, no slip occurring between the rope and the pulleys."

Mr. Percy, in his *Mechanical Engineering of Collieries*, describes the Kœpe system as follows:—

"The Kœpe system of colliery winding dispenses altogether with winding drums and substitutes a pulley. One winding rope answers for and is attached to both cages, instead of having a separate rope for each. This rope having a cage at each end simply passes about half round an ordinarily-constructed **V** pulley

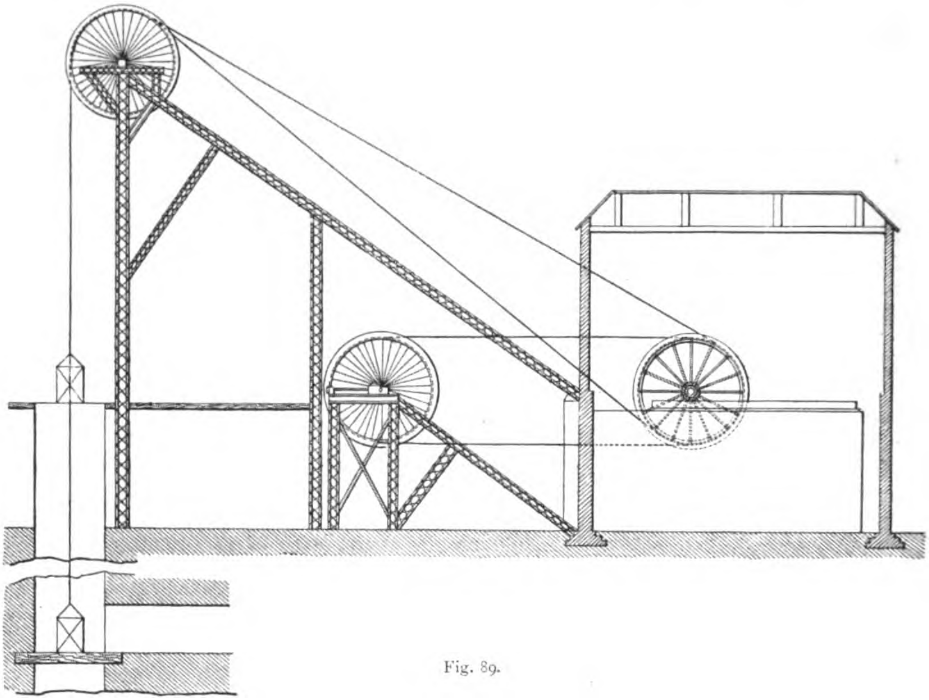


Fig. 89.

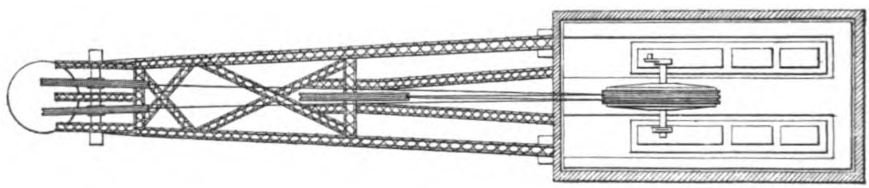


Fig. 90.

CRAVEN'S WINDING GEAR.

on the crank shaft. The balance rope under the cages is used in connection with the Kœpe system. The advantages claimed for the arrangement are manifold. First, massive winding drums are abolished, thus avoiding the enormous weight to start and stop each winding. Second, the pair of engines can be brought closer together, thus making the crank shaft shorter. Third, a smaller engine-house is required. Fourth, only one rope is actually used for winding. Fifth, the rope always coils round exactly the same diameter. Sixth, the rope always works in the same line. Seventh, the load is exactly uniform throughout the winding. Eighth, a smaller pair of engines are equal to the work. Objections have been taken to the arrangement that the rope is liable to slip upon the drum-end. The object of the internal flue is to give a greater heating surface than that of the egg-ended boiler.

Fig. 92 shows a sketch of this boiler.

The products of combustion pass from the fire-grate through the internal tube

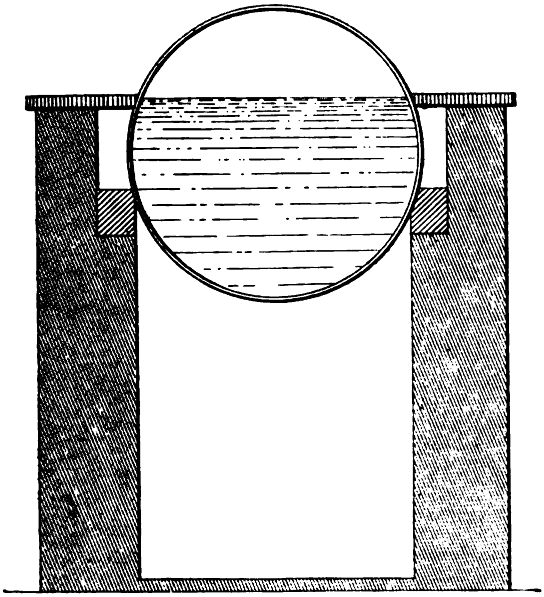


Fig. 91.—Common Cylindrical Boiler.

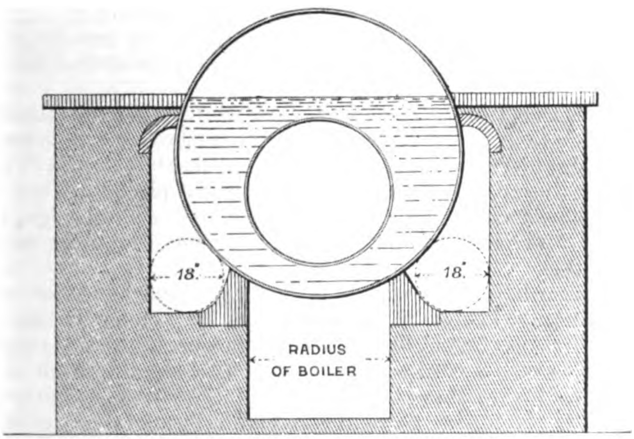


Fig. 92.—Cornish Boiler.

to the back end of the boiler, then, dividing, they return to the front end along the two side-flues. Here they pass down to the bottom flue, and, re-uniting, pass underneath the boiler to the chimney.

Where the flues are traversed thus, the gases are reduced in temperature before coming in contact with the bottom of the boiler, where all sediment collects, and all danger of burned plates on the under side of the boiler is avoided. Where *zag* riveting, as shown in Fig. 98. Where the two rows of rivets are placed opposite each other, it is called *chain* riveting, as shown in Fig. 99. Zig-zag riveting requires less lap than chain riveting, besides making a tighter joint, but the plates are not so strong. The chain-riveted makes the stronger joint, and is coming more into use.

There are some advantages in punching the holes in the plates, and some in drilling them. The strongest argument against punching plates lies in the fact that the plates are punched when flat, and afterwards bent to shape, whereas the drilling is done after the plates have been bent.

A plate with punched holes receives damage along the row of holes in bending. To some extent, also, punching injures the texture of the metal immediately surrounding the hole. Again, where the punching is carelessly performed the holes in the plates do not correspond. What is called *drifting* is then resorted

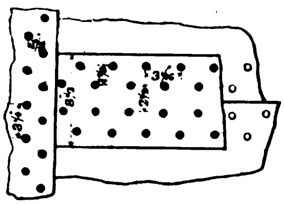


Fig. 98.—Zig-zag Riveting.

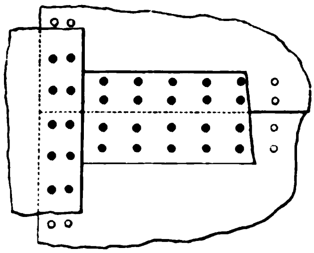


Fig. 99.—Chain Riveting.

Fig. 100.—Punched Holes.

to. It consists of bringing the two plates into good alignment for receiving the rivets by either a pointed or barrelled drift. Where the contiguous holes of two plates are so much out of correspondence as to be nearly blind, the pointed drift is used.

The practice of drifting is most objectionable, especially with the pointed drift, as the plates are injured by it. Where the holes do not quite coincide they should be drilled true. It is quite true that the plates when drilled separately present the same difficulty with respect to a want of agreement between the rows of holes.

Drilling machines are now in use whereby the plates are drilled when fixed in position, thus ensuring an absolute coincidence of the holes in the different plates.

The holes formed by drilling are of a uniform size throughout the plates, but punched holes are slightly tapered, as shown in Fig. 100. When the plates are placed together, the small ends of the holes are placed inside the joint, and the larger ends outside. There is in this a decided advantage in the punched holes, as when the rivets are driven in they hold the plates more firmly, and make a tighter joint than with the parallel holes of drilled plates.

The edges of drilled holes are sharp, and exercise a cutting action on the rivet; if slightly counter-sunk the strength of the rivets is increased, but this adds to the expense. No cutting action takes place at the outer edge of a punched hole.

From experiment it seems well established that where the plates are punched the rivets are stronger, but the plates are weakened to a greater degree; and altogether joints made with drilled holes are rather stronger than joints made with punched holes.

The rivets should be of the best Lowmoor iron or mild steel, and are heated in a common portable hearth before being hammered down. Greater care is required with steel rivets than with iron, as steel is a material which suffers even greater injury than iron from being over-heated. As the rivets cool they contract and draw the plates closer together. Formerly, all the joints were riveted by hand, but now wherever a machine can be applied it is used to do the riveting. Besides doing the work more quickly, good machine riveting is superior in strength to hand riveting. The hydraulic riveting machine is mostly used for the purpose. In machine riveting the pressure on the whole body of the rivet is more gradual than that resulting from the sharp, sudden blows of a hammer in hand riveting. This gradual pressure, resulting from powerful squeezes, forces the rivet into the hole, making its filling a matter of certainty before forming a head at all, and the joint is, therefore, more secure. The act of riveting with a mechanical riveter is momentary, and the point of the rivet is not rendered brittle by being repeatedly hammered when at a low heat or cold. Care should be taken to have the plates drawn closely together before riveting, or the compression of the body of the rivet into the hole may cause a slight shoulder to be formed between the plates, which will prevent the closing of the joint.

When the riveting of the boiler is completed, the joints should all be carefully caulked, so that they may be absolutely steam- and water-tight.

The usual practice is what is called split caulking. Fig. 101 shows this method as applied to a lap-joint, and Fig. 102 as applied to a butt-joint.

By means of a tool something like a chisel a score or split is cut as shown in the figures. This brings the extreme edge of the lap into close contact for about $\frac{1}{8}$th of an inch, but at the same time is objectionable, as in the case of lap-joints it is liable to open the plates between the extreme edge and the point where they are held tightly together by the rivets. For this reason, many makers have now largely given up split caulking.

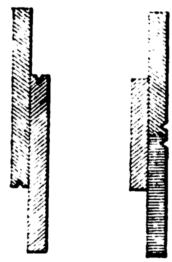


Fig. 101. Fig 102.

SPLIT CAULKING.

The best practice consists in planing the edges with a slight bevel, and then by means of a proper caulking tool the surfaces are driven into close contact without injuring the plates.

A boiler shell consists of rings formed of plates from three to four feet six inches wide, rolled with the grain running circumferentially. Each ring is usually composed of two or three plates. Steel plates are now rolled large enough to admit of one plate forming an entire ring, so that there is only one longitudinal seam in a boiler made of such plates.

Within the last few years boilers have been made with welded joints in the shells. If the soundness of these joints were assured, a boiler made of complete rings would possess advantages over one built in the ordinary manner, in having fewer joints, which, besides being the weakest parts of a boiler, are the places where leakage most frequently occurs, and external corrosion arising therefrom. The soundness of welded joints is, however, still uncertain, and for this reason boilers built with these ring plates have not become popular.

The plates in the rings are connected to each other by lap- or butt-joints, as are also the rings to each other.

The circular seams form a continuous line round the boiler, but the horizontal or longitudinal seams are not continuous, the joint in each ring being intermediate to joints in the adjoining ring.

The flat end plates are each in one piece, the portion to receive the internal flues being bored out of each plate, and are connected to the shell in different ways.

Fig. 103 shows the usual method of attaching the front plate to the shell and to the internal tube, and Fig. 104 the method of joining the back plate to the outer shell and internal tube. In Fig. 103 a ring of angle iron is placed outside the shell, and riveted to the shell and to the front plate. The internal tube is usually similarly dealt with at both the front and back ends, having a ring of angle iron placed outside it at either end, the angle iron being riveted to the tube and to the end plate as shown in Figs. 103 and 104.

Occasionally both the front and back plates are attached to the shell by inside angle iron, but usually the back end plate is flanged and then riveted to the shell plates by an ordinary lap-joint, as shown in Fig. 104.

Sometimes the internal flue is attached to the end plates by flanging the end plates inwards or outwards.

The longitudinal seams of internal boiler tubes are either welded or butt-jointed. The reason why no lap-joints are made in the internal flue is that any departure from a truly circular section gives less resistance to collapse. The cylindrical shell of a lap-jointed boiler is not perfect in form, but the *internal* pressure to which it is subjected has a tendency to rectify the defect and to bring

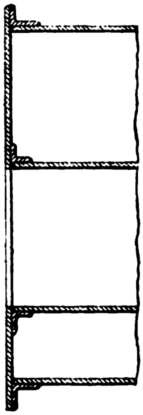


Fig. 103.—Attachment of Front-end Plate.

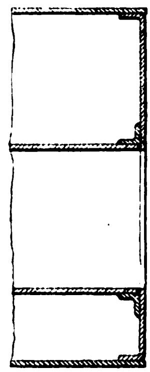


Fig. 104.—Attachment of Back-end Plate.

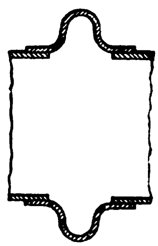


Fig. 105.—Bowling Hoop Expansion Joint.

the shell to the form of an exact cylinder. The effect of *external* pressure on an imperfectly cylindrical tube, however, increases the defect and causes a greater departure from the true circular section. By welding the longitudinal seams or by butt-jointing them, the truly cylindrical form is approached as nearly as practice allows.

In addition, the internal tubes require strengthening and also to have some provision for expansion and contraction, as they are subjected to great heat and sudden admissions of cold air. It is customary now to secure the different rings forming the flue to each other by an expansion joint. At first a ring of T iron was used for strengthening the tubes. It was riveted round the joints of each ring of plates, and although found to give sufficient strength it held the flue too rigidly and did not allow free expansion and contraction to take place.

Fig. 105 shows the expansion or bowling hoop, which has been much used for flue joints. It is weldless, and can be made in iron or steel. It is as strong as the T iron ring, with the advantage of allowing free expansion of the tube. The objection to it is that it exposes two rows of rivets and a double thickness of plates to the intense heat of the furnace, and these are therefore liable to be burned.

Another form of expansion joint is known as Adamson's flanged seam, and is shown in Fig. 106.

In it the ends of the flue plates are flanged and connected by means of rivets with a ring placed between. The object of the ring is to give a caulking edge on each side of the lap. This joint is very elastic, and allows free expansion and contraction to take place.

By its use no double thicknesses of plate and no rivets are exposed to the action of the fire. All plates which require flanging must, however, be of excellent quality, and even then, if not skilfully done, the joint gives a considerable amount of trouble.

Fig. 107 shows another method of strengthening flues and of allowing for their expansion and contraction, viz. Paxman's flue joint. It consists of welded rings of iron or steel, which are rolled out accurately in a machine to the shape shown on sketch, the connection being made by a simple lap-joint. This joint allows for expansion. The rivet heads and double thicknesses of plate, although not removed from the action of the fire, are out of immediate contact with it.

Foxe's corrugated furnace flues are stronger than the plain flue fitted with any of the strengthening rings mentioned, and yet their shape allows every facility for expansion, whilst giving greater heating surface than the ordinary flue. An

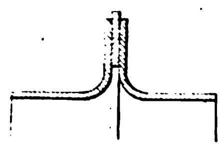


Fig. 106.—Adamson's Flanged Seam.

Fig. 107.—Paxman's Flue Joint.

objection to their use, however, lies in the facility offered for sediment and salt incrustation to gather and form in the hollows of the corrugations at the top of the flue, whilst at the bottom the corresponding hollows under the fire are filled with dead ashes, not easy to remove.

The method of fitting Galloway tubes in the flues consists in making a hole in the upper side of the flue of such a size as to allow the flange of the small end of the tube to be passed through. The hole cut at the bottom of the flue is the net size of the bottom of the Galloway tube. A row of rivets is then driven all round the flanges through the flue. The lower flange of the Galloway tube is inside and the upper one outside of the flue, as shown at H, in Fig. 131. Sometimes the Galloway tubes are welded into the flues, and this is very effectual in preventing leakages at the joints. An objection to this plan is the large hole which must afterwards be cut in the flue, if it is found necessary to remove the tube, as the welded part must be cut away with it.

As shown in Fig. 131, the last two rings of an internal tube in a Lancashire boiler are rather smaller than the others, the last but one being tapered in form, the diameter at the small end being some six inches less than at the larger end.

The egg-ended boiler, as before stated, requires no stays whatever, owing to its shape at the ends. The pressure acting on flat surfaces, however, causes those surfaces to bulge out. All flat surfaces in boilers therefore require to be stayed.

In the Cornish and Lancashire boilers the only flat surfaces are the ends, and these require to be stayed with gusset stays, or with longitudinal stays passing from end to end of the boiler, or it may be with both.

Gusset stays are usually made of a single plate of iron, and this is fixed to the end plate and to the shell by means of angle irons on each side of the strengthening plate, as shown in Fig. 131 at E. The rows of rivets at the front end gusset stays may also be seen in Fig. 122.

There are usually five gusset stays over the flue and two under at each end.

Longitudinal stays are rods of iron or steel secured at the ends by nuts and washers. One is shown at R in Fig. 131. When these stays exceed 20 feet in length they have a tendency to droop in the centre, and must, therefore, be supported there by small brackets riveted to the shell.

In staying the ends, the object desired is to strengthen the plates and yet preserve a certain amount of elasticity. If made absolutely rigid the flues would not have sufficient freedom to expand.

The following table, taken from Sir John Anderson's *Strength of Materials*, enables us to compare the relative strengths of the different forms of riveted joint.

The strength of the solid plate is taken at 100.

RIVETED JOINTS.

Description of Joint.	Riveting.	Rivet Holes.	Percentage of Strength of the Solid Plate possessed by the Joint.
Lap	Single	Punched Drilled	55 62
Lap	Double	Punched Drilled	69 75
Butt, 1 Cover	Single	Punched Drilled	55 62
Butt, 1 Cover	Double	Punched Drilled	69 75
Butt, 2 Covers	Single	Punched Drilled	57 67
Butt, 2 Covers	Double	Punched Drilled	72 79

From this table it appears that the single-riveted lap-joint is the weakest, that butt-joints with one cover are no stronger than lap-joints, but with two covers the percentage of plate strength is a little more than lap-joints.

In the construction of boiler shells it has been proposed to substitute diagonal seams for the longitudinal, but an objection to this is the waste of plates resulting from cutting the ends to the chosen angle. Doubtless, a cylindrical boiler constructed with seams in this manner would be stronger than with ordinary longitudinal seams. A cylinder is twice as strong transversely as longitudinally.

In any boiler shell having the circular and longitudinal seams similarly riveted the former has twice the resisting power of the latter. A single-riveted joint, unless badly designed and clumsily made, is rather more than half as strong as the solid plate. A single-riveted ring seam is, therefore, calculated to resist pressure equally as well as the solid plate of the same thickness in the longitudinal section. In whatever way the longitudinal seams are riveted they cannot be made as strong as the solid plate, and consequently the strength of the flue space formed by the brickwork is limited in extent and faulty in design. The seatings are built of ordinary fire-brickwork, are a foot broad, while the same kind of work is in contact with the boiler above the crowns of the side flues for 18 inches on each side.

The effect of so much brickwork in contact with the boiler renders the latter more difficult of examination, and being porous, it is calculated to absorb moisture and keep out of sight corrosion at those portions of the boiler shell covered by the brickwork.

Besides these objections to such masses of brickwork, there is the further one that the seatings formed in Fig. 108 deprive the boiler of a considerable amount of heating surface. In a boiler 28 feet long, the heating surface lost at the seatings would be $(1 \text{ ft.} + 1 \text{ ft.}) - (3'' + 3'') \times 28 = 42$ square feet as compared with Fig. 92. Allowing 10 square feet of heating surface per nominal horse-power the deprivation of heating surface results in a loss of 4·2 nominal horse-power in a boiler of the length given as compared with Fig. 92, and would be still more in one of greater length.

Again, the side flues of Fig. 92 being of ample size and suitable shape, offer no impediment to a full and searching examination when the boiler is not at work, and at other times ensures an efficient draught. The deposits of soot formed on

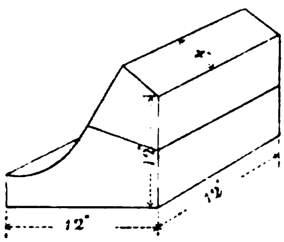


Fig. 109.—Seating Block.

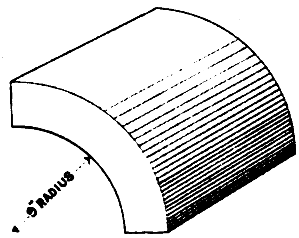


Fig. 110.—Closing-in Tile.

the sides of the shell will fall to the floor, which being level or but slightly below the level of the lowest point of the boiler shell, allows of a certain amount of accumulation without greatly interfering with the draught, whereas in Fig. 108 thick deposits of soot still further reduce the capacity of the flues, and necessitate more frequent cleaning out than in Fig. 92.

To ensure a sufficient depth in the side flues further attention must be given to the form of seating block used. It is not sufficient that this presents a small bearing surface for the boiler to rest on, for if made shallow and used for boilers of comparatively small diameter, the side flues formed as a result will be restricted in area. The seating blocks then must be of sufficient depth, as well as made to give narrow bearing surfaces for the boiler to rest on.

Fig. 109 shows a seating block suitable for a large boiler. Its inner vertical side next the boiler is 12 inches deep, by 12 inches wide at the bottom, and 3 inches thick in its outer edge, presenting a bearing surface of 4 inches at the portion in contact with the boiler plates. Opposite the ring seams, these blocks should be made in two parts as shown in Fig. 109, and the upper part left loose. This will facilitate examination for leakages at the seams of that portion of the shell resting on the blocks, as the loose portions of the blocks may be taken out, and after the examination is completed, re-placed.

The use of lime-mortar in any brickwork or blocks, actually in contact with the plates must be carefully avoided. The effect of using it in those positions is to corrode the iron, and therefore ground fire-clay is substituted to cement the fire-bricks and blocks in setting steam boilers. Fire-clay of good quality, such as that of Stourbridge, stands heat and sets well, and it has no injurious effect on boiler plates.

The effect of damp brickwork in contact with the boiler plates is to cause damage to the plates by external corrosion. It is quite possible for this to remain undiscovered until it causes an explosion of the boiler.

Sometimes the upper part of the shell is covered by a "lagging" of wood stuffed with sawdust, but this practice is no more to be admired than a brick-

Scale. 6 Feet to 1 Inch.

SECTION THROUGH LINE A. B

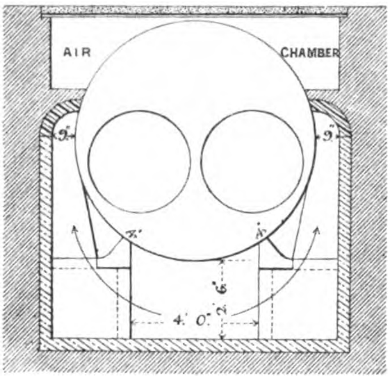


Fig. 119.

SECTION THROUGH LINE C. D.

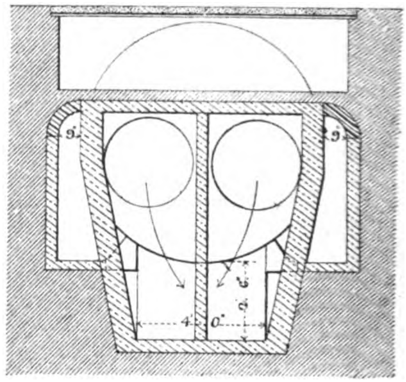


Fig. 120.

ALTERNATIVE ARRANGEMENT OF COVERING

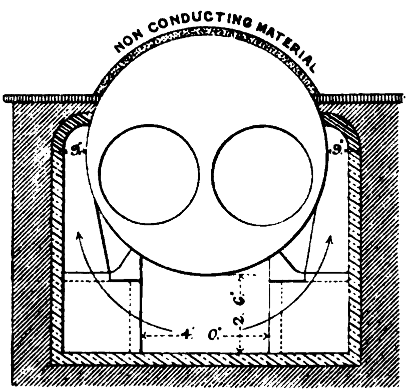


Fig. 121.

FRONT ELEVATION OF LANCASHIRE BOILER

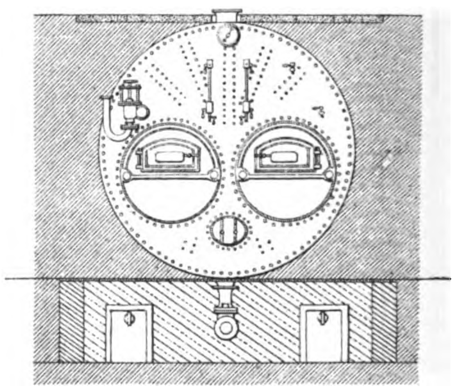


Fig. 122.

THE SEATING OF AN ORDINARY LANCASHIRE BOILER.

work covering. Coverings of patent felt and fibrous substances, which are bad conductors of heat, are much to be preferred to the previous-mentioned methods of external protection. An objection, however, to the use of these coverings is, that they do not admit of the detection of leaks so readily as in uncovered boilers. The best and most approved arrangement of external protection is that of providing a chamber closed in by thick tiles as shown in Figs. 117—120.*

This method renders the upper portion of the shell quite accessible at any

* See Report of the Chief Engineer to the Boiler Insurance and Steam Power Co., Manchester, June, 1881.

remain in contact with the shell at the seating blocks, nor would moisture from the damp situation be likely to reach the shell at all, as the top of the blocks is considerably above the side flue floors.

The front of the boilers should be parallel with and close to a railway or siding, so that coal may be brought and ashes removed conveniently, and to prevent loss of heat in pipes, the nearer the boilers are to the engines the better.

Boiler fittings.—A *man-hole* large enough to admit of a person getting into the boiler to inspect its condition and do any necessary cleaning or repairing. A *blow-off cock* usually placed in front of and below the level of the boiler; a pipe leads from the bottom of the boiler, the end of the pipe being closed by a valve able to withstand the pressure of steam. The blow-off cock, on being turned, allows the water from the boiler to run through this valve. The *water-gauge*, Fig. 123, is a thick glass tube placed in the end of the boiler, one end of which communicates with the steam space above the level of the water in the boiler, the other end below the water level. It thus indicates the level of the water in the boiler. It is

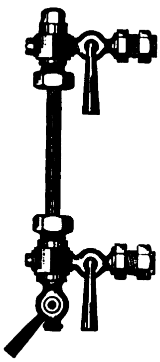


Fig. 123.—Water-gauge for Boilers.

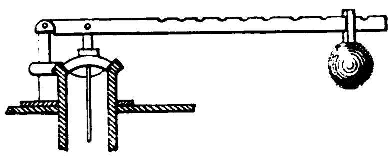


Fig. 124.—Lever Safety-Valve.

usually provided with a cock at each end by means of which it may be cut off from connection with the boiler, so that in case of a broken glass, the steam and water can at once be prevented from escaping and a fresh glass inserted. There is another cock at the lower end of the tube by which means the water in the tube can be allowed to escape from it. Another plan for ascertaining the level of the water is to place two *gauge-cocks* in the end of the boiler, the one being two or three inches higher than the other, the one being above and the other below the water level in the boiler. If the water is at its proper height steam should issue from the upper one on opening it and water from the lower one. These gauge-cocks should always be placed on the boiler in addition to the water-gauge. Still another method of ascertaining the height of the water in the boiler is by means of a *float* resting on it and communicating with a chain passed round a pulley above the boiler, at the end of which is attached a weight moving up and down with the float in the boiler. One of the most important of the fittings is the *safety-valve*, Fig. 124. The necessity for this is evident, for an engine may stand a considerable time, during which the pressure inside the boiler goes on increasing, and if some provision were not made for its escape an explosion must occur. The safety-valve should be placed directly on some convenient portion of the surface of the boiler, in preference to being attached to the pipe leading from the steam-valve, as often seen. The most common form is the lever. The valve is kept close by means of a lever fast at one end and having a sliding weight at the other, and near the fixed end a spindle passing through the valve, resting in its seat, communicates with the boiler. The objection to its use lies in the fact that it sometimes gets corroded and sticks in its seat, and it is being supplemented by the use of dead weight and spring-valves.

Fig. 125 is a sketch of Hopkinson's 1890 patent compound safety-valve, designed to guard against either excess of pressure or deficiency of water.

The valve here shown is intended for use only with pressures below 120 lbs. per square inch, that for higher pressures being different.

The drawing shows a section of a boiler fitted with one of these valves. The exterior valve is held to its seat by the combined effect of the external weight on the lever and the internal weight on the rod, while the internal valve is held down by the weight suspended under it only. On the steam attaining too great a pressure in the boiler the external valve rises and discharges round its seat, the internal valve remaining closed. If the water is allowed to get too low, the balanced float drops with the falling water level, and this causes the projection seen on the balance-lever to come into contact with the screw-adjusted collar on the weight suspension rod, thereupon raising the inner valve and allowing steam

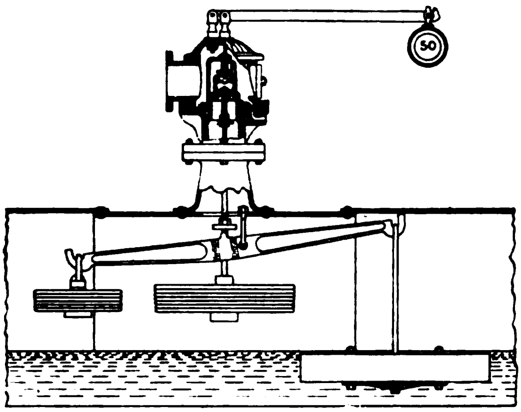


Fig. 125.—Hopkinson's, 1890, Patent Compound Safety Valve.

to escape. Whether the pressure is too high or the water too low steam is thus blown off at the valve. The external fixed cage is designed to guard against the risk of the outer valve being blown out should the internal suspending rod break.

The valve designed and made by Messrs. Hopkinson for pressures above 120 lbs. to the square inch is termed the "Hipress." In it the high-steam and low-water valves are separated. The steam valve is spring-loaded; the low-water steam blow-off valve rests on a separate fixed seat and both valves blow off steam to the same space.

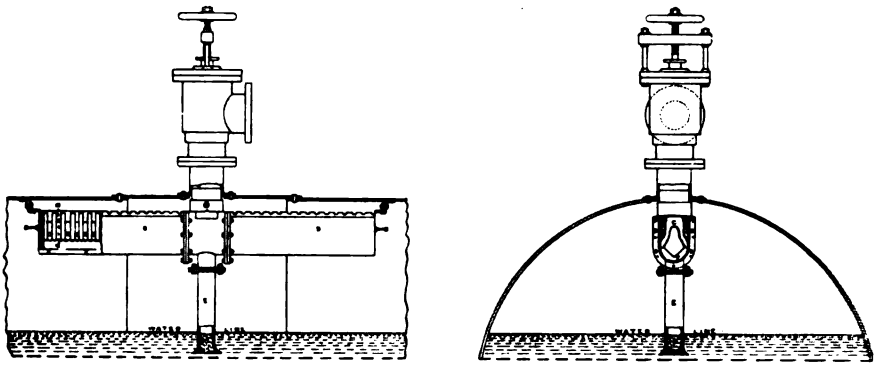
All boilers should have two safety-valves, or a safety-valve and an *escape-valve*, the latter allowing the steam to escape on attaining a certain pressure.

A good plan is to fit a boiler with a Hopkinson's patent compound safety-valve and a Hopkinson or Cowburn dead-weight safety-valve.

The *feed water-valve* is so arranged as to allow the water to flow into the boiler, at the same time preventing any from returning. It is used to regulate the supply of water to the boiler.

The *steam, crown, or stop-valve* regulates the supply of steam from the boiler to the steam-engine, and it may be closed or opened by means of a screw worked by a hand-wheel.

McDougall's patent *anti-primer* is a very interesting and novel appliance for preventing priming in steam boilers. After separating the water from the steam, the former is conveyed to the water space of the boiler by channels, in which there is no rush of steam to impede its return. The apparatus is therefore able to return the priming water to the boiler, leaving the dry steam to pass out. Figs. 126 (A) and 126 (B) show the apparatus applied to a Lancashire boiler, the former being a longitudinal and the latter a transverse section. A casing, D, contains a series of water-pockets, C, which have slots at the upper sides, B, through which the steam issuing from the boiler is compelled to pass, there being no other outlet.



A. B.

Fig. 126.—McDougall's Patent Anti-Primer.

The pockets are somewhat arched, the steam and priming water entering the slot or opening, passes on both sides of the arch, the water is conveyed along the bottom of the channels, C, to the lower chamber, F, connected with the dip pipe, E, the lower end of which passes into the water in the boiler, and thus, the dip pipe and water passages are free from any current of steam. The apparatus prevents the dirt and scum from being carried into the working parts of steam engines, &c., as they flow along with the water, are intercepted with it and returned to the boiler. The apparatus can be fitted to all types of land boilers.

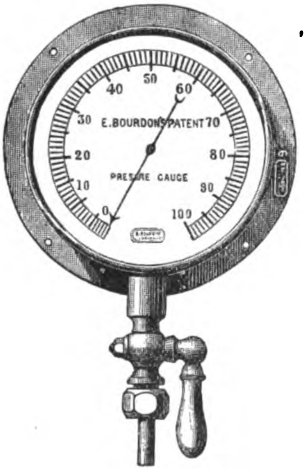


Fig. 127.—Bourdon Pressure-Gauge.

Dampers are placed in the flues to regulate the draft on the furnace, and these are hung by chains passing over pulleys with balance-weights attached.

A *fusible plug* is sometimes employed to guard against the risk of explosion. A short nozzle is attached to the boiler just below the lowest level at which the water may safely stand. On this there is screwed a cap, the central portion of which is composed of an alloy which melts at a temperature not very much exceeding the boiling point. So long as this plug is kept covered with water it remains firm, the heat being carried away from it by the water, but should the level of the water fall so low as to expose it, the centre at once melts, and allows the steam and part of the water to escape into the flues and furnace, damping or extinguishing the fire, and at the same time removing the undue pressure. An objection to its use is, that it cannot be relied on after being in some time, as it becomes incrusted or injured by the heat and loses its efficacy. The *pressure* or *steam-gauge* is the means by which we ascertain the pressure of the steam. Mercurial gauges were at one time employed, but now "Bourdon's" is chiefly used. In this there is a dial-plate with a hand on it pointing to the pressure (see Fig. 127). The steam acts upon a spring of peculiar construction, causing the index hand to move according to the steam pressure, and the figure at which the hand points denotes that the pressure inside the pipe to which it is fixed is that much above atmospheric pressure. Thus, if the hand points to 20, we know that the pressure of steam is 20 lbs. above that of the atmosphere. A pressure-gauge should be fixed to each boiler, and one also to the main steam-pipe common to all the boilers of a range. In Chapter XVI. of this work pressure-gauges are more particularly referred to.

A *mud-hole* is placed near the lowest part of the boilers. In the Lancashire boiler it is placed in the front end-plate; the orifice is oval in shape, and its cover is occasionally removed for the discharge of sediment.

An instrument largely used by engineers to test the accuracy of the pressure-gauges fixed on boilers is shown in Fig. 128. This *pressure test indicator* is manufactured by Mr. Joseph Casartelli, of Manchester, whose springs for instruments are skilfully and accurately made so as to be trustworthy throughout their range of action. The indicator consists of a barrel covered with wood containing a cylinder and piston with a spiral spring of either 25 or 50 lbs. to the inch. The scales on either side of the spring are graduated to correspond, and the pointer at the side moves up and down with the compression and tension of the spring. The springs range up to 100 and 200 lbs. respectively, but can be made stronger if required for higher pressures. The instrument is carried about with the engineer, who, when he wishes to test the accuracy of a working pressure-gauge, notes its indication. The steam is then shut off in the pipe to which it is attached and which communicates with the boiler. The pressure-gauge is then unscrewed and replaced by the pressure test indicator. On turning the cock to admit steam to the instrument the piston is forced up against the spring and the pointer shows the steam pressure on the scale. The reading thus indicated is noted and compared with that of the gauge; the indicator is then unscrewed and the pressure-gauge returned to its position on the pipe, unless its indication has been proved to be inaccurate, in which case one giving accurate readings must take its place. On many boilers now there is fixed a special cock, or tap, made to receive the nipple of the pressure test, so as to do away with the necessity of removing the ordinary gauges.

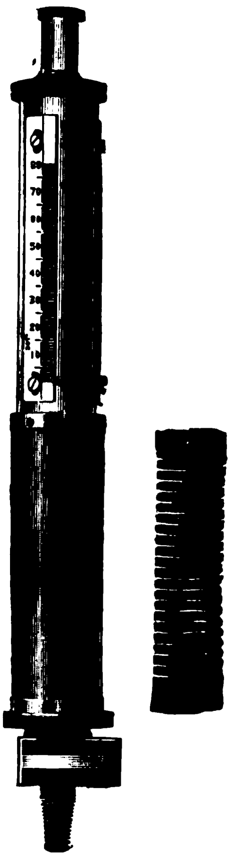


Fig. 128.—Pressure Test Indicator.

The furnace of a Cornish or Lancashire boiler consists of the mouthpiece, having doors provided with a sliding grid, shown in Fig. 122. The furnace bars are made in two lengths as shown in Fig. 131. At the front end these bars rest upon the dead plate, and at the back at a slightly lower level so that the bars may incline inwards. They are supported by the fire-brick bridge, either on a ledge formed on the bridge for the purpose, or a bearer built in it. In the middle at the joint of the two lengths the bars are supported by a cross-bearer. The bridge is usually built entirely of fire-brick to within about 20 inches of the crown of the internal flue, but sometimes a cast-iron stool is used to carry both the furnace bars and the fire-brick. The stool is provided with a sliding door, by means of which the admission of air to the furnace flue is regulated from the furnace mouth.

Some furnaces have a self-feeding arrangement. The coal in these is placed in a hopper above the furnace at the front end of the boiler, and a small revolving scoop driven by an engine constantly and slowly sprinkles the coal into it. A slow motion is also imparted to the furnace bars, so that the burning fuel is gradually carried to the back of the furnace as fresh coal is supplied in front. The objection to this plan is the complicated nature of the mechanism and the power required.

In the Bennis and also in Proctor's mechanical stokers the coal is distributed over the fire by flaps or shovels.

The economical working of steam-boilers requires not only that they shall be of proper design and maintained in efficient repair, but that strict attention shall be given to all details affecting the consumption of fuel with the view of obtaining its complete combustion. The fire-grate is, therefore, a highly important factor in the economy of boilers, and especially so where fuel can only be obtained at a high price.

It has been proved by the researches of Wye Williams, Sir William Fairbairn, and others, that a considerable saving is to be effected by the admission of air at the back bridge. Fairbairn has stated, as a result of experiments in this matter, that a clear gain of 4 per cent. was effected, and that it was thoroughly effectual in preventing smoke emission.

Caddy's patent tubular fire-bars have now been before the public since 1886, and have been adopted by the Admiralty, H.M. Office of Works, public corporations, and the leading boiler-makers throughout the world. They are not only used for ordinary furnaces, but have been applied with increasing success to most of the mechanical stokers in the market.

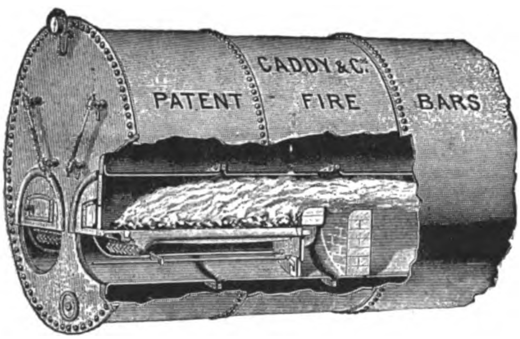


Fig. 129.—CADDY'S PATENT TUBULAR FIRE-BARS.

The patent consists of a hollow bar (see Fig. 129), cast face downwards, round an inner flattened tube of wrought-iron, and has a face-breadth of one inch, making it suitable for slack coals. Being cast in chill, face downwards, the surface is made smooth, and less liable to gather clinker.

The internal wrought-iron tube affords a passage for air from the front, to be delivered at a split bridge at the rear of the grate as an aid to smoke prevention. The air becomes heated and the bar cooled thereby; this tends to prevent the adherence of clinker.

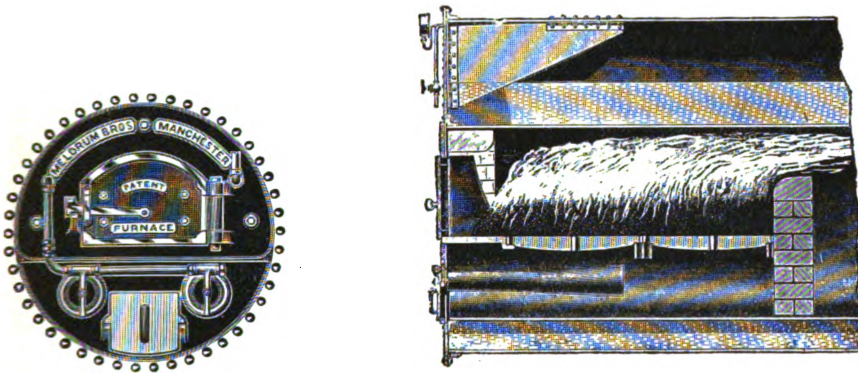
The face is somewhat rounded, and the total depth of bar is $4\frac{1}{2}$ inches. Each bar has an air-way of about three-fourths of a square inch. Hence, with air-spaces of $\frac{1}{4}$ inch, the air-way through the bars amounts to about 7 square inches per foot of breadth, or about an inch and a half per square foot of an ordinary grate.

To show the advantage of the adoption of these bars, a large number of tests have been made, under the Alkali Works Regulation Act, full details of which can be obtained from the Government Blue Book. A ten hours' test of Caddy's patent bars, made with Lancashire boiler, 34 feet × 8 feet 6 inches, with 9 tubes, gave the following results:—Water evaporated per lb. of coal from and at 212° F. =7·97 lbs. Water evaporated per lb. of carbon value from and at 212° F.=

9·74 lbs. Percentage of ashes=13·25 per cent. Temperature of gases leaving boiler=215° F. No smoke. The gas analysis showed of carbon dioxide, 9·43 per cent.; carbonic oxide, *nil;* oxygen, 8·81 per cent.; nitrogen, 81·76 per cent. The coal used was Bridgewater rough slack.

If desired, the patent fire-bars can be fitted with a rocking attachment, not shown in the illustration, patented by the same firm. By means of this gear a dirty fire may be easily and quickly cleaned, without opening the furnace-door, or using the pricker-bar, which in careless hands causes injury by pricking the bars up. The motion is so arranged that by pressing a lever every alternate bar moves longitudinally about three inches. During this backward and forward traverse the front end of the bars in motion rises and falls an inch and a half, while the other half-set of bars remains in its normal position. This compound motion cleanses and renders the fires bright and assists in the reduction of smoke.

In chimneys of ordinary height the draught, which seldom exceeds that equal to about $\frac{3}{4}$ inch of water gauge, is insufficient for the combustion under boilers of



A.—Front Elevation. B.—Longitudinal Section.

Fig. 130.—The Meldrum Furnace applied to a Cornish Boiler.

any but the better class of fuel, and in order to burn inferior kinds it has been found advantageous to employ, as an auxiliary, what is termed forced draught, and in this way combustion may be supported in a fire fed by a close-lying fuel.

The Perret system for burning coal-dust and other rejected fuels is an example of an apparatus for producing forced draught. In it specially constructed fire-bars are used, which dip into a trough or tank containing water placed in the ash-pit. As the water in the trough is evaporated, the steam tends to cool the bars and reduce their rate of waste by burning. By means of a fan air is forced through pipes to each grate, so that the space between the water-line in the trough and the coal above the bars is under pressure. This tends to prevent the formation of clinkers. The vertical currents of air between the bars tend to cool them, and also to prevent dust from falling into the tank.

The Meldrum furnace is an ingenious and carefully designed system of forced draught, by means of which refuse fuel may be utilised for steam-raising; and it is stated that by using ordinary breeze or coke-dust containing 20 per cent. of moisture, 15 per cent. of clinker, and 65 per cent. combustible, $6\frac{1}{2}$ lbs. of water per lb. of fuel have been evaporated with these furnaces.

Fig. 130 shows the furnace as employed in a Cornish boiler, A being a front elevation, and B a longitudinal section. The thin bars are placed very close

The Exhaust Injector is the invention of Messrs. Hamer, Metcalfe & Davies. The feed-water tank must be placed at a slightly higher level to that of the injector. The latter may be anywhere connected to the main exhaust-pipes, but the nearer to the boilers the better. An elbow in the exhaust-pipes makes a convenient point of connection. As the exhaust steam passes along the pipes, the injector takes sufficient to supply its own wants without interfering with the course of the rest. It is capable of feeding any boiler working at not more than 60 lbs. pressure, so that when the exhaust steam is at atmospheric pressure, or 15 lbs. absolute, the feed-water opposes and overcomes a pressure against it of 60 lbs. per square inch.

The exhaust injector in no way affects the working of the steam engines. To start it, communication is opened with the exhaust steam and with the feed-water supply. It may be arranged to work as an ordinary injector so as to feed the boiler when the engine is standing. The temperature of the feed-water must not exceed 90° F.

The construction of the exhaust injector is different from the injector. In the former the exhaust steam enters the top of the injector and passes into a conical nozzle. This nozzle is fitted with a fixed conical spindle, the object of which is to concentrate and direct the flowing exhaust into a circular jet by the time it meets the cold feed-water, and so precipitate condensation. The feed-water enters a branch at the side of the injector near the top, and on flowing into the conical chamber surrounds the steam in its passage through the nozzle. Condensation begins as the exhaust and feed-water meet in the condensing chamber, which forms the upper part of the hinged nozzle. The hinged nozzle imparts to the exhaust injector its automatic action. When not at work the nozzle hangs back, by this means giving space for the water and steam to meet. Immediately after meeting, condensation takes place, a partial vacuum is formed, and the hinged nozzle assumes a position which reduces the space through which the combined exhaust and feed-water pass.

The exhaust injector requires no attention, and is automatic in its action, starting and stopping with the engine to which it is connected. It is very efficient where engines work without stoppages, and is said to give good results even where engines do not run continuously, but whose motion is without interruption for several minutes at a time, and if a colliery winding engine runs sufficiently long during a period of winding, it may work satisfactorily with such engine. The waste steam, in passing through the injector, heats the feed-water to a temperature of about 190°. A saving in fuel from 15 to 20 per cent. is effected by the application of this injector as compared to pumping the water into boilers.

There must be the usual back-pressure valve on the boiler; and if the delivery-pipe is long, an additional back-pressure valve must be fixed two or three feet away from the injector.

It has been successfully applied to winding engines, fan engines, and hauling engines at collieries.

The main flue into which the smoke from the boiler is discharged leads to the chimney, which is necessary to create a draught. The main flue should have an area equal to that of the chimney at its base, unless more than one main flue discharge into it. The chimneys are larger at the bottom than at the top, the sides having a taper or batter of about 1 inch per yard. They are lined with firebrick for a considerable distance from the base. To determine the size necessary for the chimney, allow 5 square feet for each boiler, and let the height be equal to 25 times the internal diameter. Chimneys are built of many forms, but the circular seems to be the best.

The following table of round chimneys is taken from *Power Steam* :—

line, which causes them to be interchangeable, and reduces the strain upon the iron in the manufacture.

"Reliable tests of the efficiency of steam boilers conducted under independent

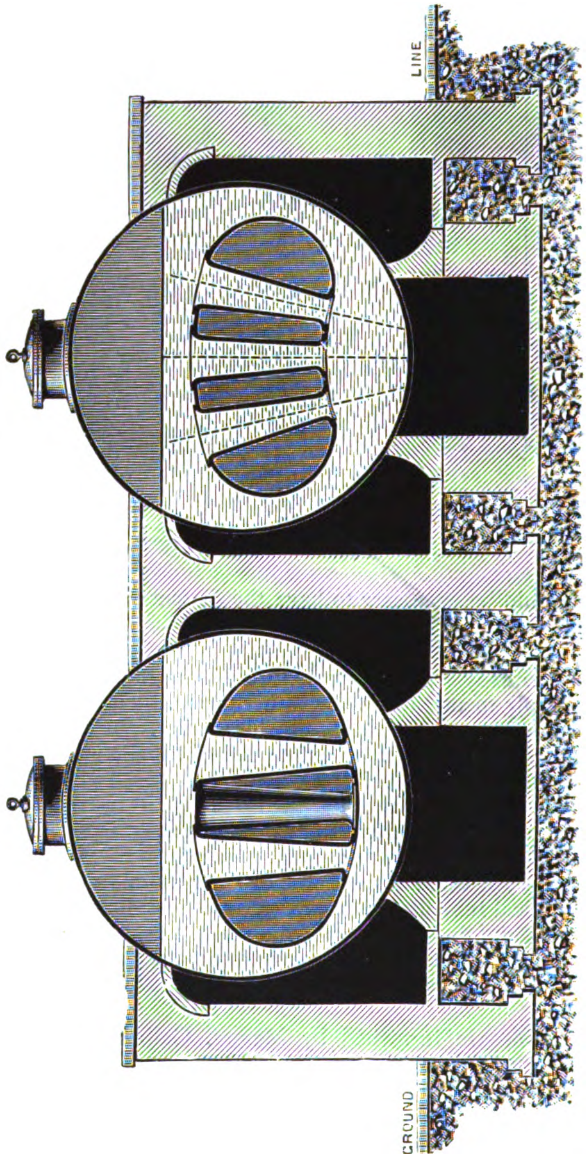


Fig. 144.—The Galloway Boiler.

On the latest patent. As formerly made.

and competent superintendence are difficult to obtain, but on many occasions the Galloway boiler has been proved to give a higher result than any other in the market. We may, however, instance the official tests at the Philadelphia Exhibition of 1876, which were carried out by a special committee, and entirely without any interference or control being possible on the part of the makers of the

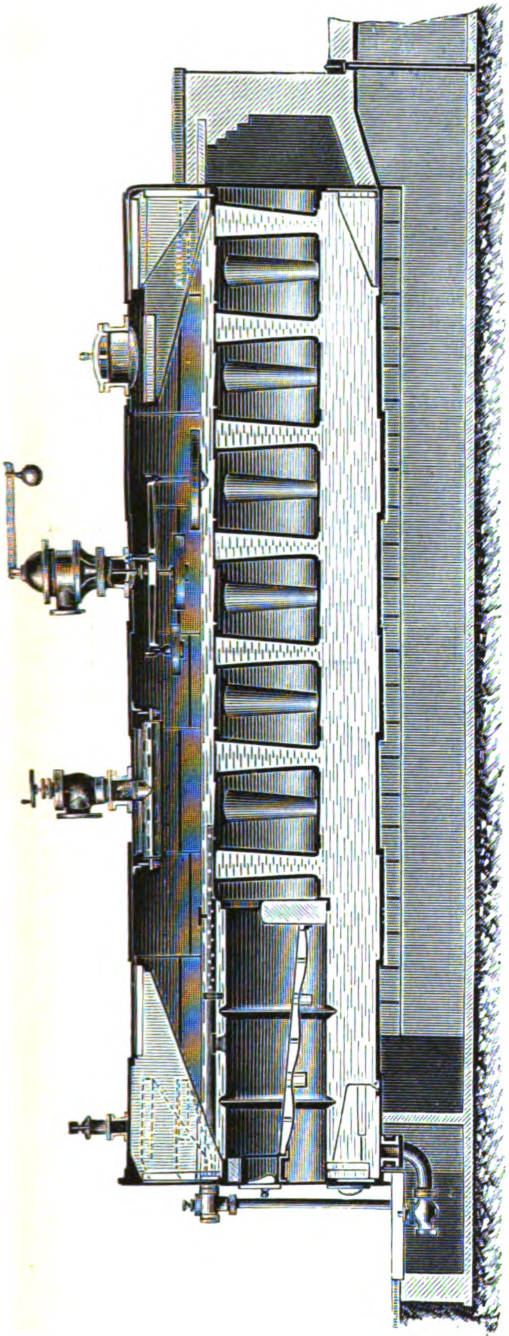


Fig. 145.—The Galloway Boiler. Longitudinal Section, with Fittings and Furnace Apparatus in Position.

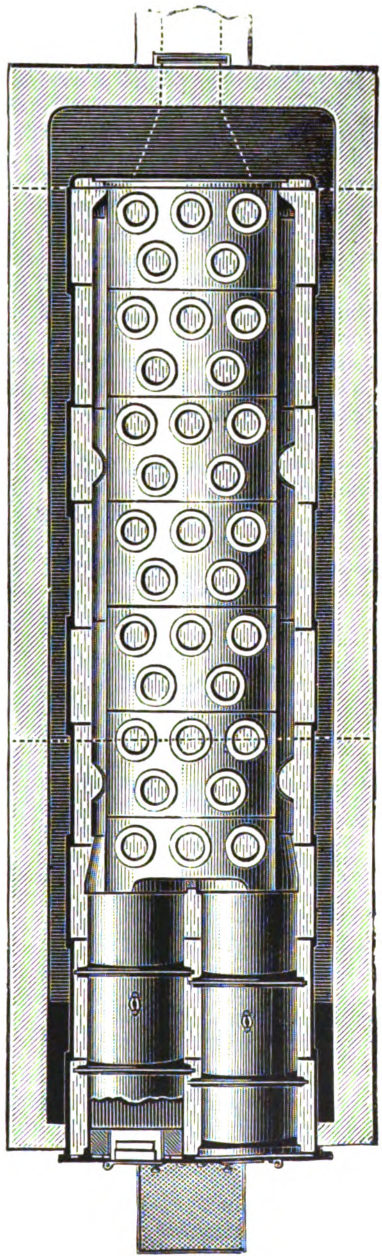


Fig. 146.—The Galloway Boiler. Plan showing Disposition of Tubes and Brickwork Bed.

boiler, where the Galloway boiler evaporated 11·72 lbs. of water, at a temperature of 212° per pound of coal; and also a test carried out at the mill of C. R. Collins, Esq., Hele, Devonshire, under the immediate supervision of the editor of *Engineering*, at which a result equal to 12·83 lbs. of water at the same temperature per pound of Welsh coal was attained. At Philadelphia, moreover, the quality of the steam generated was carefully tested, and, in addition to the highest rate of evaporation, the Galloway boiler was ascertained to give the driest steam. This is a matter of the utmost importance, as with perfectly dry steam there can be no risk of break-down occurring to an engine from that most frequent cause, priming.

"It has been ascertained in practice that a Galloway boiler is considerably more powerful than an ordinary two-flued or Lancashire one of equal dimensions, and as it burns the same amount of fuel it is correspondingly economical. A Galloway boiler 28 feet long by 7 feet diameter, when driving a condensing engine in fair order, is fully equal to 300 indicated horse-power, but in many cases one such boiler has regularly driven over 350 horse-power with compound engine, at a working pressure of 80 lbs. per square inch; this is considerably higher than can be obtained from a boiler of the ordinary construction."

Messrs. Galloways supply also the subjoined summary of official tests of boilers at the Philadelphia Exhibition, 1876, where thirteen boilers were tested for eight hours each, at a pressure of 70 lbs. to the square inch, "when the Galloway boiler gave the most economical result, and furnished the driest steam."

Description of Boiler.	Heating Surface in Square feet.	Horse-Power at 1 cubic ft. Water evaporated per Hour.	Lbs. of Water evaporated.			Per Centage of Water in Steam.	Lbs. of Coal Burnt.		Temperature of Gases leaving Boiler.	Cubic Feet of Water Space per Horse-Power.	Cubic Feet of Steam Space per Horse Power.
			Total.	Per Hour.	At 212° per lb. combustible.		Per Hour.	Per Square Foot of Grate Per Hour.			
Galloway ...	973	14·64	20824	2603	11·72	·57	283	7·269	324	14·10	4·04
Root	1590	54·29	27146	3393	11·565	not taken	381	9·09	393	2·29	·89
Fermenich ...	1078	26·46	13233	1654	11·53	not taken	185	11·79	415	4·08	2·63
Lowe	774	21·45	10729	1341	11·489	not taken	153	6·805	332	9·02	2·63
Babcock ...	1680	62·70	31358	3919	11·489	3·24	444	9·77	295	3·74	2·20
Andrews ...	540	18·94	9469	1183	10·513	not taken	148	not taken	419	4·14	1·29
Wiegand ...	1355	68·08	34042	4255	10·461	not taken	517	12·32	523	2·66	·64
Anderson ...	1135	44·44	22230	2778	10·255	not taken	350	9·747	417	1·43	1·28
Kelly	662	37·41	18710	2338	10·099	5·97	291	10·82	not taken	1·91	·68
Harrison ...	900	36·57	18285	2285	10·022	1·11	284	12·36	517	1·93	·80
Pierce	200	23·74	11876	1485	9·818	5·53	199	7·99	373	·65	1·64
Exeter ...	1525	32·65	16334	2041	9·765	4·63	280	9·35	429	2·66	1·43
Rogers & Black	399	21·13	10564	1320	9·31	2·68	181	8·05	571	1·71	1·17

In the Water-tube Boilers the flame and hot gases from the furnace act directly on rows of parallel tubes of small diameter through which the water is passed, and a receiver is placed over the tubes for the steam given off in them. They are very rapid steam generators, and like other boilers have their advantages and disadvantages.

The following description of the Babcock and Wilcox water-tube boiler is taken from the *Practical Engineer* of June 22, 1888:—

"The exhibit of the Babcock and Wilcox Co., of New York and Glasgow, consists of one of their patent water-tube boilers, as illustrated above. It has 64 tubes, 4 inches diameter and 18 feet long, with a steam receiver 4 feet diameter by 22 feet 4 inches long, and is suitable for a working pressure of 180 lbs. per square inch. It is rated at 126 H.P., and is estimated to be equal to a Lancashire boiler 30 feet by 7 feet 6 inches at 150 lbs. pressure. The rated horse-power here means 30 lbs. of water evaporated from 212°, at 70 lbs. pressure for each horse-power.

"It will be seen that the boiler consists of a series of lap-welded wrought-iron tubes, placed zig-zag one over the other, and connected at each end by vertical headers to a steam and water-receiver placed above in a horizontal position; while below, at the back end, there is a cast-iron mud chamber or sediment-collector, to which is attached the blow-off. These four different parts, when fitted together, constitute the boiler, and it is by the ingenious combination of these that the present state of perfection has been arrived at.

"The tubes are connected with the headers by being expanded into accurately bored and tapered holes, the headers again being connected by tubes to a substantial riveted block on the under-side of the water and steam-receiver. This method is stated to give every satisfaction under all pressures and conditions.

"The receiver itself is a cylindrical shell of Siemens-Martin steel, double-riveted

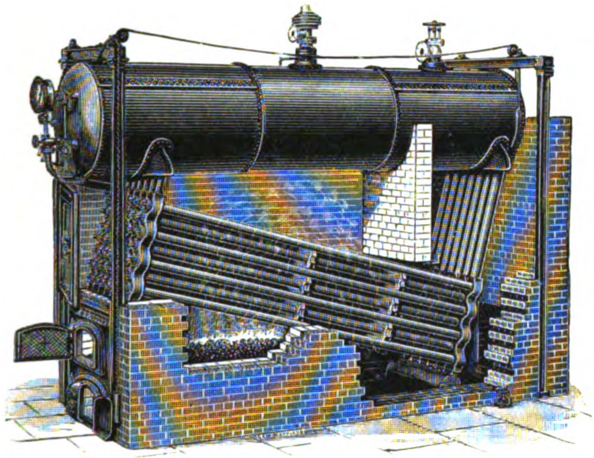


Fig. 147.—The Babcock and Wilcox Water-tube Boiler.

in the longitudinal seams, and single-riveted circumferentially. The end plates form the segment of a sphere, or are what is usually termed 'dished.' This being the strongest form in which they can be made they do not require any further staying. The manhole, for internal examination, is placed in one end, and is formed by flanging the plate inwards and facing the edge, so that a metal-to-metal joint can be made.

"The vertical headers have been usually made of cast-iron, but the firm have on exhibition a wrought-iron one, similar in design to those in use, as shown in Fig. 147. It is a beautiful piece of work, of great strength, and will considerably add to the confidence already placed in the safety of the boiler.

"The fire is placed under the tubes at the front end; the flames and heated gases rise to the under-side of the horizontal water drum, and thence pass downwards among the back portion of the tubes, and on to the chimney. There is no fear of any evil effects from this arrangement, as the boiler is suspended from above by means of hoops round each end of the steam and water receiver, and is thus entirely free to move and accommodate itself to the variable movements consequent on expansion and contraction.

of water per lb. of coal at a pressure of 70 lbs., and from a temperature of 180° with 12 per cent. of refuse.

"In 1883, at another trial in Pittsburgh, 10·9 lbs. of water were evaporated from each pound of bituminous coal—less 11 per cent. of ash—at a pressure of 70 lbs. from and at 212°.

"Again, during trials at San Francisco, with three different kinds of coal—viz., British Columbia, Welsh, and Washington—there were 11·1 lbs., 11·8 lbs., and 10·4 lbs. of water evaporated per pound of pure coal respectively. Efficiency trials have also been made in this country, and we extract the following from the company's report, dated July and August, 1885, regarding tests at the Gaythorn works of Messrs. J. and J. M. Worrall, dyers, Manchester, with a 140 H.P. boiler:—

	No. 1.		No. 2.
Heating surface	1,616 sq. ft.	...	1,616 sq. ft.
Grate surface, 6 ft. 6 in. by 4 ft. 6 in.	29 sq. ft.	...	29 sq. ft.
Ratio of heat to grate surface	55 to 1	...	55 to 1
Duration of test	27 hours	...	$18\frac{1}{2}$ hours
Average observed steam pressure	75 lbs.	...	95 lbs.
Average temperature of water fed to boiler by injector	135°	...	275°
Pounds of coal fired	25,536	...	11,704
Coal consumed per square foot of grate per hour	31·7	...	21·6
Total water evaporated	216,000 lbs.	...	118,400 lbs.
Water evaporated per hour	8,007 lbs.	...	6,400 lbs.
Water evaporated per pound of coal—actual conditions	8·46	...	9·85
Rated H.P. (one H.P. = 30 lbs. of water evaporated from 212° at 70 lbs. pressure).	140	...	140
Temperature of flue gases	700°	...	680°

"In trial No. 1 the 27 hours include three stoppages, aggregating $2\frac{1}{2}$ hours, in which but little steaming could have taken place, so that the figures represent ordinary working conditions.

"From the careful manner in which the reports of the various tests of this boiler have been drawn up, it would appear that great care has been exercised to arrive at correct and reliable results."

Arnold's boiler resembles an ordinary Lancashire or Cornish boiler, the chief points of difference being in the construction of the internal flue and the substitution of a long tube for the Galloway tubes, the tube extending from the inside of the firebridge, with a slight inclination inwards, to a point near the back end of the boiler. The long tube is connected to the internal flue at the top near the fire-bridge and at the bottom near its other extremity, and so forms a passage for the water. This, it is said, improves the circulation and adds to the heating surface of the boiler. Each ring forming a part of the flue, instead of being of a uniform diameter, is bulged into the shape of a barrel, the object being to impart greater strength, and increase the heating surfaces of the fire-boxes of steam boilers.

Vertical boilers are useful where only a small amount of power is required, but they are not economical steam producers. There are many varieties of vertical boilers; they are generally fitted with either Galloway tubes or parallel tubes to improve the water circulation in them, and also to facilitate the escape of steam.

neglecting to feed the furnaces, thus allowing the fire-bars to get bare, a passage being thereby formed through which the cold air passes; by driving the furnaces too much and generating steam rapidly, afterwards opening the furnace doors in order to prevent blowing-off at the valves; by using means when being laid off to quickly cool the boiler instead of waiting till the brickwork is cool before running the water out, and allowing time for the boiler to cool gradually; by filling or partly filling it with cold water immediately after it is emptied; and on the boiler resuming work, by forcing the fires and raising steam more rapidly than should be attempted from cold water. Intense firing should specially be avoided in boilers having thick plates.

Care is required in hand-firing boilers, otherwise much smoke is emitted. The fuel should be introduced frequently and in small quantities which must be spread thinly and evenly on the front portion of the fire-grate, and at intervals the hot fuel pushed back. No portion of the grate should at any time be bare so as to form an air-passage through the fire.

Every boiler should be thoroughly emptied of its contents periodically, say once a week or once a fortnight according to the water used, and afterwards be filled by an entirely fresh supply.

All natural water contains more or less of foreign matters, and none, therefore, is ever pure. The impurities may be gaseous, mechanical, mineral, or organic; one class will predominate in one locality and another elsewhere. The purest water obtained is rain-water, but even that is charged with the atmospheric gases oxygen, nitrogen and carbonic acid gas. In 100 parts of water there are about 2 parts of air, but the amount varies slightly. Next to rain-water in point of purity is that which flows over or through soil which is not easily dissolved, or water which receives no other soluble matters than those washed down by the rains, such as that of mountain torrents, and lakes on rocky surfaces of slate or granite.

Certain gaseous impurities in waters result from the strata through which they flow; such are carbonic acid and hydrogen sulphide. Part of the rain which falls upon the surface of the earth is evaporated, another portion finds its way directly to the surface streams, whilst the remainder sinks into the earth's crust, to issue afterwards at the surface in the form of springs. Where the course taken by the water in following the natural fissures is tortuous and long, it may pass through rocks of varying character and composition. Water has great solvent power, and in flowing through certain strata becomes charged not only with mineral matter, but also often with gas; as for instance, the waters of Bath, Harrogate, Llandrindod, &c., which have dissolved hydrogen sulphide.

Mechanical impurities are those mixed with and suspended in the water arising from its passage through or over the earth. They consist of clay, chalk, sand, &c., which are insoluble in water. They affect the purity of the water, but can be removed by subsidence or filtration.

Dissolved mineral or chemical impurities in waters result from their passage through or over the earth, and are those most commonly found. They consist of carbonates and sulphates of lime, and carbonates of magnesia, potash, and soda, chloride of sodium, &c. Water impregnated with iron is said to be chalybeate. The presence of the salts of lime and magnesia renders water hard. Sulphuric acid, arising from pyrites, seriously corrodes the metal of pipes, rails, and pumps, and rots the leather of boots.

Organic impurities in water arise from animal and vegetable organisms, either living or dead.

Glasgow is partly supplied with water from Loch Katrine, which in point of purity compares favourably with that of any city in the world, while the water supplied to London is in great part obtained from the river Thames and contains nine times as many impurities as that from Loch Katrine. The water used for domestic purposes in the town of Pontypridd is for the most part derived from a mountain torrent, and shows on analysis about double the impurities of the Loch Katrine water.

The water available for boilers should be first submitted to careful analysis by a competent chemist, and if found to be of good quality, it may require no treatment. But if certain impurities exist, efforts must be directed to neutralise their effect, and these must be guided by the analysis.

The effect of supplying boilers with water containing only mechanically suspended impurities is to have a deposit of mud formed in the boiler, and although this is not so objectionable as an incrustation, the water should not be allowed to enter the boiler, unless some means are afterwards taken to remove it before the sediment forms a deposit. Where no provision is made to purify water of this description either before or after it enters the boiler, it necessitates frequent blowing off and cleaning. If no other impurities are present in it, the water may be greatly improved by being filtered before it enters the boiler, as water percolating through sand is divested of much of its mechanical impurities.

Waters which contain certain mineral impurities require chemical treatment in order as far as possible to prevent them from forming deposits within the boiler, and from wasting the plates by internal corrosion. The amount of cleaning necessary in the boiler will depend on the success of the chemical treatment.

An incrustation formed in the boiler interposes a bad conductor of heat between one side of the boiler plates and the water at a point where the other side of the plates is exposed to the fire. This leads to considerable waste of fuel, and to danger of water coming into contact with an over-heated plate owing to cracking or loosening of the scale.

Carbonate of soda may be added to water which contains sulphate of lime, which will then be converted into carbonate and form a soft scale which may be easily removed.

Some benefit is derived, where hard water is used, by frequently blowing off the water and so getting rid of matter which if allowed to remain would form into scale. This method is wasteful as compared with purifying previous to heating the water.

Carbonate and sulphate of lime and carbonate of magnesia form the larger part of ordinary scale found in steam boilers.

The salts of lime and magnesia in the form of carbonates are easily dissolved in water which contains carbonic acid gas. In water raised to the temperature of 212° F. carbonic acid gas is driven off, and the maintenance of the same or an increased temperature is followed by the salts being deposited on the interior of the vessel in which the experiment is conducted. In steam boilers the result of such a deposit is to form scale.

If a similar experiment be tried with water which contains sulphates of lime or magnesia, it will be found that the salts will not be deposited on the water reaching a temperature of 212° F., and it is not until the water has undergone much evaporation that it loses its power of holding the sulphates in solution, after which they are precipitated on to the surface of the vessel as were the carbonates at a lower temperature.

A number of anti-incrustation compounds are manufactured for the purpose of preventing the water impurities forming into a scale on the boiler whilst the compositions do not themselves injure the plates. The addition of a compound to act chemically after the water is in the boiler, no doubt in some instances renders the deposit formed in the boiler softer and therefore easier removed, but to be effectual the composition must contain elements, specially arranged to combine with the impurities in that water wherein it is placed. It is impossible therefore for any one composition to eliminate all kinds of impurities from water.

Elaborate and more costly methods of purifying water—such as the "Porter Clark" or the "Stanhope" water purifier, &c.—may be adopted for removing the salts of lime and magnesia from the water before it is passed into the boilers.

A simple and most efficient apparatus for the purification of water is that invented by Messrs. Archbutt and Deeley, and manufactured by Messrs. Mather and Platt, of Salford. Owing to the special method of manipulation adopted, by which the precipitate is caused to settle very rapidly, relatively large volumes of water can be softened and clarified in plain tanks of moderate size, and the necessity for filtering appliances is done away with. Two tanks, placed side by side, are generally used, the processes of filling, adding the chemicals, and allowing the precipitate to settle being carried on in one, whilst the softened and clarified water is being drawn off from the other, but for smaller quantities of water one softening tank and a small storage tank only are required. Fig. 148 shows an end view and a plan of one tank. A is a valve through which the tank is filled with hard water up to the dark dotted line. While the tank is filling, lime and alkali, in suitable proportions depending upon the character of the water, are boiled with water by a steam jet in the small chemical vessel on the platform. When the tank is full, and the valve A has been closed, steam from the boiler is admitted to the blower B, causing a current of water to circulate through the rose R, the three-way cock C, down the vertical pipe P, and back into the tank through the perforations in the *upper* row of horizontal pipes. On opening the small tap T, the prepared chemical solution is slowly drawn into the circulating current and thus diffused throughout the body of water in the tank. The cock D is next opened to admit air through the pipe at the top of the blower, and by reversing the three-way cock C, this air is forced through the perforations on the under side of the *lower* row of pipes. From these it rises in streams of bubbles, powerfully agitating the water, and stirring up the precipitate or mud from previous operations which lies on the bottom of the tank. Before being mixed with this mud, the new precipitate, which is very finely divided, will not coagulate, and takes a long time to settle; but, when the old precipitate is stirred up and mixed with it, the two together settle rapidly, and leave the water practically clear and ready for running off in from thirty minutes to one hour.

The chemicals, however, have been added to the *cold* water, and it is found that when such water has been softened and afterwards becomes raised in temperature in the hot pipes, injectors, &c., connected to boilers, a further slight precipitation occurs, which gradually forms a coating on the interior of the pipes, &c. To prevent this, the water is carbonated, which has the effect of rendering permanently soluble the trace of hardening matter retained by the water. The carbonic acid is supplied by a coke stove, containing enough fuel in the cone to last for several hours; and the gas from this stove is forced, by a small steam blower, into the upper end of the hinged pipe H, and is carried down with the water into the storage tank. Baffles fixed in the pipe cause a thorough mingling of the gas and water, and thus the carbonating is simply and easily effected. The upper end of the draw-off pipe H, where the gas enters, is bent over to form a syphon, and when the ball valve V closes, the bend fills with gas. It is not until the pressure of this gas is relieved by the opening of the ball valve that the water can pass into the pipe, and when the valve opens sufficiently to allow all the gas to pass, then the water falls down the pipe with it and is carbonated.

The precipitate is prevented from unduly accumulating in the tank by its partial removal at regular intervals; this is effected in various ways, to suit different circumstances. From the nature of the precipitate it is easily dealt with. In small plants, a discharge pipe, connected to a drain, is provided in one corner of the tank, and by lifting a plug daily, a sufficient quantity of mud can be run off. In larger plants, the mud is swept out, at longer intervals, through mud doors, into a trough which conveys it on to waste ground; or, it is raised out of the tank or trough by a steam lifter, and discharged into a cart lined with waste furnace ashes, through

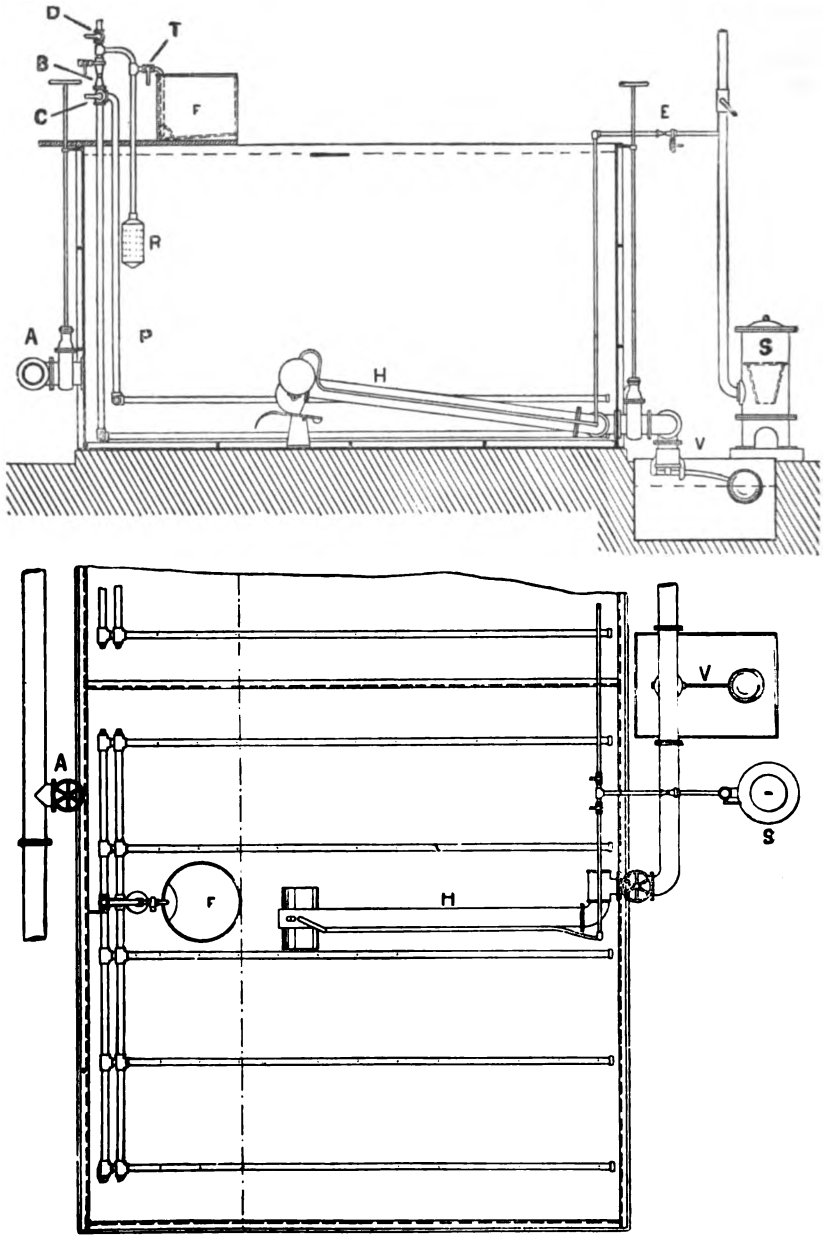


Fig. 148.—The Archbutt-Deeley Patent Water Purifier.

which the excess of water readily drains, leaving the nearly dry mud and ashes ready for tipping.

That steam boilers, and their feed apparatus and economiser tubes, can be kept quite free from scale by the above process of softening the water, has been amply proved at several works where it has been adopted. The largest quantity of water under treatment by it is at the locomotive works of the Midland Railway Company at Derby, where a plant softening 30,000 gallons per hour has been in successful operation since 1891. The hardest water at present (1895) being treated is at a mill in Nottingham; it has $35\frac{1}{2}$ degrees of hardness, and contains both carbonate and sulphate of lime, as well as a large quantity of magnesium salts; by the

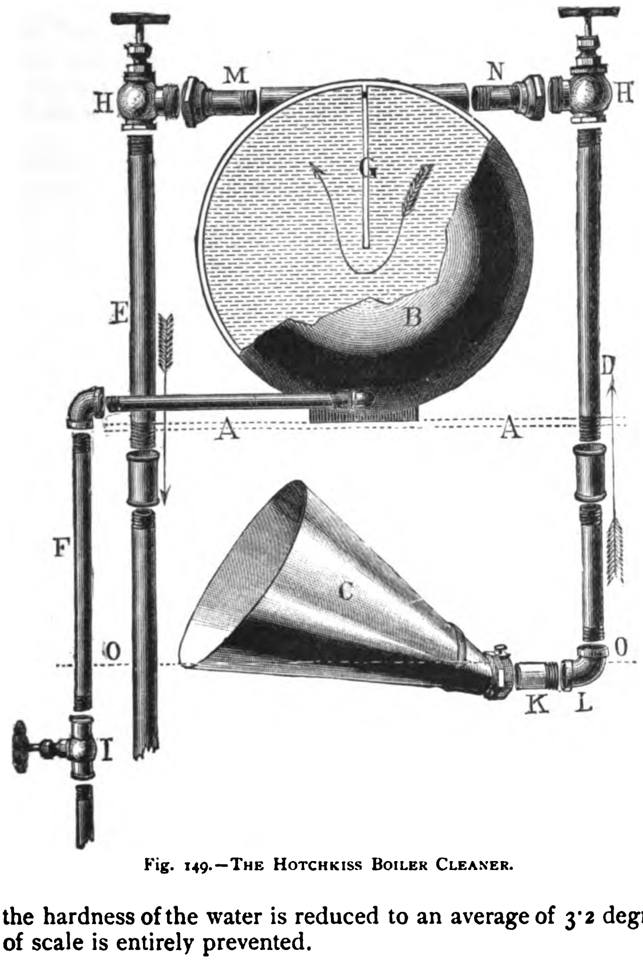


Fig. 149.—THE HOTCHKISS BOILER CLEANER.

treatment, the hardness of the water is reduced to an average of 3·2 degrees and the formation of scale is entirely prevented.

Where the feed-water is of a muddy description the *Hotchkiss boiler cleaner*, Fig. 149, as described by Mr. Horatio Nelson, 90, Worship Street, E.C., is said to be very effectual in removing scum and the floating deposits in the water.

In Fig. 149 the double dotted line AA is the top of boiler shell, which usually is found to be the most convenient position to place the reservoir. The reservoir is connected with the funnel C by the up-flow pipe D, and to a lower part of the water in the boiler by the return pipe E. Through these pipes and the reservoir a circulation is established which flows in the direction shown by the arrows. The funnel C is set within the boiler on the low-water line, as indicated by the dotted line OO. G is a diaphragm in the reservoir to divert the flow of water therein. F is the blow-off pipe, for removing the deposits B from the reservoir. HH are two valves, used to shut off reservoir from boiler if required. Angle valves, as shown, are generally used to simplify the connections. I is a valve on blow-off pipe F. K is a socket nipple, secured by thumb-screw. M and N are nipples, each with half-union to complete the connection.

This cleaner is automatic and, from natural causes, certain in its action.

The manner in which the "cleaner" acts in removing sediments from and preventing scale formation in steam boilers, is as follows:

As soon as the water in a steam boiler becomes heated, currents are established; these currents are formed by the hotter, and therefore lighter, water flowing upward and away from the source of the greatest heat, while the colder, more dense water flows to the source of heat, to replace the other, and in its turn becomes heated.

In all boilers where fire is applied at one end, the currents established will be upward and from the fire on the surface, and downwards and towards the fire in the lower part of the boiler.

In a boiler with the cleaner attached, the funnel C is set near the surface of the water, but partly submerged, and in such position that its opening will intercept the currents of hot water flowing towards it.

By the syphon action of the apparatus the hot surface water containing any floating matter that enters the funnel is caused to pass through the up-flow pipe D into the reservoir, thereby displacing the cooler water therein, which flows back to the boiler through the return pipe E, which terminates at a lower level than the funnel.

By this means, so long as firing is kept up, a constant circulation is maintained and all the water in the boiler is caused to pass through the reservoir. The water while in the reservoir is comparatively quiet, and entirely free from the agitating currents within the boiler, and while in this quiescent state the contained sedimentary matter B is precipitated, and remains in the reservoir, from which it can be blown as often as necessary by the pipe F.

The action of this apparatus creates a current in the boiler, thus utilizing the usual cold strata of water lying under the flues, and which is useless for creating steam.

Every one familiar with steam boilers is aware that deposits and incrustations seek naturally the quietest part of a boiler. The office of the Hotchkiss mechanical boiler cleaner is, therefore, simply to provide a place for their accumulation, outside of the boiler itself, and removed from heat and its agitating effects; from whence they can be readily removed as fast as they accumulate, instead of shutting down the boiler to clean them out by hand, or by blowing down the boiler in the ordinary way, thus losing a large amount of water already heated to the steaming point, which is wasteful as to fuel, and also a great loss of time.

It is not claimed for the "Hotchkiss mechanical boiler cleaner" that it will remove scale bodily from the boilers when the scale is already formed, but it is claimed for it, and *guaranteed* that it will prevent the formation of new scale by removing all the floating deposits and mineral salts, which become scale if not removed from the water before they have had time to adhere to the heating surfaces.

By preventing the formation of new scale, the old, by expansion and contraction of the heating surfaces, soon becomes loose and readily detached.

A simple means of purifying the feed-water before it enters the boiler is by *Seale's patent water purifier*, shown in Figs. 150–153, as applied to a smaller pipe, and returns through the annular space between the two pipes along its former course to near its starting-point, where, as shown by the arrows, it is discharged through slots into the boiler. By this means the feed-water becomes heated practically to the same temperature as the water in the boiler before it mixes with the latter. In its course through the pipes, as it reaches 212° F., carbonic acid will be driven from the water, and the carbonates of lime and magnesia be liberated. When a temperature of 300° F. is attained, the sulphates will also be liberated. The water is reckoned to reach this temperature just after it emerges from the open end of the internal pipe; and as the outer pipe is larger the velocity of flow is correspondingly reduced, so that the annular space between the two pipes acts as a settling tank, in which the particles of lime and magnesia are deposited, along with any other impurity the water may contain, as the water slowly travels to its exit.

The removal of the accumulated deposit is effected by means of a blow-off tap in front of the boiler, as shown on the drawings. This is used at intervals, the interior of the apparatus scoured, and the deposit thus prevented turning to a hard scale. The apparatus can be applied to any existing boiler.

In *Sanderson's patent feed-water purifier and heater*, advantage is taken of the fact that when water is heated beyond a certain point, it becomes incapable of longer holding the sulphates or carbonates of lime in solution, and further, if the salts are separated in the body of the water, they settle to the bottom of the boiler in a fine, floury state, easy of removal by blowing off or washing out, instead of being deposited in hard adherent scales by a process of slow chemical accretion on the surface of the plates. This is accomplished by introducing the feed-water into the steam space by means of a diffuser made of gun-metal. It is fixed near the crown of the shell, the feed-pipe being taken in the first place through the length of the boiler in the water space. By this means the water is partially heated, and on reaching the diffuser falls from it through the steam space in the form of a fine spray, and enters the body of the water in the boiler at practically the same temperature.

Boilers, for their proper preservation, should be placed under the periodical inspection of experts. An experienced inspector, skilled in the detection of the defects to which boilers of different types are liable, may detect early signs of deterioration, before an ordinary engineer would note anything amiss. These defects may then easily be remedied before much harm is done, and at a trifling cost. Any alterations that may from time to time become necessary or advisable, from the advancing age of boilers, such as a diminished working steam pressure, or unfitness for further work, will be reported by the expert.

Boilers are subject to explosion from a variety of causes. Defective material is a frequent cause. There may not be the necessary ductility or tensile strength in the plates, or the plates may have blisters in them.

Boiler plates whether of iron or steel are liable to blisters which are formed during the process of manufacture, but they proceed from different causes.

Those in the iron plates arise from pieces of slag or cinder getting between the layers whilst the manufacture proceeds. The slag is hammered and rolled with the iron, each process it goes through enlarging the superficial area over which it extends. Blisters weaken the plate, the weakening effect being greatest where the blister is thickest. They have been observed as large as 2 feet in diameter, and may be any smaller size. Where the piece of slag is very small, its effect may only be to cause a lamination in the plate, but where of serious size the blister may attain a thickness of $\frac{1}{4}$ of an inch.

exhaust-steam, or externally from leakages at the joints, may cause explosions, or the external corrosion may result from contact with damp brickwork at the seatings or elsewhere. The effect of corrosion is a wasting away of the plates, which become weaker and weaker as the corrosion continues. Internal corrosion follows no defined law. It may cause clear and well-defined pit-holes, or honey-combing more or less close in appearance, or a large extent of plate surface may be so evenly wasted as to defy detection unless the plates be drilled. Instances are recorded where corrosion has reduced the plates over the seating-walls to $\frac{1}{32}$ of an inch thick, and even less; if boilers, on examination, are found with such thin plates, it is easily seen how little force is necessary to rupture them.

Grooving, which has already been described, causes explosions.

Many boilers have doubtless exploded from old age.

Old second-hand boilers, bought cheap, which may have lain about and become rusted, are erected—very soon to prove an expensive bargain.

Badly-designed and ill-constructed boilers, even when made of good materials, cause explosions.

Collapsed flues frequently result from faulty construction, and explosions may follow from improperly fixed water-gauges or cocks, their position being too high or too low on the boiler.

The abuse which some boilers are subjected to when undergoing repairs may result in their explosion on being again set to work. A thoroughly well-designed and constructed boiler may be placed, for instance, in careless or ignorant hands for repair, or in those of a practical working boiler-maker whose practice is limited to making tight joints, he being unable to calculate the sizes of material necessary to resist strains or pressures. Again, in removing Galloway tubes from Galloway boilers, blank flanges are substituted instead of making some provision to replace the sustaining strength to the flue of which it was robbed, the result being a collapsed flue on re-starting. The various kinds of expansion-rings in the furnace-tubes get removed for repairs, and are replaced by flat belts which do not allow expansion and contraction to proceed. The gusset stays get removed from the flat ends of boilers so as to enable a leakage to be stopped, and nothing to compensate for the diminished strength is provided. A leaky plate in the shell may be removed, and one considerably thinner used in the repair, tending thereby to produce an explosion.

Injury to workmen engaged in cleaning boilers has often resulted from such dangerous practices as the following. To prevent annoyance from leaky stop-valves, whilst in the boilers, wooden plugs are sometimes driven into the holes of stop-valves, which are afterwards blown out and the attendant scalded to death. Again, with a group of boilers where the blow-off cocks are all connected with a common discharge-pipe, workmen have been known to open the blow-off cock of one boiler, whilst others were engaged cleaning the inside of the adjoining boiler; the consequence being that the steam and water passed from the one boiler to the other, scalding the men within it. No attempt should be made to plug holes during the cleaning of boilers, and some arrangement must be made whereby the water cannot be run from one boiler to another through the blow-off cocks.

Before cleaning, the boiler should always, where circumstances permit, be allowed to cool gradually. This may prevent accident, and also prevent the scum from adhering and baking to the plates, which frequently happens from blowing-off hot. In case it is impossible to allow time for gradual cooling, after the water has been run out, before breaking any joint about the boiler or removing the manhole cover, it should be carefully ascertained that all pressure has subsided. The pressure-gauge should be consulted, the safety-valve lifted, and the gauge-taps opened.

Rules for Heating and Grate Surfaces.

G = Fire-grate surface in square feet.
H.P. = Number of nominal horse-power.
h = Heating surface in square yards.

$$\text{H.P.} = \sqrt{h\,\text{G}}.$$

$$h = \frac{\text{H.P.}^2}{\text{G}}$$

$$\text{G} = \frac{\text{H.P.}^2}{h}$$

The maximum safe working pressure for well-made boiler shells may be found by the following rules:—

Iron.

Single riveted
$$p = \frac{12{,}500 \times t}{d} \qquad t = \frac{p \times d}{12{,}500} \qquad d = \frac{12{,}500 \times t}{p}$$

Double riveted
$$p = \frac{15{,}600 \times t}{d} \qquad t = \frac{p \times d}{15{,}600} \qquad d = \frac{15{,}600 \times t}{p}$$

where p = the pressure of steam in the boiler, t the thickness of plate in inches, and d the diameter of the shell in inches. For exceptionally riveted joints it will be better to take 5 tons as the working strength of the plate, and then by reference to the table given, the proportion of strength of joint to the solid plate can be calculated.

A steel boiler may be worked up to $\frac{1}{5}$th higher pressure than an iron boiler, and the following rules will apply for—

Steel.

Single riveted
$$p = \frac{15{,}000 \times t}{d} \qquad t = \frac{p \times d}{15{,}000} \qquad d = \frac{15{,}000 \times t}{p}$$

Double riveted
$$p = \frac{18{,}750 \times t}{d} \qquad t = \frac{p \times d}{18{,}750} \qquad d = \frac{18{,}750 \times t}{p}$$

Frequently the tensile strength of good boiler plates of iron is taken at 21 tons per square inch of section, and the working pressure of a boiler ascertained by

The cistern will then rise, carrying with it the small valve, and thus close the passage through the bent pipe, and remain in that position until the accumulation of water from condensation again fills the cistern, when the operation will be repeated.

Mr. John J. Royle, of Manchester, has devised a steam trap by means of which the condensed water may be returned automatically to the boiler. It is shown in Fig. 157. A is a steam chest on the top of the receiver B, having a three-port slide valve C—of which the port D communicates with the receiver B—the port E communicates, through a pipe not shown, with the drip box, not shown, and the port H with the atmosphere. The slide valve C is actuated by pistons I and J, working in open-ended cylinders affixed at each end of the steam chest A, as illustrated. The closed ends of the cylinder communicate, the one through the bent pipe with the inside of the receiver B through a valve actuated by the float M, and the other through a pipe communicating with the drip box through a valve also actuated by a float.

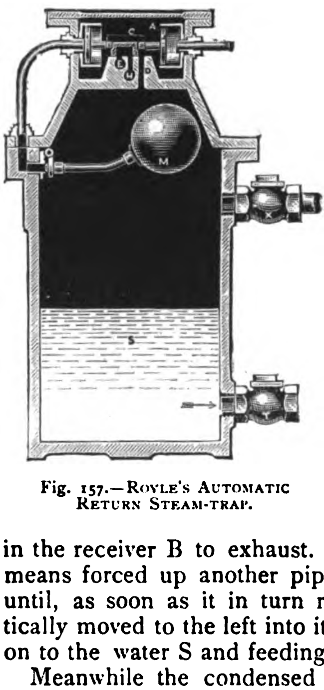


Fig. 157.—Royle's Automatic Return Steam-trap.

The fixing of the apparatus will be readily understood from Fig. 157, where the receiver B is shown fixed above, and arranged to feed back the condensed water to the boiler coming from a system of heating pipes and coils all draining into the drip box situated considerably below the boiler. Steam at the full boiler pressure is supplied through a pipe to the steam chest A, and entering through the open port D presses upon the water S shown in the receiver B, and establishes an equilibrium between the boiler and the receiver, allowing the water to gravitate through check valve T and pipe continued from it to the boiler. Meanwhile the condensed water from the heating pipes, &c., has been entering the drip box through a check valve, and as soon as the water in the drip box reaches the ball of float and lifts the valve connected with it, the equilibrium of the pistons I and J is destroyed and the slide-valve C caused to travel to the right, so reversing the position of the ports, admitting steam at boiler pressure through the pipe leading into the drip box, and allowing the steam contained in the receiver B to exhaust. The condensed water in the drip box is by this means forced up another pipe and through check valve X into the receiver B, until, as soon as it in turn reaches the float M, the slide-valve C is automatically moved to the left into its former position, so admitting full steam pressure on to the water S and feeding it to the boiler as before described.

Meanwhile the condensed water accumulates in the drip box, and as soon as it again lifts the ball of float there, the same action is repeated, and so on continuously.

The special features claimed for this steam trap are:—1. It will automatically elevate boiling water from any distance below the boiler, and feed it into the boiler against any pressure, without requiring any back pressure on the drip pipes—a feature unique, and possessed by no other return trap. 2. Heating pipes can therefore be worked at any steam pressure, and at any distance above or below the boiler. 3. As a boiler feeder this apparatus possesses special advantages for feeding boiling water lying either above or below the boiler level, and which would be impossible to feed back to the boiler by other means.

intended pressure in pounds. These weights keep the valve in its seat until the steam attains a certain pressure, when the steam lifts the valve and escapes through the opening at the top of the weights.

Johnston's patent self-acting alarm whistle guards against accidents from deficiency of water in the boiler. A hollow cast-iron float is made sufficiently heavy that on falling with the water in the boiler it opens an orifice through which the steam rushes, thereby causing the alarm whistle at the top to be sounded. The apparatus is free of all stuffing boxes, glands, cocks, or any complicated contrivances. As long as there is sufficient water in the boiler, the alarm valve is kept close against its seat by the float.

Smith's steam sentinel is an invention to prevent over-pressure in steam boilers and is a check on the safety valve, because it gives a distinct and unmistakable warning immediately the maximum pressure is exceeded. Its construction is simple. A conical valve stops a hole in the boiler, and is kept down by a spring carefully adjusted to resist the pressure of the steam up to a certain point. As soon as this pressure is exceeded, the valve is liberated by the compression of the spring, and a communication is opened for the steam into a whistle of the ordinary form, which gives a loud warning of approaching danger.

Question 29.—In a lever safety valve, the whole length of the lever is 32 inches, the distance between the fulcrum and the valve 4 inches, the diameter of the valve $2\frac{1}{2}$ inches, required what weight must be put on at the end of the lever so as to have a pressure of 50 lbs. per square inch upon the valve; also to divide the lever so as to have 40, 30, and 20 lbs. upon the valve with the same weight?

The area of the valve will be $2{\cdot}5^2 \times {\cdot}7854 = 4{\cdot}9$.

The leverage will be $\frac{32}{4} = 8$, so for a 50-lb. pressure we have $\frac{4{\cdot}9 \times 50}{8} =$ $30\frac{5}{8}$ lbs. as the weight to be put on the end of the lever to give 50 lbs. per square inch. And $\frac{4{\cdot}9 \times 40}{30\frac{5}{8}} = 6{\cdot}4$; and $6{\cdot}4 \times 4 = 25{\cdot}6$ inches, the distance from the fulcrum at which the weight must be put to have a pressure on the valve of 40 lbs. Similarly $\frac{4{\cdot}9 \times 30}{30\frac{5}{8}} = 4{\cdot}8$ and $4{\cdot}8 \times 4 = 19{\cdot}2$ inches, the distance from the fulcrum at which the weight must be put to have a pressure on the valve of 30 lbs. Again, $\frac{4{\cdot}9 \times 20}{30\frac{5}{8}} = 3{\cdot}2$ and $3{\cdot}2 \times 4 = 12{\cdot}8$ inches, the distance from the fulcrum the weight must be put to have a pressure on the valve of 20 lbs. The weight, it will be thus seen, must be moved towards the fulcrum $(32 - 25{\cdot}6)$ $6{\cdot}4$ inches for every 10 lbs. taken off the pressure on the valve.

Question 30.—What is the pressure per square inch in a boiler, the whole length of the lever being 32 inches, the distance between the fulcrum and valve 4 inches, the diameter of the valve being $2\frac{1}{2}$ inches, a weight of $30\frac{5}{8}$ lbs. being placed at the end of the lever?

Here we have $\frac{32}{4} = 8$ for the leverage, therefore the whole weight on the valve is $30\frac{5}{8} \times 8 = 245$ lbs., and $\frac{245}{2{\cdot}5^2 \times {\cdot}7854} = 50$ lbs. as the pressure per square inch in the boiler.

Question 31.—If the safety valve is $4\frac{3}{4}$ inches in diameter, and the lever is 38 inches long to the centre of the weight, and $4\frac{1}{2}$ inches from the fulcrum

Question 38.—What size of steam pipe would you lay from the receiver to a 30-inch cylinder engine?

The maximum velocity of the steam in the main steam pipe should not exceed 100 feet per second. The piston speed of the engine is not given, but taking it at 300 feet per minute, the size of pipe for a single cylinder engine would be found thus:—A 30-inch diameter cylinder has an area of 706·86, therefore $\frac{300 \times 706{\cdot}86}{100 \times 60} = 35{\cdot}343$ area in inches, and $\sqrt{\frac{35{\cdot}343}{{\cdot}7854}} = 6{\cdot}7$ inches, or say $6\frac{3}{4}$ inches in diameter. With a double-cylindered engine it would be $\frac{300 \times 706{\cdot}86 \times 2}{100 \times 60}$ $= 70{\cdot}686$ area in inches, and $\sqrt{\frac{70{\cdot}686}{{\cdot}7854}} = 9{\cdot}487$ inches, or say $9\frac{1}{2}$ inches in diameter. If the engine is a compound one, only the size of the high-pressure cylinder need be considered in estimating the size of the steam pipe.

Question 39.—In a colliery where engines were working, and having 1,800 horse-power, what boilers would you consider necessary?

$\frac{1800}{200} = 9 + 1 = 10$ Lancashire boilers 28 feet long and 7 feet 6 inches diameter.

Question 40.—What height and size of chimney would you construct for such range of boilers?

$10 \times 5 = 50$ feet area = say 8 feet diameter at the base, and to be well proportioned the chimney should be 25 times the diameter in height $= 8 \times 25 = 200$ feet high.

Question 41.—What is the nominal horse-power of a Lancashire boiler whose length is 30 feet and diameter 6 feet?

$\frac{30 \times 6}{6} = 30$ horse-power.

Question 42.—Are boilers better calculated to resist pressure lengthwise or crosswise, and in what proportion?

A common cylindrical boiler is twice as strong to resist pressures acting lengthwise than to resist pressures crosswise, and for this reason all the horizontal seams should be double riveted.

Question 43.—How much stronger are boilers having double-riveted plates than those having single-riveted plates?

Double riveting weakens the plates about one-fourth, single riveting about one-half, therefore double-riveted boilers are stronger in the proportion of about 3 to 2.

Question 44.—Find the bursting strength of an iron cylindrical boiler 6 feet in diameter, and made of $\frac{1}{2}$-inch plates, with doubled-riveted joints.

$$\frac{{\cdot}5 \times 21 \times 2240 \times {\cdot}69}{6 \times 12} = 225{\cdot}4 \text{ lbs. per square inch.}$$

At most collieries railways are required, but the circumstances of each must decide what railways shall be made. Siding room for full and empty trucks

CHAPTER VII.

TIMBERING AND WALLING.

The kind of Timber used at Collieries—Storing it Underground—Method of fixing Props and Lids—Temporary Props and Lids—"Dog" for drawing Props—"Sets" of Timber and their fixing in Main Roadways—Timber for Collars—Sills under Props—Timbering for a bad Roof, where the Floor and Sides are good—Lagging—Timbering for a bad Roof and Side, the other Side and Floor being good—Timbering for a bad Roof and Sides, with a good Floor—Timbering for a bad Roof, Floor and Sides - Lagging of Trees and Brushwood—Sizes of Timbers and their distance apart—"Cogs" or "Chocks"—Methods of Timbering in France—Notching the Timber—Cast-Iron Props—Wrought-Iron and Steel Supports—Storing the Timber on the Surface—Creosoting as a means of preserving Timber from Decay—Customs as to Setting and Drawing the Timber—Walling the Main Roads from the Shaft—Material used in Walling—Semi-circular arched Roadway—Invert under Side Walls—"Horse-shoe" Arch—Elliptical Arch for Roadway—Process of building Arches—Necessity of removing all Timber, and tightly packing behind the Walls of Arches—Packing the Top and Sides with Sand.

TIMBERING is the cheapest way of securing roads, regard being had to first cost only; but if the roads are used a number of years, and the cost of maintenance is taken into account, walling may be a much better and cheaper plan.

The timber used at collieries to support the roof and sides is chiefly pine, fir, and oak. The sizes vary from 4 to 12 inches in diameter, the size and arrangement depending upon the material to be supported and the excavation itself. Where used to support the roof only, the timber requires very little preparation. It is cut into suitable lengths at the surface, sent into the pit, and, in accordance with the Mines Act, 1887, for the convenience of the workmen who have to timber the working places, a proper supply must be kept stored near at hand.

The usual manner of supporting the roof in the working places is by means of single props (called "posts," "trees," &c.), having short lids or caps (in Somersetshire called "traps") on the top. If these props are cut from the tops of old trees, they are spongy in texture, and less durable than when cut from the lower portions of young trees. The bark should always be left on them, as it helps to preserve the wood.

In fixing props, the workman with one hand holds the lid under the roof requiring support, whilst with the other he moves the top of the prop, the bottom of which rests on the floor, until it touches the lid, which is firmly held by the post, while the latter is driven well under the lid by means of a sledge-hammer. The post should be upright if the seam lies flat; if not, the prop will not be upright, but at right angles to the floor and roof, or, as the roof will sink a little notwithstanding the prop, the latter may be set in a direction which deviates slightly from the perpendicular between floor and roof towards the vertical. A single prop and lid is sometimes though not often fixed in the main roadways as well as the working places. Fig. 161 shows a single prop and lid. At times it is required to fix a single prop and lid for a temporary or passing purpose.

For instance, where a double row of "chocks" is used in Longwall workings, as a protection to a continually advancing face, the back row is often taken down and re-fixed in front of that which was previously the front row. Each of these chocks good floor. Here "sets" of timbers are placed as before described, and lagging placed next the roof and against both sides. Flat, half round, or round lagging may be used, according to the pressure it is likely to have to resist, vertically, or laterally.

It often happens that the roof, floor and sides are all weak and require support. Where this is the case, if the roadway is to be maintained for any length of time, walling will be the most effectual and economical way of preserving the road. But if only required for a short time, it may be timbered as shown in Fig. 173. In this case, the "sets" of timber have placed under them pieces similar to the collars, and these may be of half-round pieces if the pressure is not great. Where the half-round pieces are not strong enough, whole pieces must be used; and again, where these do not effectually resist the pressure, lagging must be placed under them, similar to the lagging at the sides and roof as shown in Fig. 173.

In the southern portion of the Somerset coalfield, and also in some of the Pembrokeshire mines, where the roads are very difficult to keep open, branches of trees and brushwood are frequently used as lagging. These form a network against the sides and roof and distribute the weight more evenly over the supports than ordinary lagging.

The distance between the sets of timber or the collars must depend upon the state of the strata. In some cases they may be placed only a few inches apart, the road being quite lined with them, or in others at intervals of 3 feet or upwards.

Where the timbers are not of a uniform size along a roadway, the larger and smaller should be made to alternate, so that a weak pair may come between two strong ones, but this method is not desirable.

The diameters of the timbers should increase with their lengths, so that those cut for a high or wide road must be thicker than those used in roadways of smaller dimensions.

In timbering Longwall workings, or the pillar workings of Post and Stall, besides the props cut the height of the seam with lids placed over them of about 15 inches long, "cogs" or "chocks" are used. No drawing is here given of these, as many examples are shown in the next chapter. They are pieces of timber about 2 feet long and from 6 to 12 inches square. If it is intended to recover them, a little rubbish is laid on the floor, and two of these timbers are laid on this parallel to each other and about 18 inches apart. Two similar pieces are then placed on these crosswise, also 18 inches apart, parallel to each other, and at right angles to the two first laid. Two more similar pieces are placed over the last in a line with the first two laid, and so on till the roof is reached, where they are wedged with pieces of chip.

Chocks which are to be taken out and used over and over again are generally made of hard wood, such as oak, elm, or ash, but in cases where no attempt is made to recover them ordinary soft round pit timber is used with the bark on, the latter being often 4 or 6-foot timbers. If intended to be left in, the space between them is filled with rubbish as they are built up.

Figs. 174 and 175 show a form of timbering employed in France, as described by André in his *Treatise on Coal Mining*, the chief features of which are the struts supporting the props and the collar at the point where they have a tendency to give way, and also the use of longitudinal pieces to bind the different sets together.

Fig. 174 shows the arrangement for a road having a single, and Fig. 175 that for a road having a double line of rails. The operation of fixing the bracing pieces inside the other usual set of timber is as follows:—

The longitudinal immediately under the collar is placed and temporarily held walling, if even the road has to be timbered for a year or two before the walling is built. Sometimes, in working thick seams, a bad roof over the coal prevents roads being carried immediately under it. In this case the top bed of coal is often allowed to remain, thus affording a better roof than that over the coal.

Wherever timber is placed in damp situations it is affected injuriously by the watery vapour in the atmosphere. Some kinds of timber decay more rapidly than others where so exposed. English larch stands a considerable time in damp places, but any timber used in wet roadways may be rendered twice as durable by "creosoting." 100 parts of coal tar contain, when distilled, 65 parts of pitch, 20 of essential oil (creosote), 10 of naphtha, and 5 of ammonia. The oil produced from this distillation is used for creosoting timber. It prevents the absorption of moisture in any form, under any temperature. It is noxious to animal and vegetable life, repelling the attacks of insects, and preventing the propagation of fungi. The oil is injected at a temperature of 120° F. under a pressure of 150 lbs. per square inch, so that ordinary fir timber absorbs about 8 or 10 lbs. weight of creosote per cubic foot.

In return airways which are damp and warm ordinary timber soon rots, and if its use cannot be dispensed with in these situations, it should be creosoted before being placed there.

In many districts props and sprags are used at the working faces to prevent the coal from falling on the workmen whilst engaged at their work. As the system of propping the coal is inseparably connected with the mode of working, which again is regulated by considerations respecting the seam, we reserve our remarks on these subjects until the next chapter, when the systems of propping and spragging the coal will be dealt with, and many examples shown of what is being actually done.

In Northumberland and Durham skilled workmen called deputies are employed. The duties of a deputy for the most part consist in setting and drawing timber in any district over which he has charge. Besides the timbering, he lays the rails where required, and takes up any from recently abandoned roads, and also attends to the bratticing, ventilating doors, &c. Usually he has about a dozen men to attend upon in his district.

In the counties mentioned the workmen at the face are relieved of the responsibility of propping the roof, which responsibility devolves on the deputies, who are officials acting on behalf of the owners, and in these districts, where the roofs are fairly good, this plan seems to work well.

In Derbyshire and Staffordshire a kind of Butty system is in vogue, in which stallmen have charge of the Longwall faces, receiving a tonnage or contract price on the coal sent out, and these stallmen employ workmen to hole and take down the coal, whilst they or others employed by them put in the props and build the pack-walls. A stallman has from 8 to 16 men under him.

The usual arrangement is for the miners or persons working at the face to set all the timber required there for their own protection, and a great deal may be said in favour of the arrangement, but it should always be under supervision from the officials, and subject to some kind of regulation as to the extreme distance allowed between props, &c., according to the requirements of the case.

The timbering on the main roads is done by timbermen appointed for the purpose.

Walling.—The roadways forming the main underground arteries are frequently walled. Sometimes the walling is continuous over long distances, at others where the road has passed here and there through weak beds the walling is in short is kept in advance of the side walls and the arching, and centering is used in building it.

The side walls are built on the invert and are kept in advance of the arching, so that the masonry may be said to proceed in three sections. A staging is erected from which the masons build the arch over the side walls. If no invert is

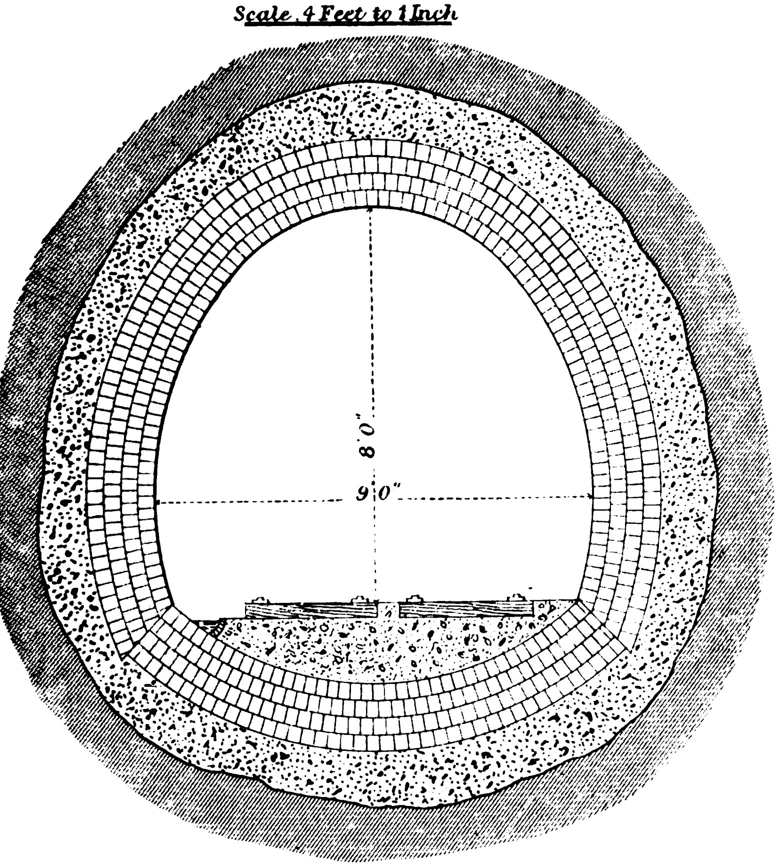


Fig. 185.—Arching for Underground Roadway where the Sides and Roof form part of an Ellipse, and the Floor an "Invert" or Flat Segment Arch.

required, the masonry proceeds in two sections, the side walls being kept in advance of the arching.

Iron centres instead of the usual wooden ones are used for turning the invert and arch. Blocks are fixed on the floor to take those for the invert. The centres for the arching may rest on blocks slightly projecting from the side walls after they are built.

All old timber should be taken out if possible, and the space behind the walls and over the arch should be tightly packed with rough concrete, or any suitable material. Timber left behind the masonry would rot in time and leave spaces between the masonry and the rock. If the masonry is proceeding in a road to which accumulations of fire-damp may possibly extend, care must be taken to prevent open lights being taken above the arch, even if these are permitted in the roadway.

Roads requiring to be walled, as shown in Fig. 183, may have the necessary

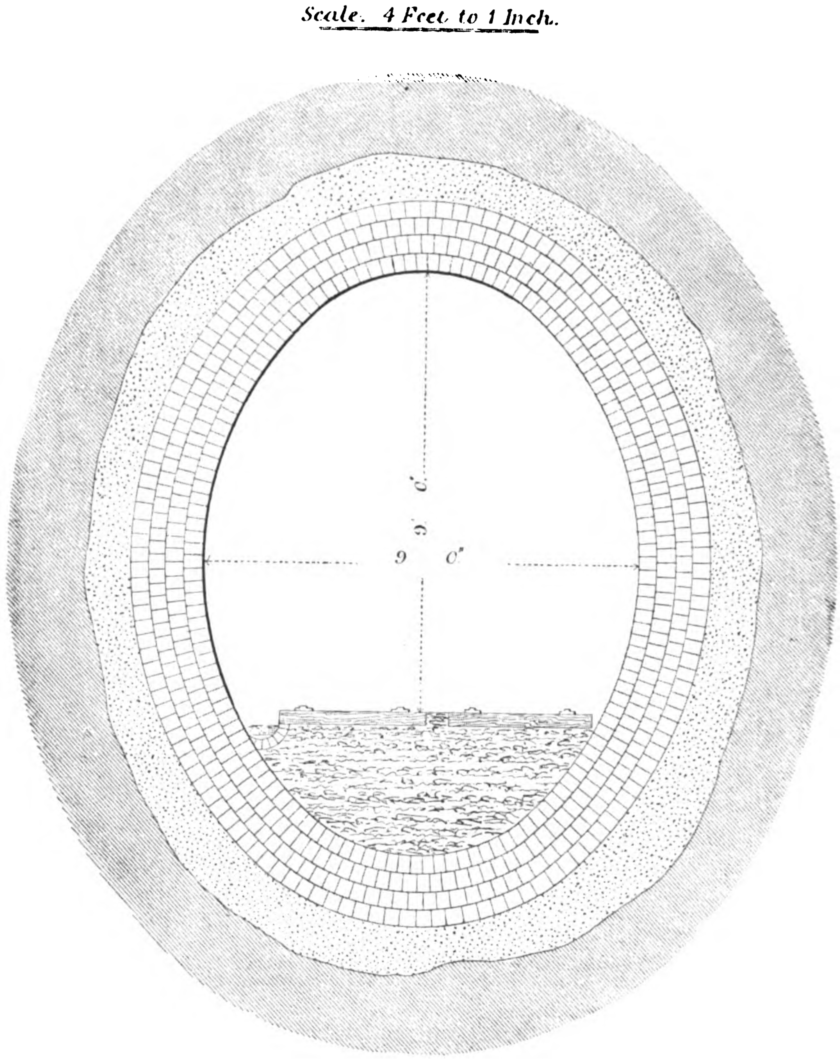


Fig. 186.—Section of Elliptically Arched Roadway, having One Foot of Sand interposed between the Arching and Stratification, and showing a Double Line of Rails laid, and Gutter formed in it.

material taken from the sides and roof, to admit of the masonry being put in during the day-time, without interfering with the usual traffic (if that be limited); so also, by using the iron centerings which do not obstruct the roadway so much as the ordinary wooden ones do, the masons may put in the walling whilst the usual work of the colliery proceeds. But if the quantity of coal passing along the road is large, it will be better to arrange that the work be done at night. Where an invert to the arch is required, the work must be carried on during the night unless the road is free from traffic.

It is better in all these operations to use hydraulic mortar, because it sets so quickly: some hydraulic mortars become solid in a quarter of an hour either in the air or under water.

Where the depth from the surface, and consequently the crush, is great, arches however strongly built are often destroyed by its force. It has been found that by packing the top and sides with sand, to a thickness of not less than one foot, the weight is distributed over the whole surface of the arch, and the walling has remained intact. The thickness of masonry required to resist a given pressure is less if packed firmly behind with sand than would be necessary if no packing be used. In the drawings Figs. 183 to 186, the different forms of arch are shown with the packing of sand filled in behind.

CHAPTER VIII.

NARROW WORK AND METHODS OF WORKING.

Shaft Pillars—Water-Levels—Cross-measure Drifts from Shafts sunk through inclined strata—Stone Drifts through faults—Longwall Method of Working—Post and Stall System—Different Arrangements of Single Road Stall Working—Double Road Stall Method and its Modifications—Method of Working and Timbering adopted at the following Collieries:—Celynen, Risca, and the Ocean—Wicket System of North Wales—The Bank System of South Yorkshire—Method of Working and Timbering adopted at the following Collieries: —Lundhill, Kiveton Park, High Park, Wearmouth, Silksworth, Florence, Great Fenton, Cannock and Rugely, Pemberton, Clifton Hall, Pendlebury, Sovereign, Radstock, Kingswood, Allanshaw, Cowdenbeath—Working thin seams in Northern France and Belgium—Square-work Working of the Staffordshire thick coal seam—Working the thick coal seams of Poland, Upper Silesia, and Bohemia—Dealing with excessively thick coal seams by Longwall and Post and Stall—Questions and Answers bearing on the subjects of the Chapter.

After the shafts have been sunk, drivings will be necessary to win the coal, and one of the first things to consider is the size of pillar or pillars to be left for the support of the shaft. If no pillar were left, but a longwall face opened at once from the shaft on either side of it, the subsidence of the roof, except in very thin seams, consequent on such proceeding would disturb the strata near the pit, and might cause injury to the shaft-walling, displace the shaft-fittings, and entail a considerable after-expense in restoring the shaft to a working condition. The size of the shaft pillar or pillars should be such that, when the coal is worked away beyond a sufficient area round, the shaft will be unaffected by the "draw"—the lateral disturbance of the strata beyond the point actually worked. The depth from the surface, the nature of the strata above and below the coal seam, as well as that of the coal itself, and the amount of dip all influence this. For any depth to 100 yards, it may be sufficient to leave a pillar 40 yards square. Adopting this size as a minimum we may fix any size of pillar for greater depths by increasing the pillar 5 yards for every 20 yards in depth, so that for a shaft 150 yards deep, we should require a pillar $52\frac{1}{2}$ yards square, for a shaft 200 yards deep, 65 yards square, for a shaft 300 yards deep, 90 yards square, for a shaft 400 yards deep, 115 yards square, and so on.

The shaft should always be in the centre of the pillar or pillars left for its support, to ensure the same amount of protection on each side.

If water is likely to be met with, water-levels will be required, and these should be started some feet below the seam at the pit bottom. Roadways in the seam to be "water-level" should rise slightly, about $\frac{3}{16}$ths of an inch per yard, to allow the water to flow out to the shaft. For the purpose of ventilation, the two shafts, which are necessary to every colliery, are also connected as soon as possible by driving in the seam. Levels are usually driven on both sides of the shaft, and there may be either two or three on each side, driven parallel to one another, and about 20 or 25 yards apart. They are generally driven from 7 to 10 feet wide; if the roof be very bad, it may be desirable to make them as narrow as 5 feet. With regard to height, if the seam is thicker than 7 or 8 feet, the level is usually carried that height, and the upper portion of coal left as the roof. If the seam is less than 5 feet high, the roof is ripped down or the bottom cut to

These double communications are necessary for an intake and return air-way, and when made, the working of the No. 1 seam may be resumed inside the fault.

Although only these examples are here given, the student will readily imagine others in which the seams are cut off or interfered with by faults of different magnitudes.

The usual systems of working are what are termed "Longwall" and "Post and Stall," sometimes called "Bord and Pillar," and in Scotland "Stoop and Room." Some systems of working are practised which are modifications of these, and seams of exceptional thickness are often worked by a special method applicable only to the particular circumstances prevailing. The circumstances most favourable to Longwall are, a seam, not too thick, of rather hard coal, capable of bearing pressure, and which parts freely from the roof; a seam of coal having stone bands in it or ironstone over it to be worked with the coal, yielding material for packing. The circumstances most favourable to Post and Stall are, seams situated near the surface, the working of which on the Longwall would probably injure the buildings, but which may be worked on the Post and Stall in the whole mine, leaving the pillars, more or less robbed, to support the surface; this system is preferable also where the coal is tender under a heavy roof.

The advantages of working the Longwall where it is applicable are, a better yield of large coal, less injury to upper seams as the intermediate strata settle gradually, simplicity of working, ease of ventilating, and greater economy, for the superincumbent weight reduces the labour of "holing." These advantages are so manifest as to indicate the desirability of working all seams of usual thickness situated 100 fathoms or more below the surface on the Longwall system. There are many modifications of Longwall, and this is one of its merits; it is capable of being varied more readily than the Post and Stall to suit local circumstances. In all cases it consists of extracting all the coal at one operation, the roof settling down behind as the "face" advances. In practice it is generally found better to take out all the coal with the exception of the shaft pillars, but sometimes pillars are left between, and on each side of, the main roads. There is an advantage in letting the face advance across the cleavage of the coal, but some coals have no defined cleavage, and sometimes, even where there is a cleavage, the dip is not suitable for the face to advance across it. In Fig. 190 the gob roads are carried to the rise and across the cleavage of the coal. The distance of these gob roads apart varies, being seldom under 14 yards or over 50 yards. The roads are carried in the middle of the stall, for convenience in working the coal and in bringing it from both sides as shown on the plan, Fig. 190. The pack-walls are built as shown, and if the seam be thin, height is made by ripping either top or bottom. The gob must be packed with the rubbish yielded by the seam in being worked and by the rippings, and the closer it is packed the better will be the result. This is especially necessary in fiery seams, for it must be remembered that any portions of the waste not closely gobbed afterwards become receptacles for fire-damp, unless ventilated. At times, when the barometer is low, a portion of this gas finds its way into the roads, and is at all times a source of dread and anxiety. If the roof is bad a double row, and sometimes a treble row, of props, with lids placed over them, is kept next the face, the back ones being taken out, where this can be done with comparative safety, and re-set in front as the face advances. In some cases a double row of chocks is used instead of props. Often the water levels from the shaft, instead of running in a line with the cleavage, as shown in Fig. 190, will cross it at some angle; if this be a right angle and it is desirable to carry the "face" across the cleavage, Fig. 191 shows the method usually adopted.

The face thus advances against the cleavage, and if the dip and rise be rather great there is an advantage in keeping the gob road at or near one end of the

Scale. 3 Chains to 1 Inch.

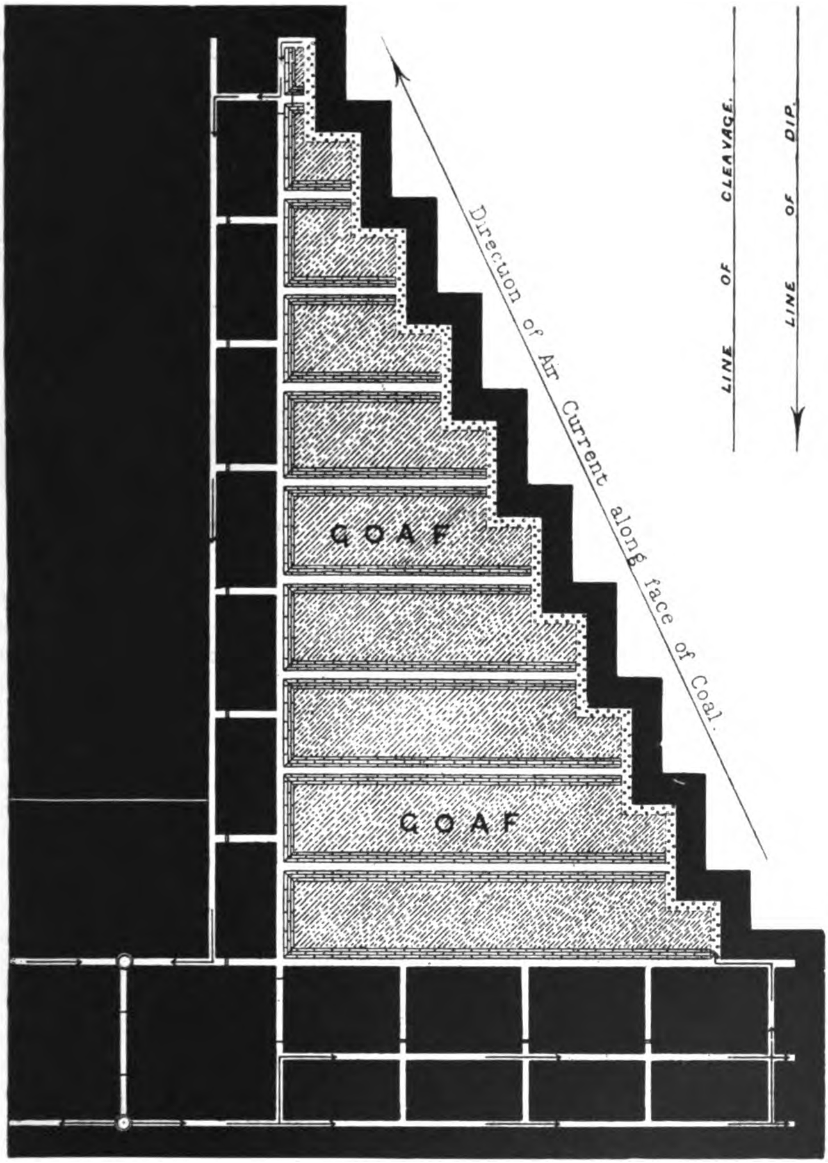


Fig. 191.—PLAN SHOWING LONGWALL WORKINGS WITH GOB ROADS ADVANCING LEVEL COURSE AND ACROSS THE CLEAVAGE.

working face instead of in the middle, so that the coal may be brought "down hill" to the road. In very tender seams, there is an advantage in working the face *with* the cleavage, instead of *against* it, and frequently, where there is no cleavage in the coal, the face, instead of being marked out in steps, is connected

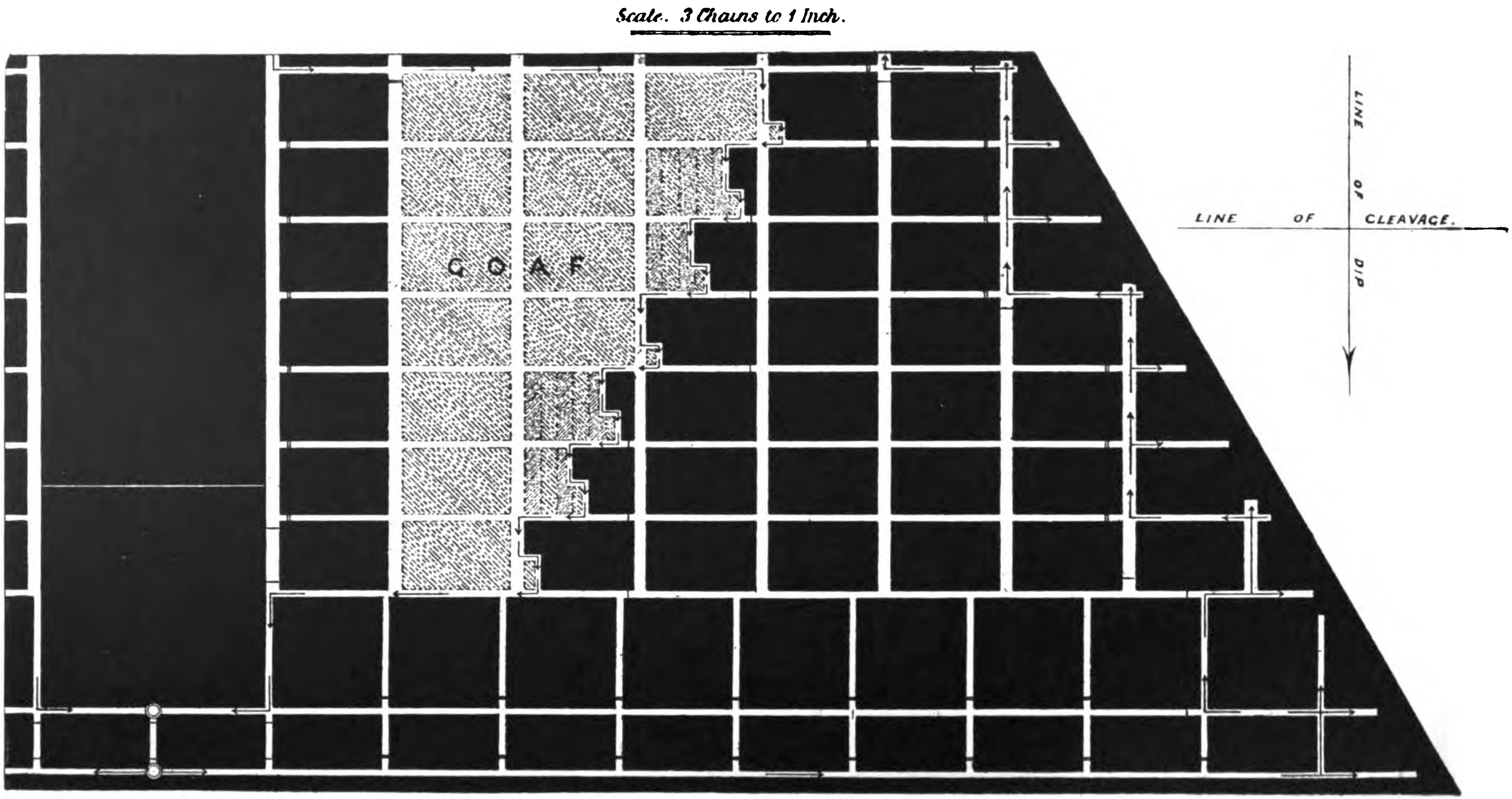


Fig. 193.—Post and Stall system of Working where the Levels are proceeding in the direction of the Cleavage.

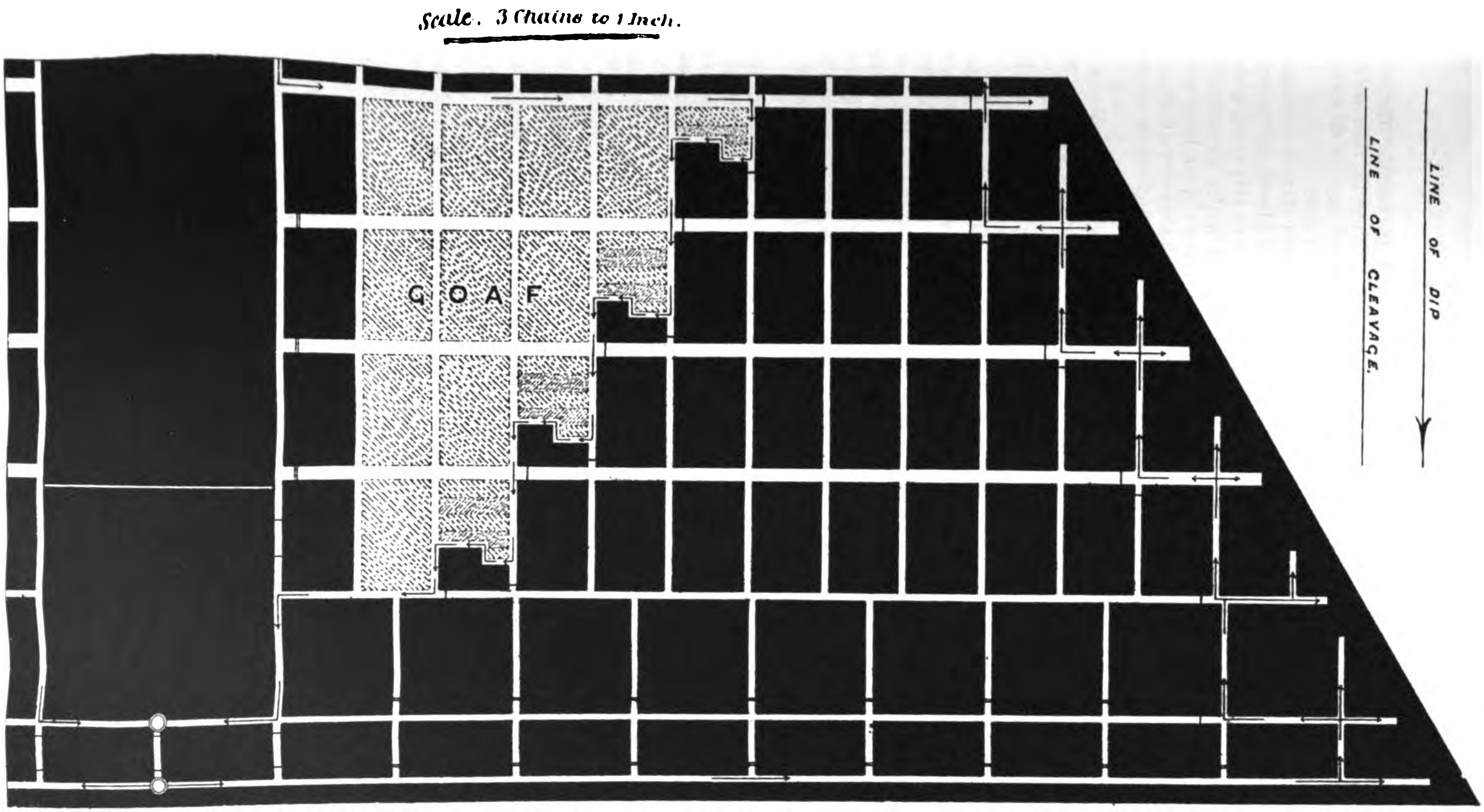


Fig. 194.—Post and Stall System of Working where the Levels are proceeding across the Planes of Cleavage.

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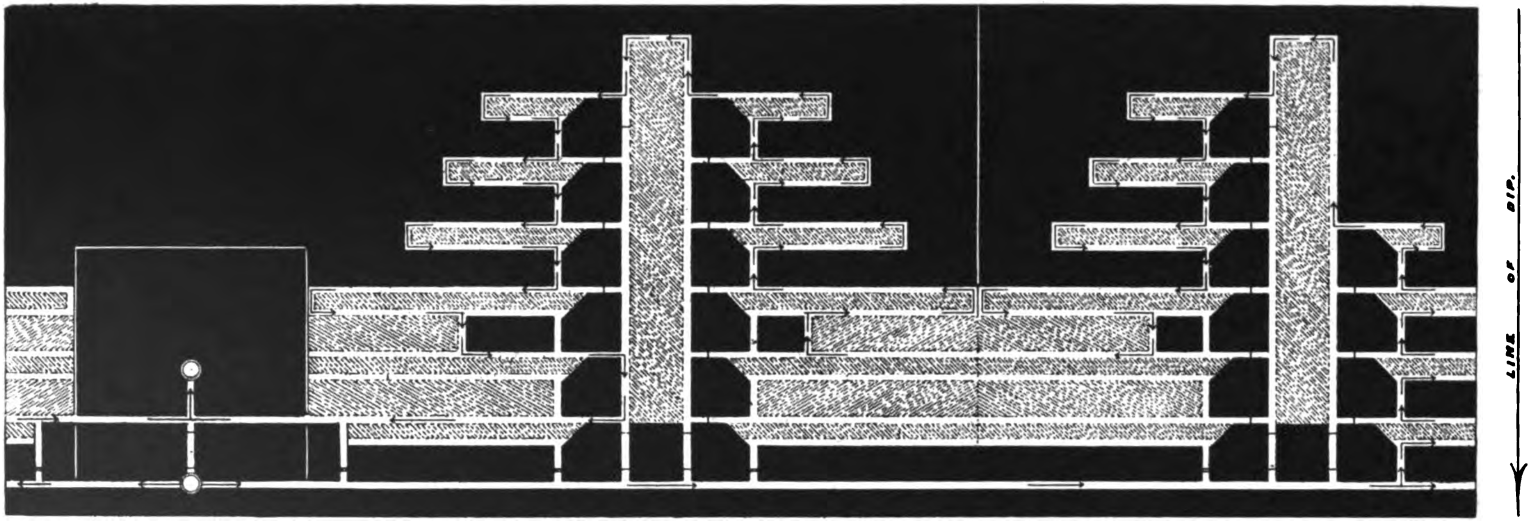


Fig. 196.—Arrangement of the Single Road Stall System of Working Coal in South Wales, where the Roof is very Good.

pillar, some advantage may result from taking, say, 14 or 15 yards from the rise side of the road and 8 or 9 yards from the low side.

One man and a boy usually work together in a single stall road, the man

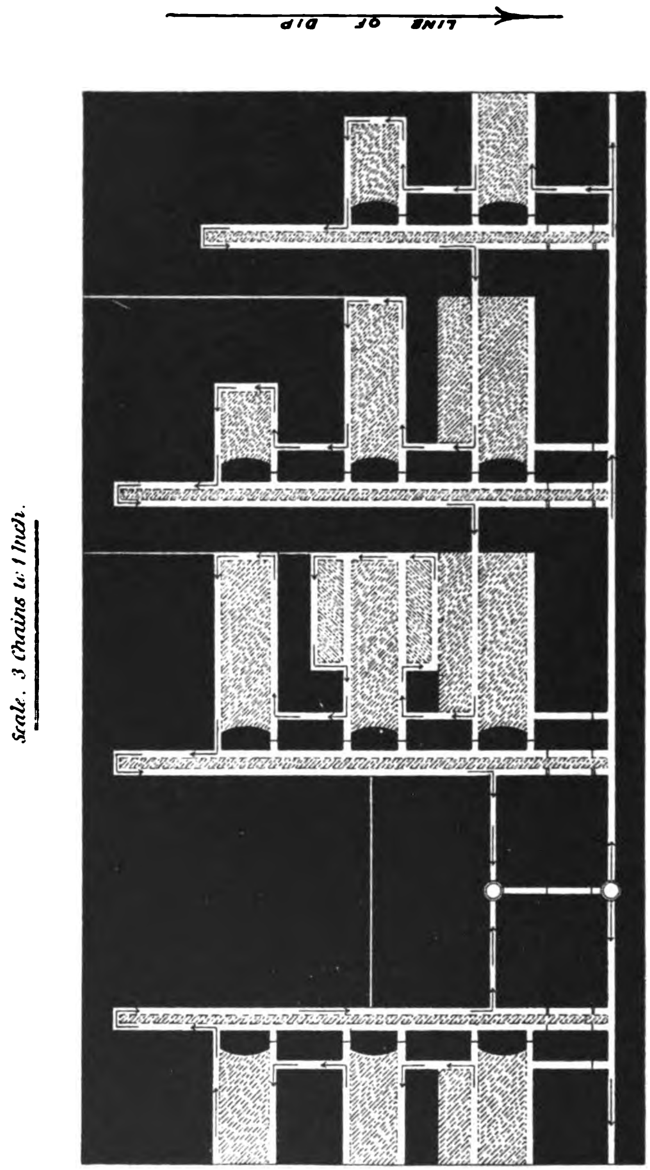


Fig. 197.—Plan showing the Double Stall System of Working Coal in South Wales.

receiving a tonnage price which includes timbering at the face, tramming the coal, and making the road. He pays the boy a daily wage, chiefly to fill the trams for him.

A reference to Fig. 197, and a comparison between it and Figs. 195 and proposed in 1881 to sink pits from this thin seam to the goaves and so drain off the gas.

The stalls in the next pair of headings are driven in exactly the same way as the last, so that when the headings are finished and stowed up, there is a rib of coal 5 yards in width separating the goaf formed in one pair of headings from that formed in another. This rib is lost. Pairs of levels, parallel to the shaft levels, and on the rise side of them, are driven at intervals of 315 yards, so as to

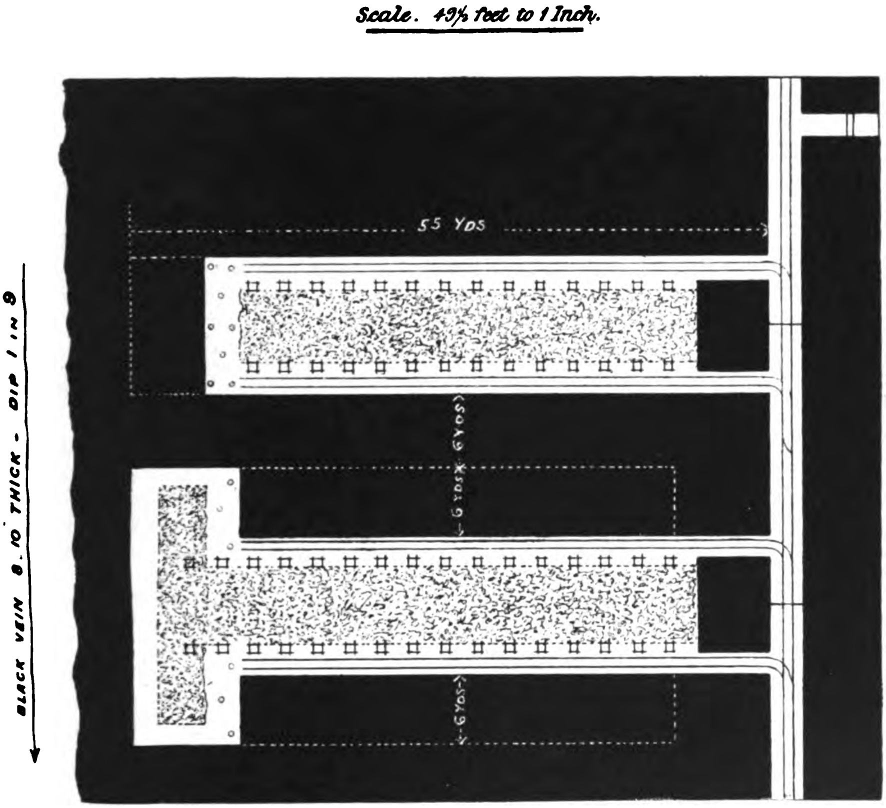


Fig. 200.—Arrangement of the Broken in the Double Stall Workings on the Black Vein at Celynen Colliery.

leave a barrier of coal 40 yards thick between the upper end of the headings and the next level above. The same system of headings and stalls is driven from the other levels above. After the levels have reached the boundary and the coal is worked out from all the headings, the pillars between them are worked back.

Fig. 200 is an enlarged plan showing two stalls, as worked at Celynen, and is designed to indicate more clearly the details of working the stalls forward and the pillars backward, with the timber, &c., used in the operation.

The props and collars used are notched in the Welsh style (see Figs. 181 and 182), and these sets of timber, not shown on the drawing (for clearness), are placed at intervals of from 3 to 6 feet along the road from the heading, and the tram rails the pressure causes it to rise in the road. The top coal of 3 ft. is left on for a roof.

Fig. 202 shows the method of working. The stall-roads are only 9 yards apart, and are cut off by cross roads at about every 50 yards. The roadsides are faced with stone brought down from the surface at a cost of 1*s.* per ton. The goaf spaces between the walls thus made are filled with small

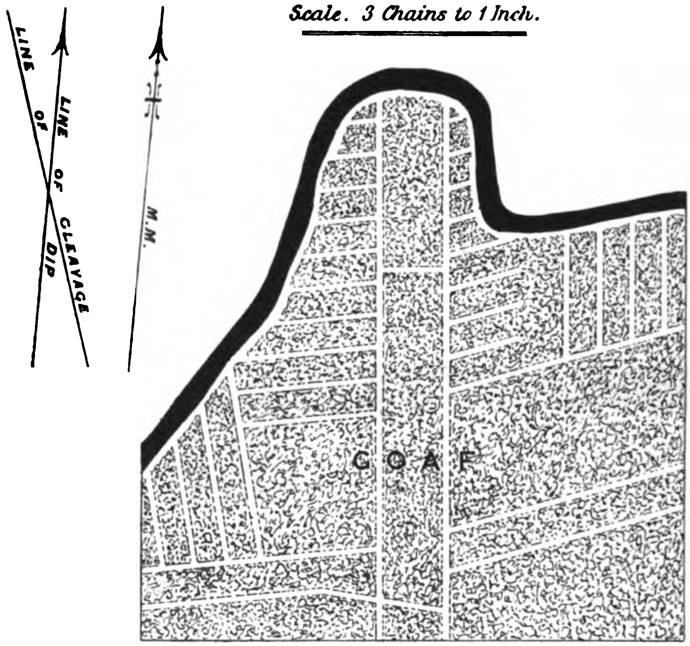


Fig. 202.—Risca Colliery, near Newport, Monmouthshire. Plan showing Longwall Method of Working the Black Vein.

coal, and stones and rubbish, obtained from the repairing of roads. The collier puts in props where required, there being no rule to fix the distances between them.

None of the props along the face are recovered, because the seam is fiery. If the props were taken out, holes would be left in the roof in which the gas would collect. In the roads 10-inch props with lids are put in about a yard apart, but those for securing the faces are from 5 to 6 inches in diameter. Two men and a boy work together in each stall, and can send out from 9 to 10 tons of coal in their 8-hour shift. In 1881 they were paid 1*s.* 6*d.* a ton for large hand-filled coal; they take charge of the stall and the roadway for 40 yards back. The line of cleavage is shown on the plan, Fig. 202, the cleavage planes being very distinctly marked. This renders the coal somewhat easy to work, as the coal comes off in large pieces, the collier merely lifting these with his pick. No holing is required under the coal unless in very exceptional circumstances.

The top coal left on makes a good roof near the face, but at a distance of 150 yards back, the full subsidence has taken place, and it becomes necessary to take down this top coal in the roads to make height. This ripping of the top is followed by sets of timber, which are notched in the Welsh method, as they are fixed in the road. The pressure on the sides of the road is very great. Larch

Scale. 3 Chains to 1 Inch.

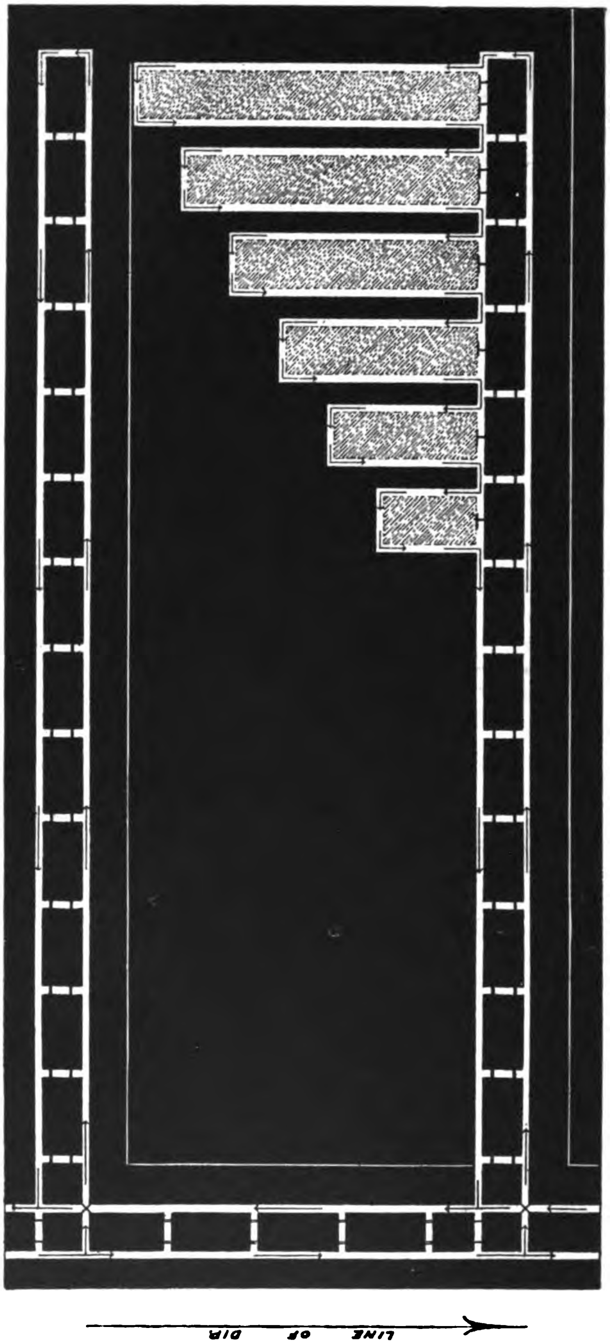


Fig. 205.—Plan showing the Wicket System of Working Coal in North Wales

connect these levels at intervals of 30 to 40 yards. At a suitable distance from the downcast shaft a bordgate, 6 feet wide, is driven towards the full rise, to the bank level. Here another bordgate is started as a companion to the first, being separated by 8 yards of coal. This pair of bordgates advance abreast of each other, and are connected by openings through the coal at every 20 yards. When the three levels have advanced 110 yards, another pair of bordgates is driven out of them similar to the first pair. In the centre of the block of coal left between the two pairs of bordgates a leading bank, 18 yards wide, is started from the bank level and is worked towards the rise. In the meantime the bordgates have advanced sufficiently, viz. 40 yards from bank level, to allow of slits being driven level course out of the bordgates towards the approaching leading bank. When the slits are holed into the bank (one on each side) they become the roads through which the coals are conveyed from the face. "Following-up-banks" are now commenced at the bank level, and consist of a face of 6 yards from the sides of the leading bank. These "following-up-banks," one on either side of the leading bank, are carried up to the slit above, and, as the leading bank has been progressing, it has now holed into the second pair of slits. Afterwards the "following-up-banks" are continued forward and second ones started from the bank level, taking another portion of coal 6 yards wide from that left. This operation is repeated as each leading bank reaches the slits, until the coal between the *goaf* and bordgates has been reduced as much as may be deemed safe or advisable.

In each of the 110-yard blocks along the level, the bordgates and banks are repeated in the same order as before described, and are carried forward to any required distance. In order to maintain a passage for the air and travelling road down each side of the goaf formed by the leading bank, packs 5 feet wide and 6 feet from the coal rib are built with the blue metal which is allowed to fall behind the face of the bank. Another pack of similar size is carried up the centre of the leading bank, and these three packs are built and carried on as the face advances. Each following-up-bank builds one pack as it advances, 6 feet from the ribside, so as to maintain a roadway. All the packs in course of time are overthrown by falls in the goaf, except those at the side, which must be kept to maintain the air-way. The pillars left in driving the bordgates, that is, those formed in each pair, and any coal it may have been deemed prudent to leave next the bordgates, are worked at the last when the banks have been worked out. This pillar coal is first worked at the highest and inside point and worked backwards.

In working the coal the holing was done in the "slottings," the hard coal above it being then wedged down. The clay seam was carefully separated from the rest, and the low bed of soft coal taken down. The timbering was placed under the top bed of softs; as the face advanced, the subsidence of the roof caused the top soft coal to fall at the goaf side of the props and was filled into the trams.

The air was taken from the downcast to the extremity of the level, doors being placed in all the working bordgates. From the face of the levels it was carried up the inside pair of bordgates, around the face of the new leading bank, up the next bordgate to the face, returning to the slit leading into the next bank. Here the air divided as it entered the bank, a small portion descending the air-way maintained by the pack-wall to the following-up-bank on that side and returned by the middle level to the other side of the bank, ascended the air-way corresponding to the one it descended on the other side, and joined the current, which had traversed the leading bank face, at the highest holed slit. The united volume passed along the slit to the next pair of bordgates, where it ascended the one, returning along the other to the highest slit, dividing, as before, at the first bankgate. Here a division took place, one portion descending to air the following-up-banks on that side and passing into the middle level. The main current, after passing the leading bank and descending to the highest slit, again divided, giving off a split to ventilate the following-up-banks below. This current joined that which had aired the following-up-banks on the other side in the middle level, and the two currents passed on up the next bordgate to join the main current at the highest holed slit; these re-combined currents passed into the first pair of bordgates from the shaft, by one of which it descended to the upcast shaft.

It will thus be seen that whatever advantage was obtained from draining the gas into the bordgates was quite neutralised by these ventilating arrangements. The air, after traversing the bordgates, would become charged with the gas emitted there, after which it was passed to the workmen at the face of the banks.

The Lundhill pits are 210 yards deep to the Barnsley Seam, and prove the following seams:—

Melton Field .	4	feet	thick	at	40	yards	from the surface.
Abdy	3	,,	,,	,,	70	,,	,, ,,
Kents Thin . .	2	,,	,,	,,	106	,,	,, ,,
Kents Thick .	3	,,	,,	,,	144	,,	,, ,,
Barnsley Seam .	8	,,	,,	,,	210	,,	,, ,,

Under the town of Barnsley the next coal seam of importance below the Barnsley Bed is the Swallow-Wood Coal, 60 yards below; the next is the Flockton, 140 yards lower. The Parkgate is 80 yards below the Flockton, after which

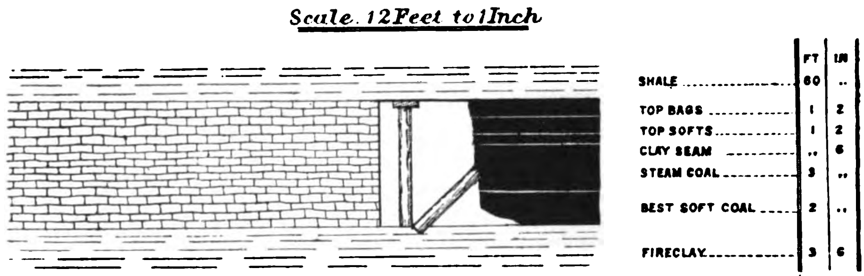


Fig. 207.—Lundhill Colliery, near Barnsley. Mode of Spragging and Propping in the Pillar Workings of the Barnsley Seam.

comes the Thorncliffe Thin, 35 yards still lower. Next in descending order is the celebrated Silkstone Seam, 52 yards below the Thorncliffe Thin, but none of these seams approach the thickness of the Barnsley Bed.

The Bank System of working has now given place to the Bord and Pillar. In 1881 the operations at Lundhill were confined to three shafts, one being used solely as a ventilation pit. All are circular shafts, the two downcasts being 11 feet 6 inches and 12 feet in diameter respectively, and the upcast 14 feet 6 inches. A pair of horizontal high-pressure winding engines, 25-inch diameter cylinders, 5-foot stroke, and 14 feet 6 inch drum, winds coal at one of the downcasts, and a similar engine at the other. Together they raise 750 tons in one shift of 9½ hours. A furnace, having a grate surface of 144 square feet, placed at the bottom of the upcast shaft, produces the ventilation. The quantity of air passing up the upcast is 300,000 cubic feet per minute. This air is not passed over the furnace, but enters the upcast shaft by means of a dumb drift 35 yards from the bottom. The total volume of 300,000 cubic feet includes 50,000 cubic feet of air used to ventilate the stables and feed the furnaces.

Fig. 207 shows a section of the Barnsley seam taken in 1881, from which it is seen that the coal has a total thickness of 7 feet 10 inches.

The method of working now adopted is shown on the plan, Fig. 208, and is an arrangement of Bord and Pillar. The pillars are 40 yards square, and the roads the method of working, which is by Longwall with the roads advancing to the rise.

One of the roads, most suitable for the purpose, is made a "jinney," or self-acting incline, and levels from it cut off the roads going to the rise every 200 yards. The distance between the stall roads is 60 yards. At each road-head iron plates are placed, and rails for the tubs laid along the face. The tubs are "spragged" in being taken down the road leading to the incline. In a 60-yard stall two stall-men work. They are small contractors, and receive a tonnage price on all coal sent from the stall and delivered at the top of the "jinney."

They employ other men in the stall. The contractors do the packing, the timbering, and the ripping at the stall roads.

The system of building pack-walls next the roads and in the waste is similar to that at High Park, shown in Fig. 217, and described later on.

The ripping in the roads yields material for building the packs at the sides of the road, which is 10 feet wide. These packs are made 6 feet wide, and others parallel to them are built in the waste of the material yielded in the holing and fallen stones. The face is protected by two rows of 6-inch timber, carried along the face behind the rails. The rows are 6 feet apart and a similar distance separates any two props in the rows. As the pack-walls are built, the rear row of props is taken down, and re-set in advance of the other row. As the props are drawn, the roof in the waste falls. Although the stall-men have entire charge of the place, they must build the pack-walls and place the props as directed by the deputy. Commencing at the road-head, these men hole along both sides to the extremities of their stall. They hole 5 feet under the coal, the height of the holing in front being 18 inches. Sprags are put in under the coal 6 feet apart as the holing proceeds. See Fig. 213. Where the coal is tender, besides the sprags, "cockermegs" are put in. These consist of a sloping prop reaching from the floor, and another reaching from the roof which hold a third prop placed horizontally along the face. See Fig. 213. When the holing all along the coal is completed, and held in position by the sprags, a cut or shearing is made in it at the road-head, after which one or two sprags are taken out, thus allowing a portion of the coal to fall. As this coal is removed in the tubs, other sprags are taken out, and more coal taken down. This process is continued along the stall on both sides of the road until the extremities are reached, the rails being laid forward as the coal is taken down in front. After a holer has worked to the end of his "bank," he returns to the road-head, and there begins another holing at the point from which the coal has just been removed.

The stall-men have two fillers, and one holer, so that 5 men work in a 60-yard stall, from which they send out daily from 20 to 25 tons of coal. No timber is used in the roads, the roof being excellent. The timber for the face costs 2*d*. to 2½*d*. a ton.

At the High Park Colliery, Langley Mills, Notts, the Barnsley seam is called the Top Hard Coal and is worked by Longwall. The colliery has been working since about 1861, the Top Hard seam being won by two shafts at a depth of 200 yards. These are both downcast and winding shafts, one engine hauling coals from both. Each shaft is 10 feet in diameter, and carries only one cage. Two trams, each holding 11 cwt. are placed in each cage. About 900 tons a day are landed, all the coal being picked and lifted from the trams into the waggons by hand, and no screens used. The upcast shaft is 12 feet in diameter and 116 yards deep, it being situated 1,100 yards away (on the rise side of the measures) from the downcast shafts. Near it, is another upcast shaft for a separate colliery. Each upcast has a Waddle fan 30 feet in diameter, so placed that, should one ventilator break down, the other can be used to draw the air from both shafts.

The fans run 70 revolutions per minute.

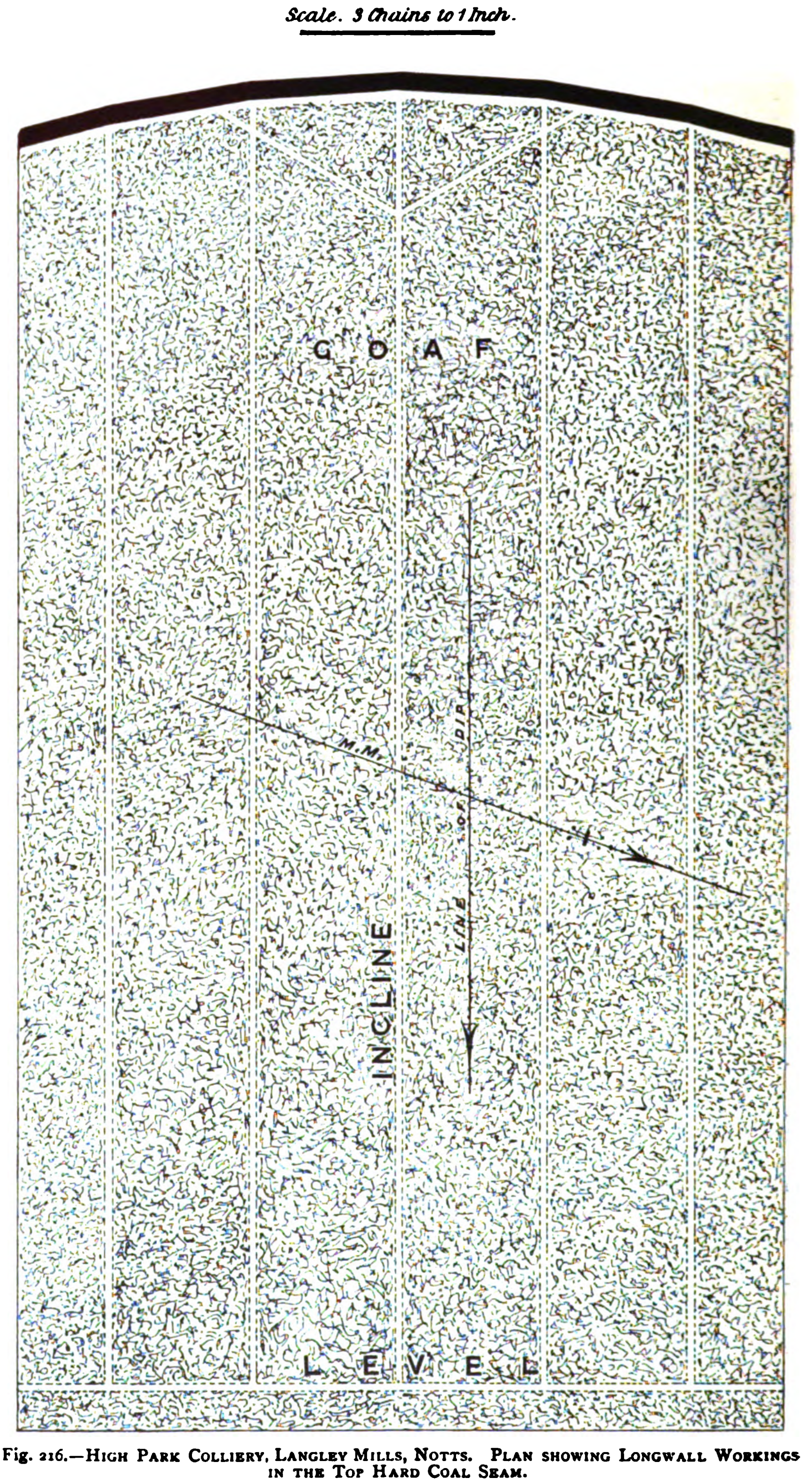


Fig. 216.—High Park Colliery, Langley Mills, Notts. Plan showing Longwall Workings in the Top Hard Coal Seam.

and are about 5 feet 3 inches long. The collars are 4 feet long and 5 inches by 4 inches in section. Having removed the coal first got, other sprags are taken out, allowing another portion of coal to fall, which in its turn is filled into the trams and removed, this process being continued until the end of the stall has been reached on both sides. Before commencing to add a fresh portion of building to the packs there are 2 rows of props about 5 feet apart behind the props, and sets of timber over the roadway. The stall-man draws these props as he builds the packs. The roof in places is tender and much broken, and where this is so the packs are built of the débris from the roof, about 6 feet wide, with intervals of 9 feet between them. The ripping of the road yields material for packing at the sides.

The duties of the deputies in the pit are much the same as that of the firemen in other districts. They superintend the stall-men, examine the places for firedamp,

Scale. 43⅓ feet to 1 Inch.

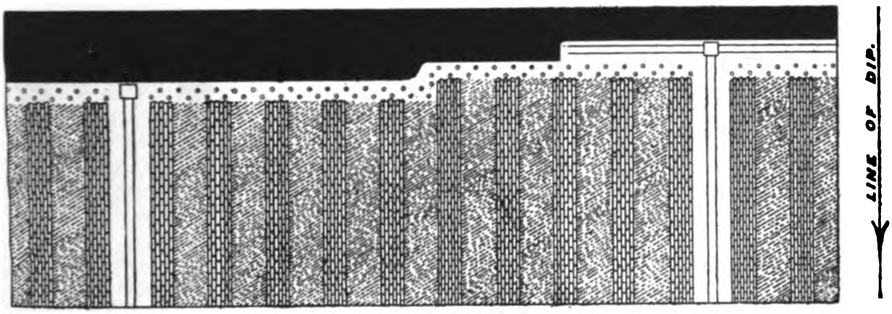


Fig. 217.—High Park Colliery, Langley Mills, Notts. Plan showing part of two Stalls with the packwalls, in the Top Hard Coal Seam.

attend to the ventilation, &c.; but, subject to this superintendence, the stall-men set up and draw out all the timber, and rip the roads. Naked lights are used in the workings, but a safety lamp is fixed in the highest part of the ripping, so as to give warning in case any firedamp appears. Powder is used in ripping the roads.

Very few props are used in the main roads, as the latter sink quickly, and in the main levels this necessitates ripping up to the 2 feet 6 inches of coal, which makes a good roof. The extreme subsidence takes place at a point 50 yards back from the face. The roads being dusty are watered, but not regularly.

The method of working the Maudlin Seam, Wearmouth Colliery, Sunderland, is by Pillar and Stall. The colliery has been working coal since 1835, but sinking operations were commenced there in 1826. There are two shafts, an upcast 11 feet 6 inches in diameter, and the downcast 12 feet. Both are winding shafts, and together in 1881 they raised 2,000 tons a day, the day here consisting of 24 hours. The tubs used carry 8 cwt. each. Two 4-decked cages run in each shaft. Two tubs are placed in each deck, so that the cage carries 8 tubs. The furnace which helps to produce the ventilation is not placed at the bottom of the upcast, but in the Maudlin Seam, 44 yards from the bottom. It has a firegrate area of 144 square feet, and is greatly assisted by the furnaces of 6 boilers. The total quantity of air varies from 180,000 cubic feet per minute to 200,000 cubic feet, being highest in the winter months, when the natural ventilation is greatest.

The shafts prove the Maudlin and Hutton seams at a depth of 530 yards and 574 yards respectively, both seams being worked. They lie almost flat, the dip to the east being so slight as to cause no inconvenience in carrying roads in any direction. The working faces in most of the districts are from 2 to 3½ miles distant from the shaft, and the coals are brought out along engine planes. The tail-rope system of haulage is adopted, and there are more than 20 miles of steel-wire rope in daily use in the pits. In one of the longest engine planes, where the empty train is taken in at the same time that the loaded one is brought out, there are 126 tubs on the road at one time.

A section of the Maudlin Seam is shown in Fig. 218.

The openings in the coal are 12 feet and 9 feet wide, the pillars left being 40 yards square. No holing under the coal requires to be made by the workmen, as the coal is tender, and the workmen "scallop" it, that is, they hack it out with picks while standing upright. In the screens used the bars are an inch apart,

Gears in working place.
Fig. 218.

Gears in main road.
Fig. 219.

METHODS OF TIMBERING IN THE MAUDLIN SEAM, WEARMOUTH COLLIERY, SUNDERLAND.

and 55 per cent. of the coal got passes through the screens. There is no objection to this, as the coal is used for gas making. The colliers do not place their own timber at the working faces. This is done by the deputies, unless in cases of emergency, when the collier puts up timber, which is left in his place for the purpose. In the solid working, sets of timber are placed in the roads every 3 feet. The props are 3½ inches in diameter, 6 feet high, 5 feet 6 inches apart at the bottom, and 4 feet 6 inches apart at the top, as shown in Fig. 218. The rails are laid in the centre of the roads between the two upright props. The collars are made of 5-inch props split through the middle, and the half-round side is placed next the roof.

The props are put in by the deputies as the places are being driven, and remain in until another end is through. The deputies then draw them, after which the roof usually falls to a height of 2 or 3 feet; but sometimes the fall is sufficient to fill the place up. The pillars crumble off at the sides, owing to the tender nature of the coal.

The Broken is worked by splitting the pillars, and bringing back the jud on both sides. A deputy attends about 12 men, and they send out about 60 tons a day.

The collier receives 10*d*. a ton for hewing and filling, in the whole mine.

The roof over the coal is good, the cost for timbering being 1¼*d*. per ton in the roads and solid places, and 2*d*. in the pillar working.

The main roads are timbered, as they require it, with props, or sets of timber. At some parts the sets are placed 3 feet apart, and are 6 inches square and 9 feet long. The props in the main roads are 4½ inches in diameter, 8 feet long, *Scale. 49⅓ feet to 1 Inch.*

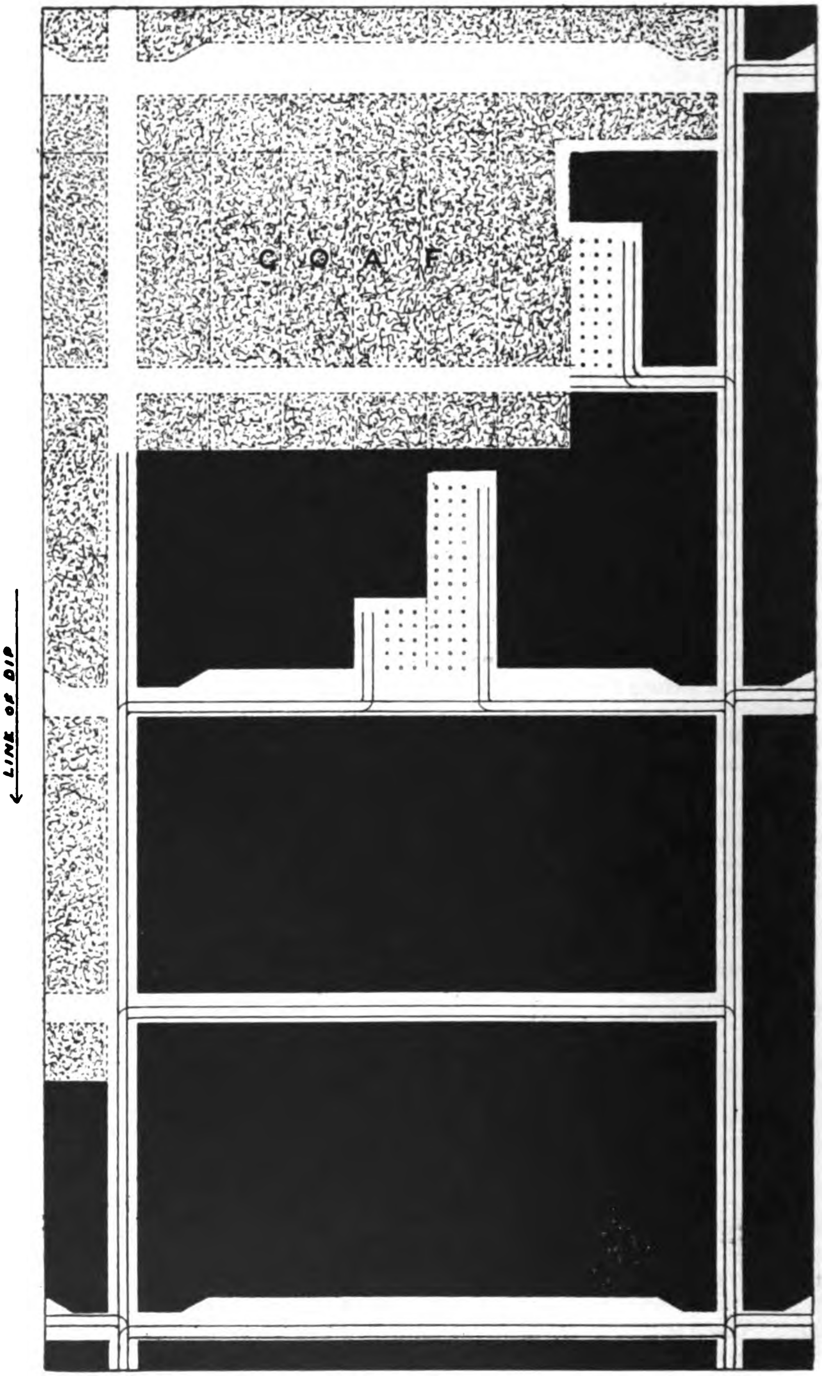


Fig. 220.—Silksworth Colliery, near Sunderland, South Durham. Plan showing the usual method of Pillar Working in the Maudlin Seam

horizontal winding engines, with 42-inch cylinders, 7-foot stroke, double beat valves, and a 28-foot diameter cylindrical drum, are used. The engine is supplied with an automatic cut-off, to stop the cage when it reaches the surface. Coals are placed in the cages at levels or loading stages in the shaft, besides the bottom; these levels being at the Bassey, Chalky and Ash Seams, which are all worked above the Great Row.

In 1881 a Waddle fan, 45 feet diameter, was being erected at the upcast shaft.

Fig. 223 shows a section of the Great Row Seam, the average thickness of which at this colliery is, however, 6 feet. Over the coal are 8 feet of fireclay, and above this a bed of coal 2 feet 6 inches thick. Resting on this coal are beds of fireclay and bass (hard dark shale), 8 feet thick. Above this again come 8 feet of coal and partings, and then higher, 32 feet of "Binds, Slums, and Marls." The thill is composed of 37 feet of "Bass, Binds, and Clod."

The seam dips 1 in 7. Fig. 224 is a plan showing the method of working. From the pit bottom, for a distance of 350 yards, a pair of levels 10 yards apart were driven, and, on reaching a certain point, a "breasting" or face of coal 25

Scale. 12 Feet to 1 Inch.

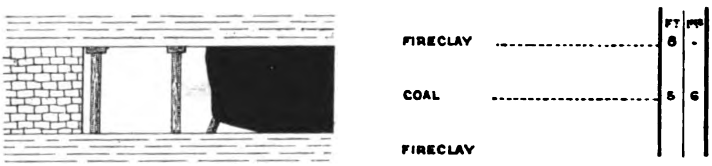


Fig. 223.—Florence Colliery, Longton, North Staffordshire. Mode of Propping and Spragging at the Working Face on the Great Row Seam.

yards wide was taken out, and a "gob" road formed in it 4 yards from the deep side-rib. The stall-roads are turned out of this gob road every 88 yards, and carried to the rise. They are cut off by a level every 120 yards. The faces are "stepped," one being 15 or 20 yards in advance of another. The coal is conveyed along the main level by horses, but a "jig," or self-acting incline works in every going stall-road, by means of which the coal is let down to the level.

For this purpose a 12-inch wheel is placed in a fork, and the end of the fork is passed through a prop at the road-head. A brake, consisting of a piece of iron, is pressed on the rim of the wheel by a handle working a dumb screw fastened to the fork. Only one tram is run at a time; each tram holds 8 cwts., and the brake is just powerful enough to stop it at any point during the run. A chain is used on the inclines, and as the face advances the wheel is easily moved forward.

The bottom level advances, taking a "breasting" or face of 25 yards. The road is 6 feet wide, and on either side of it a pack-wall, 3 yards wide, is built. The face is protected by two rows of props, which are 6 inches in diameter, the rows being placed 4 feet 6 inches apart, with a $5\frac{1}{2}$ or 6-foot space between the props. The lids used over the props consist of broken props, when there are sufficient for that purpose. Fresh building is put in every 5 feet, the rear row of props being drawn and re-set in advance. Sprags 6 feet apart are used under the coal. The space between the pack-wall and the coal on the deep-side is kept open as long as possible, but when this can only be done with difficulty, a hole is driven through the building from the road, and a fresh air-course carried on from this point. A chock of broken timber is fixed at the corner of each hole, and there is an interval of about 40 yards between these holes.

The stalls are 88 yards wide, and packs 3 yards wide are built parallel to the road (which is wide enough to admit of a double road for the self-acting inclines) 7 yards apart all the way from one stall-road to another.

This will be seen in Fig. 225, which is an enlarged plan showing the packs, timbering, &c. The packs formed next the roads, it will be observed, are wider than those in the waste, being 4 yards. The stones obtained from the 3 feet of ripping are used to build the packs on both sides of the roads. In every 88-yard stall, are two stall-men, one for each side of the road. In their employ are 4 holers, 2 buttockers, and 2 packers.

The stall-men build the packs, set and draw the timber, and despatch the coals down the incline to the level, for which they receive 15*s.* per score of tubs of coal—7*s.* being allowed for slack—all of it being raised. The workmen rake the coal into iron boxes or trays 2 feet square and 6 inches deep at the one end,

Scale. $49\frac{1}{3}$ feet to 1 Inch.

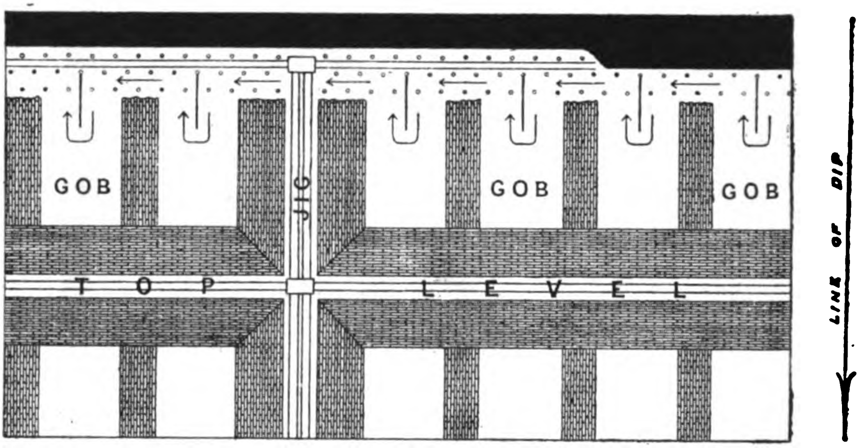


Fig. 225.—Florence Colliery, Longton, North Staffordshire. Plan of Longwall Face with the Gate-road and Packwalls connected with it, in the Great Row Coal.

tapering out at the other. For carrying, these have two handles, which also enable the workman to empty the coal into the tubs conveniently. The men in one of these stalls send out during the day, or shift, 100 tubs, each of which holds 8 cwt. Besides the score price referred to, the stall-man receives 5*s.* 6*d.* a yard, for top ripping and building the road packs. Any other stones required for packs are obtained from the waste. The holing under the coal is made 4 feet in, being 2 feet high in front, the holer placing sprags under the coal every 6 feet, as shown in Fig. 223. The sprags are 2 feet long and 6 inches in diameter. The holing across the stall being finished, the sprags are knocked out; the coal is then blown down by powder. The stall-man fires the necessary shots. The coal is shot down in advance of the point where the rails are laid, but for the convenience of filling, the rails are kept very close up to the "buttock" or piece of coal, next to be blown down, and advanced as required. The holers are not obliged to remove from one side of the stall to the other, as the shots are fired, but only those working on the side where the firing takes place. After the shot is fired, they return to their work and continue as before. The props at the face are in two rows, 4 feet 6 inches apart, there being 6 feet between the props forming a row. They are from $5\frac{1}{2}$ to 6 inches in diameter at the thin end, and are put in by advances along the face, sprags are placed under the coal every 6 feet as sdown in Fig. 229, in addition to the cockermegs already fixed. The packs are kept up within 6 feet of the face, as required by the Special Rules in force, but interposed between the packs and the face, and placed parallel with it, are set two rows of 8-inch props, having 6-foot spaces between the props, the two rows being 4 feet 6 inches apart. The rails are laid along the face between these rows. Besides this double row of props, two rows of timbering are placed between the pack-walls in the waste, these being set 6 feet apart under the top coal. Each of these rows of timbering comprises three props and a chock. The collier shears the top coal which is supported by these props, then draws the timber out, allowing the top coal to fall. He next sets props with a lid and a sill on the rubbish (see Fig. 162), thus securing the roof from which he has just taken down the top coal, and he is

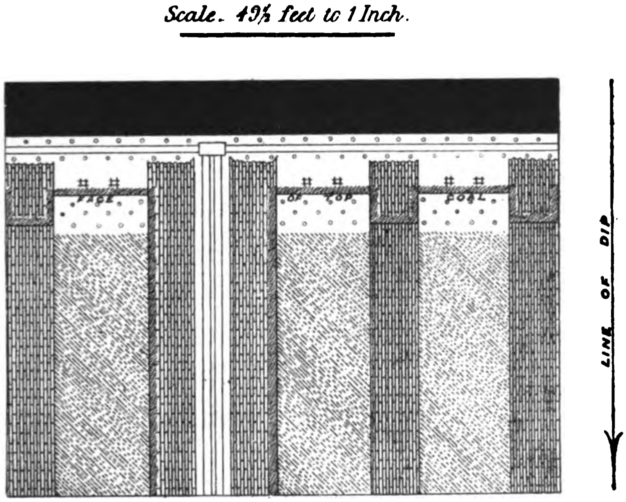


Fig. 230.—Great Fenton Collieries, Stoke-upon-Trent. Plan of Longwall face in the Great Row Coal Seam, showing Props and Chocks.

now able to work at that portion of the top coal remaining over the packs. To obtain this coal, he holes in on the 4-yard packs, for 2 yards on each side, thus allowing the coal to drop. All the coal over the packs is not got, but as much of it as it is prudent to take without running risk from the roof. The props set on the sills over the rubbish are now withdrawn with a dog and chain, by the man specially appointed for the purpose, the chocks preventing the breaking of the roof, which immediately follows, from reaching the face. The top coal is not taken down in the roads, nor from over the packs formed at the sides of the roads. See Fig. 230. In time, however, as the roof sinks, this top coal is ripped in an arched form, leaving a good road which requires little timbering.

There is nothing special at this colliery in the manner of timbering the main roads. Props 8 inches in diameter are used, having lids 6 inches by 5, and 2 feet 6 inches long. In some of the main roads 7-inch collars are notched into the coal on one side and supported by a short 3-foot prop resting on the coal at the other. The roads are dusty and are watered as they require it. The roof generally is very good and safe to work under, requiring little timber.

At the Cannock and Rugeley Colliery, Hednesford, South Staffordshire, the Deep Coal Seam is worked on the Longwall system. The colliery has been in road in between must not exceed 5 feet, and a prop must be set within 4 feet 6 inches of the buttock of coal.

"The sprags under the coal must not exceed 5 feet apart, and they must be well set.

"The stall-men or contractors to see that the timber is set as above, and to see that the workmen employed by them do not work in danger.

"The stall-men or contractors to examine the way end, and all parts of the stall, and if there is any danger the men must not be allowed to work.

"The chain and bar must be used in drawing timber. Workmen found endangering their lives by neglecting to use the chain and bar, or by not setting timber to secure the roofs and sides whilst drawing timber, will be sent out of the mine at once and summoned before the magistrates.

"The competent men must see that the timber is set, as above, and examine the coal, roof, gob road and wastes of each stall, and if they find any danger, to

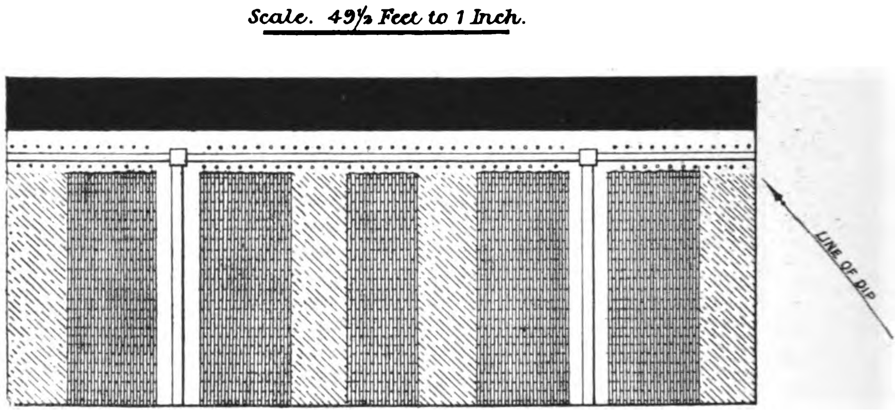


Fig. 233.—Cannock Wood Collieries, South Staffordshire. Plan showing Longwall Face in the Deep Coal Seam, and the Stalls, Packs, and Props connected with it.

stop the stall, and report the same at once to the underground manager. The underground manager to go and examine the stall, and if he finds it in a dangerous state for want of the timber not being properly set, the stall must be stopped until the danger is removed, and the stall-men to be fined not less than the sum of two shillings and sixpence for each offence."

These regulations apply to the Deep Coal, and a corresponding set are framed for the Shallow Seam, in which the props at the face must not be more than 4 feet apart and not more than 5 feet between the first prop and the building.

The coal in the Deep Seam is holed under, 5 feet 6 inches in, and is supported by sprags placed 4 feet 6 inches apart. The sprags are 2 feet long and 8 inches in diameter. 2 feet back from the face, a row of 8-inch props, having broken props for lids, are set 3 feet apart parallel to the face. See Fig. 231. From the centre of each prop on the side towards the coal, is placed a short 8-inch sprag, at the other end of which is a lid bearing on the face of the coal. 5 feet back from this, another row of props with lids is set, the props being 3 feet apart, and parallel to the row in front. Close behind the last-named rows are the packs. The rails are laid along the face between the two rows of props as shown in Fig. 233. This method of getting the coal gives a large percentage of small coal, and an experiment is being made in another way. This is called "banicking."

and above this white metal. Under the coal are 18 inches of warren earth or fireclay, and below this is a hard grey metal. The 7 inches of "Daugh" or fireclay is used to hole in, and the holing is carried a yard in, after which the 1 foot 5 inches of top coal is taken down. The holing is continued another yard, and the top coal over it taken down. This is continued until 9 feet of bottom coal is bared, which is then got by blasting.

The seams dip at an inclination of 1 in $3\frac{1}{2}$.

The method of working the Doe Coal Seam will be understood by reference to Fig. 238. From the extremity of the cross-measure drift between the Trencher-

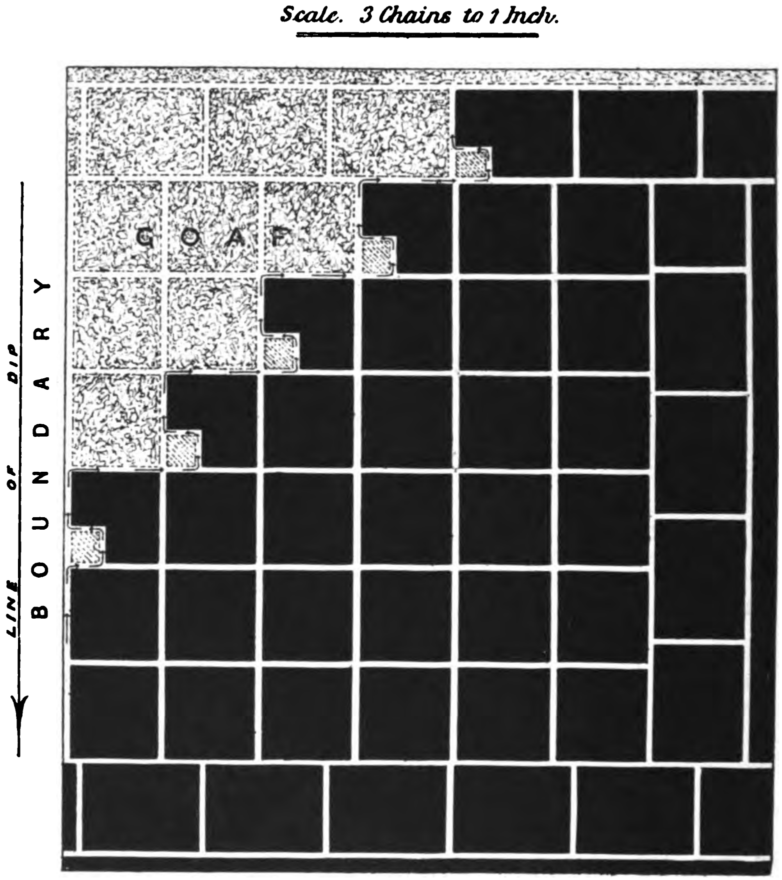


Fig. 238.—Clifton Hall Colliery, near Manchester. Method of Working the Doe Seam.

bone and Doe Coal Seams a pair of levels separated by 30 yards of coal are driven to the boundary. Openings or cut-throughs connect the levels every 40 yards. At a point 200 yards to the rise another pair of levels is driven parallel to the first pair, the coal between the two pairs being at first left solid. On the lower pair of levels reaching the boundary a pair of places, separated by 30 yards of coal, is driven out of them to the full rise from a point 200 yards back from the boundary, and these places are continued until they hole into the upper levels Levels and throughers, all 7 feet wide, are then driven so as to divide or split up the coals, fill and let them down the "jig brow," build the packs, and set and draw all the timber required. They are paid 6*s.* 3*d.* for getting 10 waggons, or 3½ tons of coal, and 4*s.* a yard for building the packs. The coal is riddled through a ¾-inch riddle, and for 3½ tons of small coal they receive 2*s.* 6*d.* Twenty per cent. of the total quantity got is small. The 3 men and lad send out about 10 tons a day. The fireman superintends these men, and sees that the necessary props and chocks are put up.

The roof in the roads is not good; here the top coal is left on, but is taken down in the waste. Timber costs 2½*d.* a ton in this seam.

Fig. 240 shows a section of the Five-Quarters Seam, which is 24 yards below the Doe Seam. The coal is 3 feet 4 inches thick with the fireclay partings included. Of this only the 1 foot 10 inches, which is of excellent quality, is sent out. Over the coal is hard sandstone reaching up to the Doe Coal. The

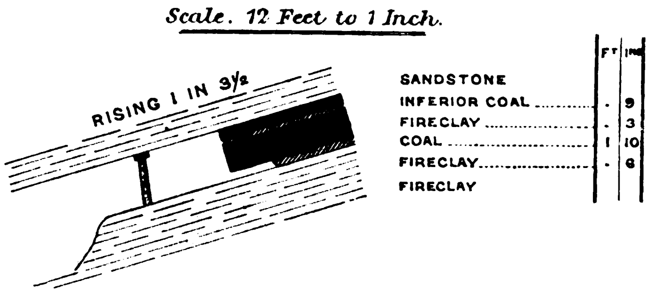


Fig. 240.—Clifton Hall Colliery, near Manchester. Section of Five-Quarters Coal Seam.

6 inches of "daugh" or fireclay is used to hole in. Under the coal is "warren earth" or fireclay. To make height in the roads, water is put on this to render it soft, and it is afterwards easily taken up.

The method of working pursued on this seam differs slightly from that last described on the Doe Coal. Here, as shown in Fig. 241, instead of driving a pair of 7-foot wide levels separated by 30 yards of coal, a face or breasting of coal 13 yards wide is driven, and a single level formed in it. This road is not carried in the middle of the breasting, but to one side, in a manner similar to that in the Ram's Mine at Pendlebury Colliery, shown in Fig. 247, and fully described later on. A building 6 yards wide is formed on the rise side of the road, and another 3 yards wide on the dip side. Between the pack and the ribside on the rise side is an aircourse 3 feet wide, the aircourse being carried alongside the coal as the level advances.

A similar level is driven from a point 140 yards to the rise of this (see Fig. 241), the levels being carried thus to the boundary. When this is reached a place is driven to the full rise between the lower and the upper level from a point 140 yards back from the boundary. A block of 140 yards square is thus left. Commencing at the face of the level this 140-yard square pillar is now worked by lifts 20 yards wide being carried to the rise across it between the two levels. The road is carried in the centre of the lift, and protected on each side by a facing built of the 9 inches of inferior coal. The rubbish yielded in ripping the roads and in holing the coal is sufficient to fill the whole space in the waste. An aircourse is formed along the ribside. No chocks are used, but props are set 6 feet apart along the face. The props are 3 inches thick, and 3½ feet long, the lids over them being 2 feet × 5 inches × 2½ inches. The props are not set at right angles to the floor of the seam, nor in a vertical position, but at an inclination between the two, so that as they afterwards sink they assume a position more nearly at right angles to the floor. Only one lift at a time is taken out across the pillar. The collier receives 8*s.* 8*d.* for $3\frac{1}{2}$ tons of riddled coal, but he has to draw the coal to the foot of the second jig brow for this, some 200

Scale. 3 Chains to 1 Inch.

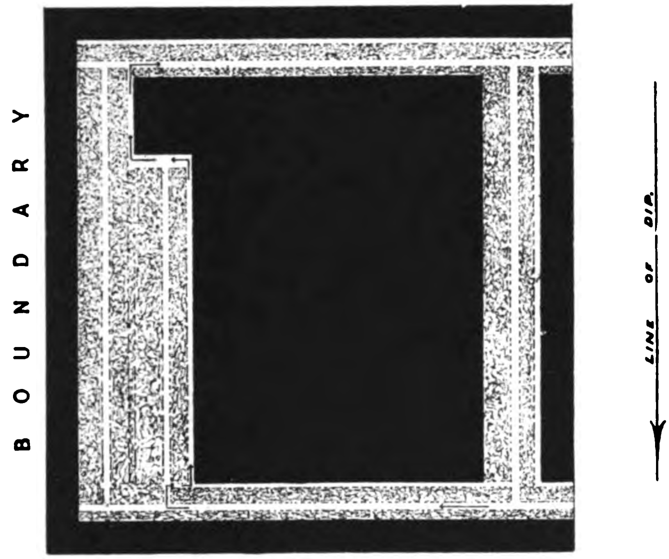


Fig. 241.—Clifton Hall Colliery, near Manchester. Plan showing Method of Working the Five-Quarters Coal Seam.

Scale. 12 Feet to 1 Inch.

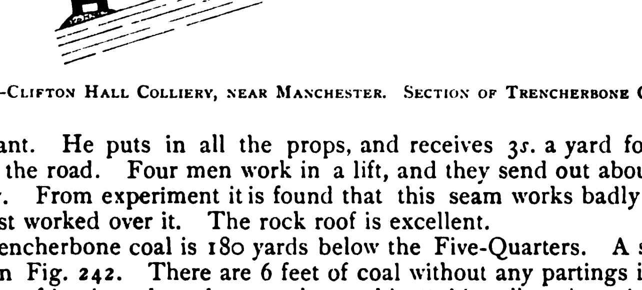


Fig. 242.—Clifton Hall Colliery, near Manchester. Section of Trencherbone Coal Seam.

yards distant. He puts in all the props, and receives 3*s.* a yard for the pack facings in the road. Four men work in a lift, and they send out about 7 tons of coal a day. From experiment it is found that this seam works badly if the Doe Coal is first worked over it. The rock roof is excellent.

The Trencherbone coal is 180 yards below the Five-Quarters. A section of it is shown in Fig. 242. There are 6 feet of coal without any partings in it. Over it are 4 feet of laminated sandstone; then a thin "chitter" coal 15 inches thick, rock of the Carboniferous period (one found at Bacup being of granite), some of them are of great size, and there must have been some unknown means of transport, to distribute them over areas so widely separated as England and America. The whole of this interesting problem is as yet unsolved.

The method of working the Trencherbone Seam at the Clifton Hall Colliery is similar to that of the Five-Quarters last described. The levels are driven in precisely the same way, but the pillars are only 100 yards square, and instead of only one lift being taken off the pillar at a time, at least two, and sometimes three, are taken up-hill at the same time, as well as one or two going down-hill. The collier uses a windlass to draw the coals up from those places going down-hill. Fig. 243 shows three lifts proceeding at the same time to the rise. They are driven 20 yards wide, the face of each being kept from 20 to 28 yards in advance of the one behind it. The road is formed in the centre of each lift, and on either side of the road a pack 3 yards wide is built. A double row of chocks is kept next the face and parallel to it. The rails are laid in between the chocks, which are 5 feet apart as shown in Fig. 243. As the face advances sufficiently, another row of chocks is put in and the rear one withdrawn. Before proceeding to take down the rear row, props are set around each chock. The rubbish on which the chock was built is cleared away, and the chock knocked out at the bottom. The props giving security round the chock are then taken out, and the roof falls. The sprags shown in Fig. 242 are placed to prevent the coal riding over. They are 5 feet 6 inches long, and the lids over them are wedge-shaped.

The roads are dry and dusty. Generally the roof is good, and the main roads do not require much timbering; in places where the roof is bad props and sets of timber support it, cross-pieces and laggings being placed over the collars. The two props of a set are not of the same thickness, the one to the dip being 5 inches in diameter, and that to the rise 4 inches. The collar is 7 inches in diameter. It is said that an advantage arising from this system of working is the fact of keeping the weight on the face and off the roads.

No blasting is done in the pillar working.

The Pendlebury Colliery adjoins the Clifton Hall, the two collieries belonging to the same company. It has been working since 1848, and at present works the Shuttle, Crumbouke, and Ram's Mines, by means of an upcast and a downcast shaft. The latter is 400 yards deep to the Shuttle and Crumbouke coal. At the pit bottom, a direct-acting pump raises the water in one lift to the surface.

The upcast, Furnace, or No. 2 Pit, is used to wind coal from the Ram's Mine. The workmen also ride up and down it, and as it is very hot, to guard against serious consequences through the cage being stopped by accident in the shaft, a signal is provided by which communication is made with the pit bottom from the cage. The onsetter on receiving such signal can open the separation doors and let the fresh air into the shaft.

A section of the Ram's Mine is shown in Fig. 244. The coal is 5 feet thick, besides the 4 inches of inferior coal which is left on in the working, but taken down in the roads. Over this inferior coal are 7 yards of blue metal, 3 feet of which are taken down to make height in the roads. Under the coal is blue metal. The coal is first holed in the middle at the parting, and the top coal then taken down, and kept 3 feet in advance of the bottom coal, which is blasted. A competent person is employed to fire the shots. In some places, the seam is holed in the 4 inches of daugh under the coal, but this does not run continuously through the seam. Where holed under the seam, sprags 18 inches long and 6 inches high are put in with wedges on the top.

The Ram's Seam is 200 yards above the Doe Seam, being worked at Pendlebury in the same way as the Trencherbone Seam is worked at Clifton Hall. An underground hauling engine, supplied with steam from boilers placed at the bottom of the upcast, hauls coal up a road driven 500 yards to the dip. The whole of this road is secured with brick arching. Levels are driven out of this dip road. At the far end of the dip, and for a short distance along the lowest level leading out of it, are some double-headed rails used as props, see Figs. 245 and 246. Two uprights 6 feet long support a curved crown 10 feet long.

Scale. 12 Feet to 1 Inch.

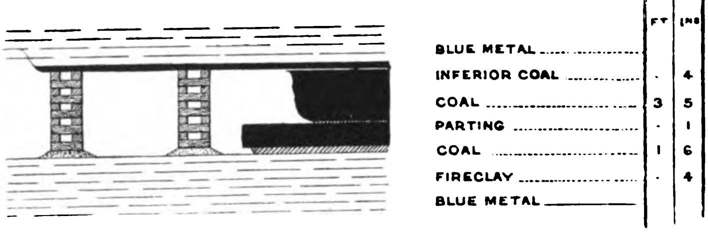


Fig. 244.—PENDLEBURY COLLIERY, NEAR MANCHESTER. SECTION OF RAM'S MINE COAL SEAM, AND TIMBERING IN IT.

The uprights are set on sills, 12 ins. × 5 × 7. The crowns are curved slightly, having a versed sine of 6 inches. To keep them securely in place, short props 6 inches long and 6 inches thick are wedged in between the end of the rail and the side as shown in Fig. 245. These sets of rails are fixed every 4 feet along

Scale. 12 Feet to 1 Inch.

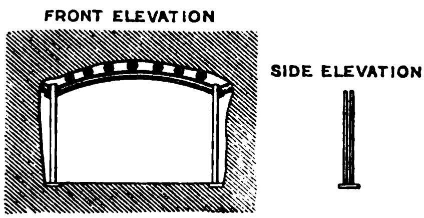


Fig. 245. Fig. 246.

PENDLEBURY COLLIERY, NEAR MANCHESTER. METHOD OF SECURING ROOF BY DOUBLE-HEADED RAILS.

Scale. 49½ feet to 1 Inch.

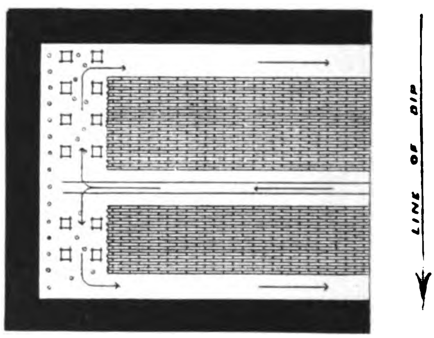


Fig. 247.—PENDLEBURY COLLIERY, NEAR MANCHESTER. PLAN SHOWING THE FACE OF LEVEL IN THE RAM'S MINE.

the level for some distance, and rafters placed over the crowns. Little or no timber is used in the rest of the level. Owing to a fresh arrangement the boundary line was altered after this level had been driven and the coal worked back. It became necessary, through the boundary extension, to drive the level on again. At the point it had been standing, the roof was tender and required props and sets of timber to secure it at that point, but not elsewhere. The level is driven 20 yards wide, as shown in Fig. 247. On each side of the road, which is 9 feet wide, packs are built. The pack on the rise side of the road is 8 yards wide, and above it is an aircourse 9 feet wide extending from the pack to the coal. The pack on the low side of the road is 6 yards wide, and beyond the pack on the low side is an aircourse 6 feet wide. The face is protected by 2 rows of chocks, a row on the rise side consisting of 4 chocks and on the low side 2

Two colliers work in a 20-yard stall, each having a working face of 10 yards from the road. Each employs a drawer. The collier receives 1*s*. 4*d*. per ton for large coal and 1*s*. per ton for small. For this he holes the coal, sets the sprags, gobs the rubbish, and delivers his coal at the bottom of the jig brow. Of the coal got 95 per cent. is large.

A collier sends out an average of 7 tons of coal in a shift of 9 hours. Special contractors put in the packs and chocks. They are paid 6*d*. a ton on the coal sent out for building the packs, setting and drawing the props and chocks. The roads from the pit bottom inwards are under their charge, and they work on them both by day and night. Four-fifths of the work is, however, performed during the daytime. There are two pairs of contractors in the pit, one to each set of faces.

Scale. 3 Chains to 1 Inch.

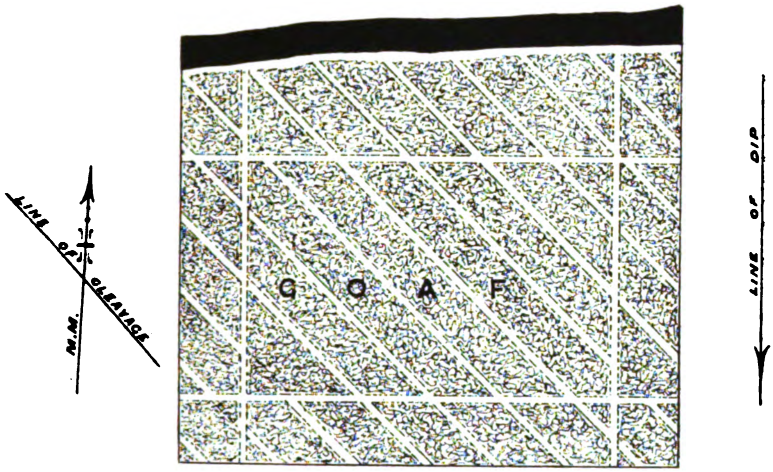


Fig. 249.—Sovereign Pit, West Leigh, near Manchester. Plan showing the Method of Working the West Leigh Five-Feet Seam.

Each pair employs 18 or 19 men to look after a face 700 yards in length. There is a proportion of one of these contractor's men to 3 colliers. About 85 per cent. of the faces work on end. In the pit are 74 colliers, who send out 400 tons of coal a day. About 53 of these employ a drawer, and 21 draw their own coal. Their working day extends from 5 A.M. to 3.30 P.M. One overman directs operations in the pit, in which, besides the workmen, 7 horses are employed. The cost for timber at the face is ½*d*. per ton.

The price paid for ordinary pit timber at the colliery is 9*s*. 8*d*. per 100 lineal feet, and 1*s*. 3½*d*. per cubic foot for larch. The roads are not dusty, but somewhat dry.

At the Radstock Collieries, Somersetshire, the thin seams of the Radstock group are worked on the Longwall system. The Somerset Coal Measures consist of two productive coal series, divided by a thick mass of almost unproductive hard grey sandstone and grit called Pennant. It is from 2,000 to 2,500 feet thick and separates the upper or Radstock from the lower or Kingswood and Bristol series of coal seams.

The upper series is locally subdivided into two groups, the upper group conby means of a "guss" and chain hauls out his load. The "guss" is a rope band worn round his middle, the chain is suspended from it in front, and a "crook" of iron is used for readily hitching it to the sledge or board ring. The chain is passed between his legs, and he goes on his hands and knees in a "thick" seam, but serpent-like, clutching at the props with his hands and bearing against them with his feet to help him on in the thin seams.

From the road-head larger sledges, which are piled up much higher than they can be along the stall, are taken out to the top of the incline by "twin boys," or boys running in the "twinway."

A small pulley is secured at a road-post or at one of the sleepers, and a chain passed round it, with one end attached to the back part of the loaded sledge near the road-head, and the other to the front end of the sledge at the foot of the topple. Only one tramway is laid in the topple, and that not of iron rails but wooden "crease." It is made similar in shape to the tram rails for keen-edged wheels, but in being laid is reversed, the vertical part being placed outside the horizontal. The sledges are shod with iron, and this "crease" forms a groove in which they slide, without running on wheels down steep places. When the loaded sledge is started at the top of the topple it proceeds slowly, followed by the "twin boy" till it comes into collision with the empty. The latter is then turned on its side and dragged on a yard or two whilst in this position by the full sledge in its farther descent. When it is clear of the loaded sledge the twin boy turns it over fairly into the "crease" and then leaves it to follow the load down as the empty proceeds upwards. At the foot of the topple, the sledge is "carriaged." A carriage is a skeleton frame running on wheels, made for the reception of the sledge. The twin boy pushes the carriage sufficiently away from the foot of the topple for a large board or stage to be dropped over the rails (iron) running along the level road, and called a "twinway." The carriage is then brought against the board, their upper surfaces being nearly on a level when so placed, the sledge drawn from the foot of the topple on to the board, and from there placed on the carriage. Here it is run out to the incline, where it is taken off the carriage and let down the incline accompanied by and attached to two or three others without wheels. A double line of wooden rails is used in the incline, and to prevent the lumps of coal dropping off the puts which are piled high, a chain is passed over them. At the foot of the incline the coal is transferred into tubs and taken out to the shaft by horses.

The top shot down in the roads is used to build the pack-walls on either side. These are mere facings, and the rubbish made in holing is stowed in the gob. In a thin seam, the waste will not hold all the rubbish, and some of it is loaded and sent out to the shaft. The thick seams hold their own rubbish, and in cases where it is not sufficient to closely fill the waste, it is thrown into "tumps," that is, it is built in the waste with alternate spaces and rough packs. The collier receives a rate per ton, which varies according to the thickness of the seam. He sets all his own face timber, and throws his rubbish back. He also rips the road and builds the packs there, but for doing so he receives in addition to the tonnage rate on coal a yardage price in proportion to the height he makes the road. The carting boy is not paid by the collier, but he and the twin boy both receive a tonnage price, varying in the one case in proportion to the thickness of the seam, and in the other to the distance the coal is taken. Timbermen are appointed who receive a daily wage for withdrawing the props at the face and building the tumps. A large number of the props are not recovered. When they are, they are struck out by blows from a hammer, and not by a "dog" and chain. The roads crush very much after being made, and require frequent shooting down to maintain them at their height. Sets of timber are used in the main roads. They are placed without any kind of notching, and are of various sizes according to the difficulty of keeping open any section of the road.

No cogs are used at the face or in any of the main roads.

In those areas where the seams lie flat, or at an easy inclination, instead of the puts used in topples all the roads are made 6 feet high and the horse takes the tubs to the face. If a pit at Radstock raises 150 tons of coal in a day of 8 hours, it is considered good work, but it must be remembered that rubbish is hauled as well, and the natural disadvantages to contend with are very great. The shafts are small, usually not exceeding 8 feet in diameter.

At the Kingswood Collieries, near Bristol, the seams of the lower series are worked by Longwall.* The seams are steep and have a bad roof, making them expensive to work.

Fig. 251 is a plan showing the method of working the Great Seam or Vein at Kingswood, the seam being about 4 feet 6 inches thick here. From the bottom,

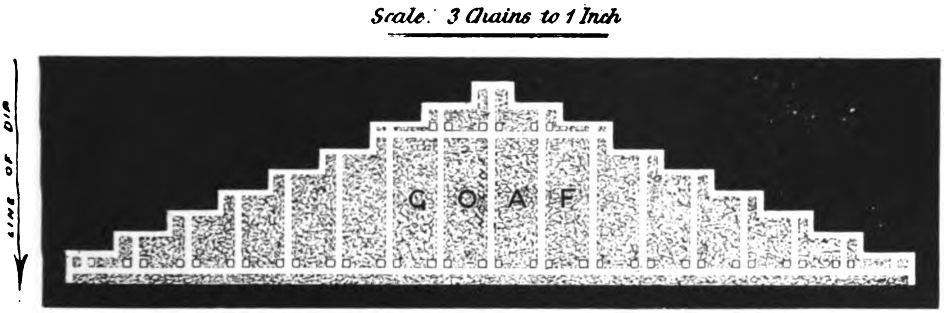


Fig. 251.—LONGWALL METHOD OF WORKING THE GREAT VEIN OF THE LOWER SERIES IN THE SOMERSETSHIRE COALFIELD AT KINGSWOOD, BRISTOL.

or main level, "hatchens," as they are called, are carried to the rise. At the corners of these hatchens, packs 4 feet square are built as shown in Fig. 251 on either side of the road. These points are made filling places, plates being placed so that the tubs can be pushed in round the corner clear of the rails. The top is not shot down in these hatchens, but the coal is brought down in one of three ways, determined by several conditions such as, inclination of the hatchen, scarcity of boys, &c. First, where lads are not employed, the coal is let down in shoots, and the supply of coal through it into the tubs is regulated by a hopper. Secondly, by making the hatchen a self-acting incline, a small pulley being used round which the chain passes and the pulley is shifted as the face advances. Thirdly, by lads taking "sleds" up and down. In this case a chain is secured at the top of the hatchen and lies in the middle of the road, the lads using it as a hand-rail to pull by in ascending the road, and to act as a "drag" in bringing the loaded "sled" out.

After being driven 44 yards, the hatchen intended for an incline is ripped and the rails are laid, a drum is fixed at the top, and the hatchen becomes a "Gug" or incline. A level road, one on each side from the top of this incline, cuts off the old hatchens from the lower level.

The levels are ripped right into the face, by one shift of rippers following two shifts of colliers. The face in a level road is carried 9 yards wide, 5 yards above and 4 yards below the road; the pack on the rise side is built 4 yards wide, and that on the deep side about 3 yards.

The packs built on each side of the hatchens vary from 3 to 4 yards wide. From the lower side of the main level a cross-cut is turned to the low side, and

* See Transactions, North of England Institute of Mining Engineers, vol. xxvii., pp. 96–97.

continued parallel with the main level as soon as there is room enough to give 5 yards of coal on the rise side. This cross-cut (not shown in Fig. 251) is ripped and packed precisely the same as the levels. Its chief use is as an intake for the air going inbye.

Two colliers work in one place. The air, after circulating round the face, passes into a higher district or by a cross-measure drift or branch into the overlying seam, called the Thorofare, on which, the roof being excellent, the returns are carried.

The Upper and Little Toad Veins are worked in a manner similar, differing chiefly in the length the hatchens are carried, which are 80 yards in the former, and 60 yards in the latter seam.

The roof of the Great Vein is so bad that the main roads are carried in the Little Toad Vein, and these roads are connected to the Great Vein workings by cross-measure drifts of 120 yards in length as occasion requires, the roads on the Great Vein being then abandoned and allowed to fall.

At the Allanshaw Colliery, Hamilton, Scotland, the Ell Coal Seam is worked by the Pillar and Stall or " Stoop and Room " method.

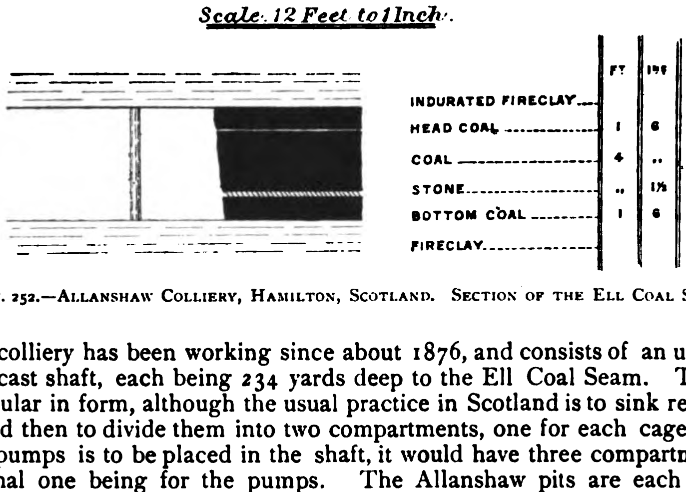


Fig. 252.—Allanshaw Colliery, Hamilton, Scotland. Section of the Ell Coal Seam.

The colliery has been working since about 1876, and consists of an upcast and a downcast shaft, each being 234 yards deep to the Ell Coal Seam. The shafts are circular in form, although the usual practice in Scotland is to sink rectangular pits, and then to divide them into two compartments, one for each cage; or if a set of pumps is to be placed in the shaft, it would have three compartments, the additional one being for the pumps. The Allanshaw pits are each 13 feet 6 inches in diameter. In 1881 the downcast solely was used for winding, although a pair of horizontal, high-pressure winding engines, with 26-inch cylinders and 5-foot stroke, had been placed at each shaft. In 1890 each shaft was used for winding. Single-decked cages, each carrying one tub or hutch which holds a ton, are used.

A Guibal fan 20 feet in diameter and 5 feet wide, exhausts the air at the upcast, and runs 40 revolutions per minute. At this speed it gives 40,000 cubic feet per minute with ·5-inch water-gauge. About 450 tons of coals are landed in a 10-hour day.

Fig. 252 shows a section of the Ell Coal Seam. There are 7 feet of coal, which parts badly from roof and thill or pavement. Over the coal are 4 feet of indurated fireclay, above which are 20 feet of rock. The pavement is composed of fireclay 6 feet thick.

Fig. 253 shows the method of working. The pillars are 20 yards wide by 30 yards long. The openings round them are 9 feet wide the short way and 12 feet wide the long way. The short way of the pillar faces the cleavage or cleat. The seam lies very flat, the dip being 1 in 20 to the North.

In the solid workings on a 12-foot wide place, a row of props or trees is set up, the timber. The colliers working off the pillars, or "stooping," as it is called in Scotland, use "Scotch gauze" lamps, and no blasting is allowed. The rails are laid in the road to about 3 feet from the coal, and a row of props set 3 feet apart, between the rails and the coal. On the waste side of the road are placed rows of props 14 inches apart, and about 3 feet between the rows. These trees are $4\frac{1}{2}$ inches in diameter and have small lids, 6 inches square and 1 inch thick, over them. Three men draw the timber from a lift, one of whom must be the fireman. In compliance with the Special Rules, he draws the props in the afternoon when his examinations are completed. Two of the men strike out the trees by blows from a hammer, whilst the third removes them to a place of safety on a timber tub made for the purpose. In a lift containing 300 trees, it takes about 3 hours to draw the timber.

At the Cowdenbeath Collieries, in Fifeshire, the Dunfermline Splint Coal Seam is worked on the Longwall system.

The collieries have been in operation since about 1851, the work at present being carried on at three shafts, the Nos. 3, 7, and 8. The last two are downcasts,

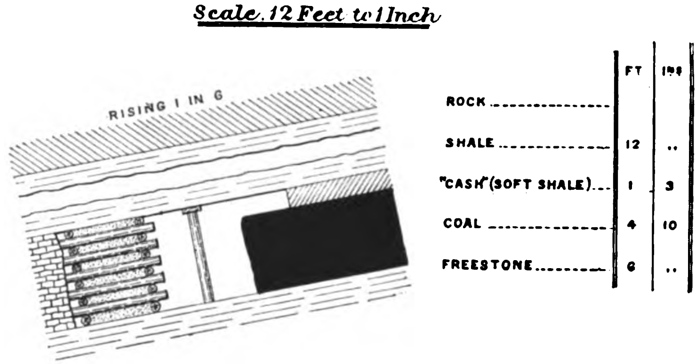


Fig. 254.—Cowdenbeath Collieries, Scotland. Section of the Dunfermline Splint Coal Seam.

and the No. 3 an upcast. The upcast is $\frac{1}{4}$ of a mile distant from Nos. 7 and 8 which are within 120 yards of each other. All the pits are of a rectangular form; No. 7, being 17 feet × 10; No. 8, 14 feet × 6; and No. 3, 14 feet × $5\frac{1}{2}$. The No. 3 Pit is 216 yards deep to the Dunfermline Splint coal, whilst No. 7 Pit is 270 yards deep to the same seam; and No. 8 Pit is 180 yards deep to the Lochgelly Splint and Parrot Seam.

A Guibal fan, 24 feet in diameter and 8 feet broad, is placed at the upcast shaft. It is driven at 60 revolutions per minute, and gives 50,000 cubic feet of air with 1·3 inch water-gauge.

Heavy pumping machinery is erected at both Nos. 7 and 8 Pits. The No. 7 Pit pumping-engine raises 800 gallons of water per minute, and that at No. 8 Pit raises 600 gallons. All the shafts are used to wind coal in, the total landings being about 800 tons a day. In 1881 the following seams were worked at the No. 7 Pit: the Dunfermline Splint 4 feet 6 inches thick; Five-feet, 4 feet 8 inches thick; Mynheer, 4 feet thick; and the Lochgelly Splint and Parrot Seam, 12 feet thick. The strata dip to the North at an inclination varying from 1 in 6 to 1 in 3.

Fig. 254 shows a section of the Dunfermline Splint Seam. There are 4 feet 10 inches of coal. Resting on it are 15 inches of "Cash," or soft shale, used by the colliers to hole in. Above this is the roof, composed of shale, which is 12 feet thick, and over this is rock. The pavement is freestone, about 6 feet thick. The shale over the seam makes a bad roof.

Scale. 3 Chains to 1 Inch.

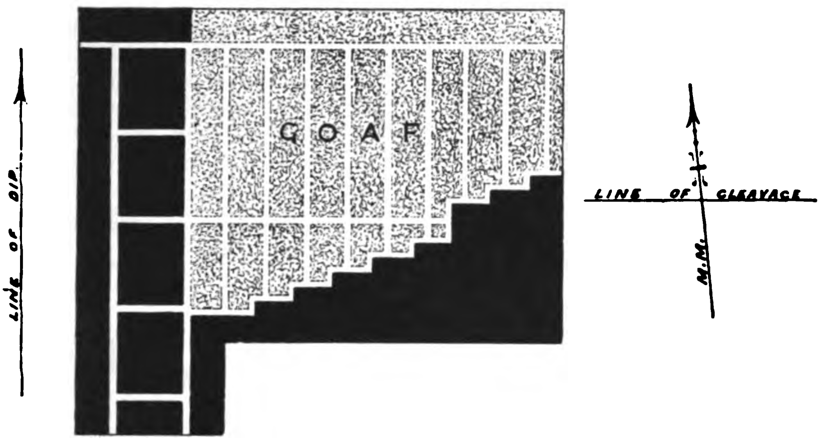


Fig. 255.—Cowdenbeath Collieries, East Scotland. Plan showing Longwall Working of the Dunfermline Splint Coal Seam.

Fig. 255 shows the method of working this seam. The levels are driven on end, or nearly parallel with the planes of cleavage, and the faces advance against the cleavage. The roads to the rise are driven at right angles to the levels, and they are turned every 14 yards from the level. Level roads cut off the rise roads every 60 yards.

Scale. 49½ feet to 1 Inch.

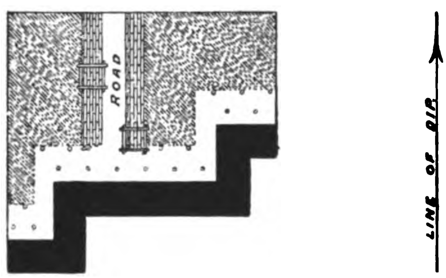


Fig. 256.—Cowdenbeath Collieries, East Scotland. Plan showing Face of Longwall Working in the Dunfermline Splint Coal Seam.

The men draw the coals from the faces to the wheel-braes or inclines, thence they reach the shaft by self-acting incline planes. Cut-chain inclines (fully described in Chapter IX. of this work) are used to let the coals down from the upper levels to the main wheel-braes.

It is found better in working the seam, to step the wall faces, keeping them 5 yards in advance of each other as shown in Figs. 255 and 256, on account of the heavy bad roof. The weight is thus confined to the limit of each wall, and the roof settles down more gradually than when a straight line of face is kept.

Two men work in a place, and besides hewing the coal, they put up all props, chocks and build the packs. They send out about 5 tons in a day of 8 hours, and receive 2*s*. a ton on large or round coal. Powder is not used in working this seam. Under the direction of the fireman, the colliers set their props where they think they are most required, there being no specified distance between them. The roof is so bad that withdrawing the props is dangerous and most of them are left in. The props are $3\frac{1}{2}$ inches in diameter, the lids over them being of broken props. The "cash" obtained in holing is used to build the packs, but it makes poor buildings, and, to strengthen the packs, chocks, 6 feet square, made of props and filled inside with rubbish, are placed on the sides of the roads, as shown in Fig. 256. When the roads are finished and abandoned, these chocks are taken out.

The fireman visits each place three times during his shift, and although he sets up no timber himself, directs the colliers to do so where he thinks they are necessary. The main roads are secured by props and sets of timber, or gears placed at no specified distance apart, but at distances considered necessary. The roof being bad makes the cost of timbering high. The cost per ton on round coal is 5*d.*, the price of pit timber at the colliery being 5*s.* per 100 lineal feet.

In the working of Thin Seams the most advantageous system is the Longwall, and an arrangement of roads should be designed to suit the inclination of the

Scale. 3 Chains to 1 Inch.

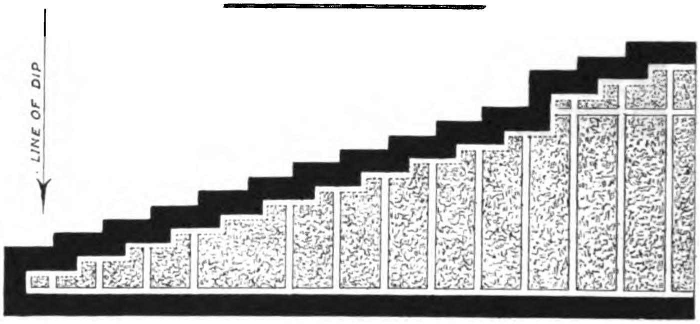


Fig. 257.—No. 1 Mode of Working Thin Coal Seams in Northern France and Belgium, adopted where the Inclination does not exceed Thirty Degrees.

seam or seams to be worked. The cost of working must of necessity be higher than in the case of thicker seams. Still, thin seams are successfully and remuneratively worked both in Great Britain and in other countries.

In Northern France and Belgium three different systems of working thin seams are employed.* No. 1 system, shown in Fig. 257, is an arrangement for seams in which the inclination does not exceed 30 degrees. Roads are carried to the full rise out of the chief level, the distance between these rise-roads being from 12 to 14 yards. The faces follow each other in step-like order, as shown on the drawing. At first the coal is taken down each of these rise-roads by means of a small self-acting inclined plane. When the faces have advanced a certain distance a new level is turned, which cuts off all the longer inclined planes, except one or two, which are retained as main roads. The lower level is at the same time advancing and opening out fresh ground, thus supplying more working faces. The main roads are ripped, and on the inclined planes tubs carrying 10 cwt. are used. A full description of the self-acting inclined plane in operation here is given in Chapter IX. of this work.

No. 2 system, shown in Fig. 258, is suitable for seams whose inclination ranges from 30 degrees to 60.

In this case, a succession of faces, in step-like order, are driven. These faces advance in the direction of the strike of the seam, not towards the rise, as in the No. 1 system. Where the seams are so highly inclined and the face advances to

* See Transactions, North of England Institute of Mining Engineers, vol. xxvii., pp. 174–180.

the rise, getting the coal is attended with a considerable amount of danger to the colliers, as large masses of coal fall out upon them during their work. A considerable loss arises, too, from the fact that the loosened coal lying on the floor slips downwards towards the waste by its own weight, and portions become lost amidst the rubbish.

In the No. 2 system, however, the danger and loss referred to are avoided. A reference to the sketch shows that the tramway is laid along the main level, and any coal loosened by the collier, or accidentally falling, rolls along the face to the tramway, where its progress is arrested. It is afterwards filled into the tubs and sent out. Each face is 20 yards long (measured along the slope), and four men work in it. For their own comfort and convenience these men place pieces of board across the floor horizontally from prop to prop, by this means regulating and controlling the descent of the coal along the face.

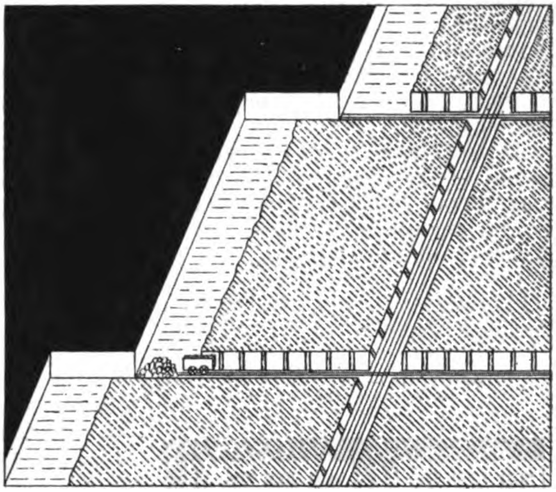


Fig. 258.—No. 2 Method of Working Thin Coal Seams in Northern France and Belgium, adopted where the Inclination ranges from Thirty to Sixty Degrees.

No. 3 system is suitable for seams in which the inclination varies from 60 degrees upwards.

The work here is stepped out into a series of faces one above the other. Each face is 6 feet high, and forms the working place of one man, as shown in Fig. 259. In this arrangement of steps the lower workmen are in advance of the higher, so that each man is protected against danger of anything falling on him from above, by the ledge of coal projecting 6 feet behind where he is working.

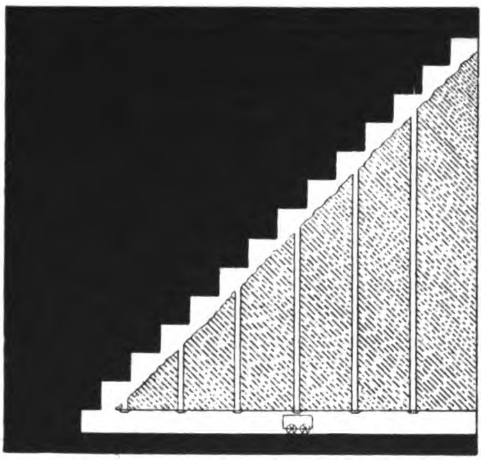


Fig. 259.—No. 3 Method of Working Thin Coal Seams in Northern France and Belgium, adopted where the Inclination varies from Sixty to Ninety Degrees.

The refuse yielded in working the seam and the inferior coal are thrown back behind the colliers, to fill up the waste. Where the seam does not yield sufficient rubbish to pack the goaf, the workmen stand upon scaffolds, formed by placing planks horizontally across the props.

The whole range of work consists of about a dozen steps, and at the bottom solid bed without bands or partings, and being of anthracitic quality is unsuitable for coke-making.

The method of working this exceptionally thick seam is shown in Fig. 263.

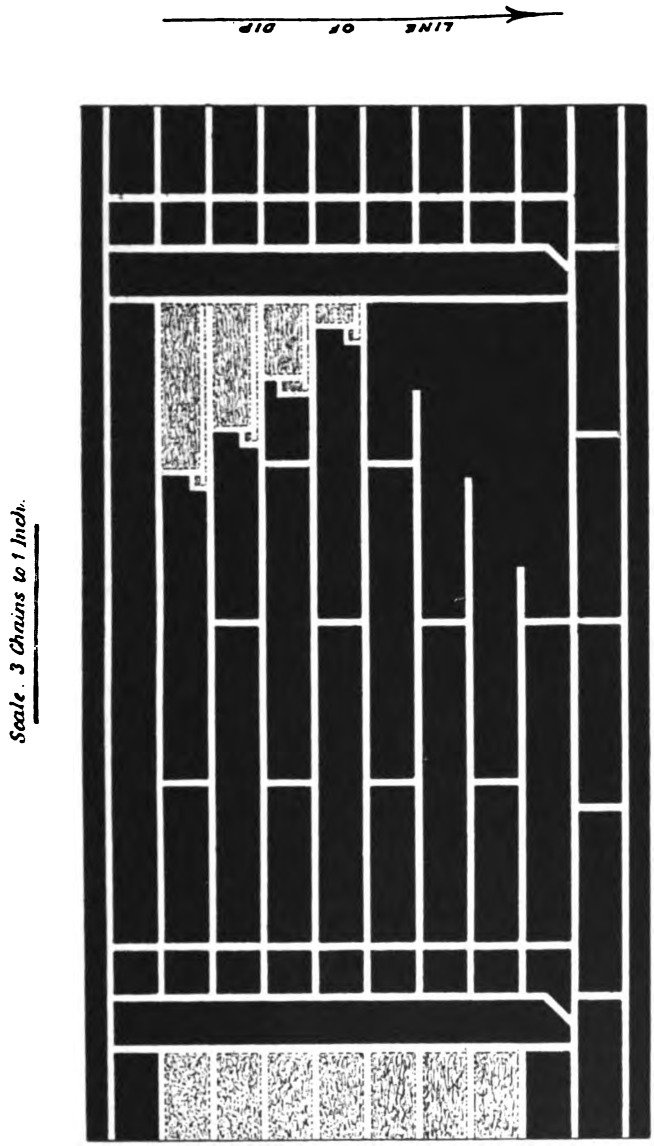


Fig. 263.—Plan showing Method of Working the Schuckmann Flötz Coal Seam at the Königin Luise Mines in Upper Silesia.

It is first divided into blocks of 220 yards by 14, with the necessary cross-holings for ventilation, by driving a preliminary network of roads in the lower portion of the coal, viz., next the floor. These roads are driven 8 feet square in cross-section, and the pillars or blocks formed lie with their longer sides parallel to the strike of the seam. Two or three men, assisted by boys, work together in

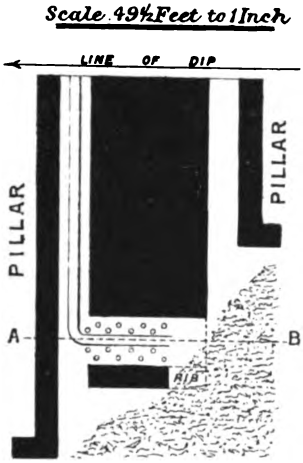


Fig. 264.

Fig. 265.

SHOWING METHOD OF WORKING A THICK COAL SEAM AT THE KLADNO PLAN MINES, NEAR PRAGUE, AUSTRIA.

and levelling of available routes taken in conjunction with the nature of the ground, whether mossy, soft and yielding, or rocky, level, or mountainous.

If the railway can be economically constructed without sharp curves and with easy gradients, there are great advantages in bringing the railway trucks by locomotives to the colliery sidings. If the branch line is worked by a railway company's locomotives the gradients and curves must be such as to comply with the regulations of that particular company, as must also the weight per running yard of the rails used, heights under bridges, &c. The public railway may be much above the colliery, and the connecting branch be so steep as to render a rope haulage desirable. Again, it may be considerably below, allowing the use of a self-acting incline all or part of the way, according to the configuration of the land and the expenditure incurred in the making. If power is obtained to cross public roads on the level and a locomotive is used, gates worked by a watchman at the crossing will be necessary, but if the crossing is worked by horses, protecting gates may not be required. Bridges over or under roads and over streams must be constructed to give the desired width for a single, double, or more lines of rails as may be desired. Soft yielding ground, deep cuttings through hard rocks, bridges, viaducts, tunnels, &c.. add to the cost of railway making and must therefore be avoided as far as possible.

The plan of a proposed railway shows its course throughout, the beginning, ending, and radius of all curves, the position and dimensions of all sidings, and the area of ground required to form the cuttings and embankments necessary for the proper width of the railway. If there are bridges these are shown, as also are the authorized road or stream diversions.

The section which accompanies the plan shows the surface line along the route in one colour, while in another the railway formation level is marked, and the height above datum line to both are lettered so that the heights of all cuttings and the depths of all embankments are clearly given. In the calculations made it is customary as far as possible to let the amount of cuttings equal what will just form the embankments, or, if not, provision must be made for obtaining or disposing of extra material. Each gradient and the length over which it extends is plainly marked throughout the section extending from the beginning to the termination of the railway.

The course and levels of the proposed railway are staked out on the ground and the construction is usually undertaken by a contractor.

After the cuttings and embankments have been made to their specified width, the latter usually subside a little, and the levels must be tested before the ballast is laid.

The ballast consists of gravel, broken stone or slag, or any other substance that is pervious enough to ensure perfect drainage. It must be hard enough not to be crushed or powdered by the passage of trains and to resist the action of frost. The size to which the stones or slag are broken is such that no stone shall exceed six ounces in weight. A uniform depth of about twelve inches of ballast is laid over the formation level of the railway, the sleepers are laid on this and the rails on the sleepers. The spaces between sleepers and at the sides are afterwards filled in with ballast, six or nine inches of upper ballasting being thus laid. The object of the ballast is to keep the road dry and hold the rails firmly in place. If water were allowed to remain between the rails they would sink. Longitudinal and cross drains are made where required within the railway ballasting, the latter communicating with drains leading into the side ditches. Side drains must invariably be formed in the excavations, and if found necessary in the embankments, they are formed at intervals down the slopes.

The rails are laid upon wooden or metallic sleepers, which are placed transversely on the under ballast and beaten firmly into place with wooden mauls. When the upper ballast is laid, if properly packed about the sides and ends of the and together with the sleepers having outside clips, hold the rail securely in place.

A sleeper with the chairs or gripping jaws formed out of its own material, and without loose pieces of attachment, is the simplest and most economical. Fig. 285 shows a steel sleeper of this kind which has been used in the collieries of South Wales. The rail may be canted into place between the rigid upturned metal, after which the wooden key is driven to hold it firmly in place. This sleeper allows of great rapidity in laying a tramway.

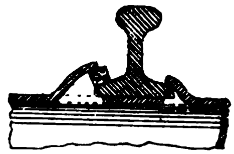


Fig. 285.—Steel Sleeper with Jaws formed on it for an Underground Tramway.

The drawback to the use of steel sleepers having smooth surfaces for horse-roads is the increased liability to horses slipping on them, but a better foot-hold for the horses is obtained if a corrugated form of sleeper is used.

Messrs. J. & F. Howard, of Bedford, who are makers of portable railway plant for mining and other purposes, have adopted the form of sleeper shown in Figs. 286 and 287. The sleeper is made of plate steel corrugated and flanged in order to give the greatest strength with the least weight of material. In constructing the sleeper the chairs for the rails are formed by pressure on the crown

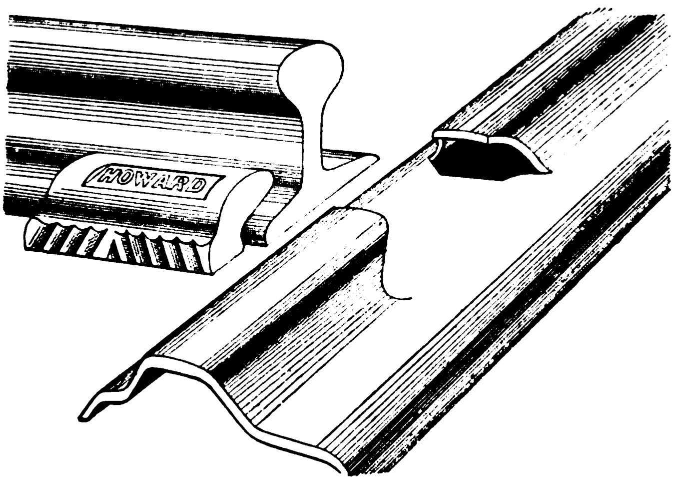


Fig. 286.—Steel Sleeper, Key and Rail.

of the sleeper, none of the metal being removed or cut away, but, on the contrary, the parts of the chairs upon which the rails are intended to rest are increased in thickness and durability. The chairs are formed on the sleepers by special hydraulic machinery with a precision of position in all sleepers that ensures accuracy of gauge being afterwards obtained. The rails rest upon the whole width of the sleeper, thus having a great bearing surface. No bolts, dogs, rivets, or fish-plates are required, the only fastening being the simple metal key shown in the drawing, and this is serrated on one side to fit into the cheeks of the chair when driven home.

For underground work a modified form of sleeper can be used, threaded on to the rails, and the use of keys is thus dispensed with.

The method of laying the line is shown in Fig. 287. The jointing sleepers are formed in the same manner as the single sleepers, except that two corrugations are rolled on one plate; the chairs or rail seats are pressed out of each corrugation in such a manner that the rail ends abut and are held in each seating respectively by its own serrated key. The keys are so arranged that either of them may be taken out without disturbing the others. This method of securing and joining the lengths of rails enables the laying down, and also the taking up, of the road for removal and re-layal to be effected with the greatest ease and expedition.

The corrugated form of sleeper allows the ballast to settle down very firmly under it, and the rail seats being below the crown of the sleeper, prevent any

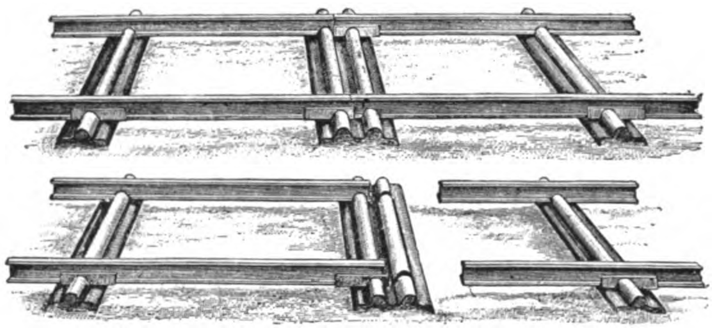


Fig 287.—Portable Railway, showing Jointing Sleepers.

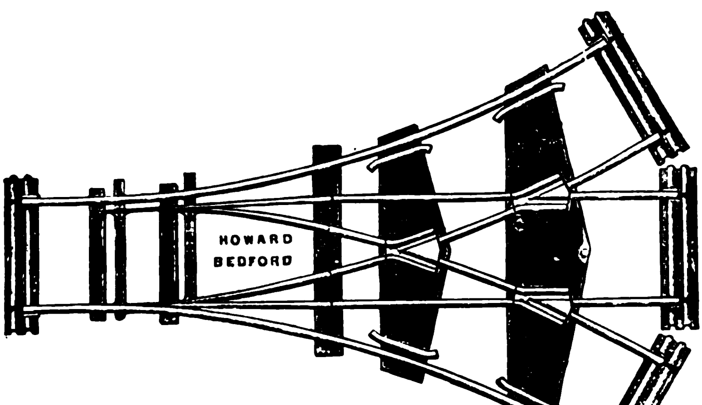


Fig. 288.—Three-way Points.

tendency for it to shift sideways on curves. For lines worked by horses this is the strongest form of sleeper, as the corrugation prevents any bending likely to be caused by the constant treading of the animals employed, and also gives a good foothold, thus preventing the slipping which so frequently occurs on flat-crowned sleepers.

The advantages of a portable line of rails of this form may be summed up as follows:—

Simplicity, strength, and durability.

Accuracy of gauge, which cannot vary, as the seats for the rails are formed by special machinery.

The gauge does not depend on the skill of the workman, and will be preserved as long as the line lasts.

No bolts, rivets, or spikes, with their many disadvantages, are required to fasten the rails to the sleepers.

A metal *safety key* fastens the rail to the steel sleeper, so that it cannot shake loose.

The rails may be laid down, removed, and relaid with despatch and without the aid of skilled labour.

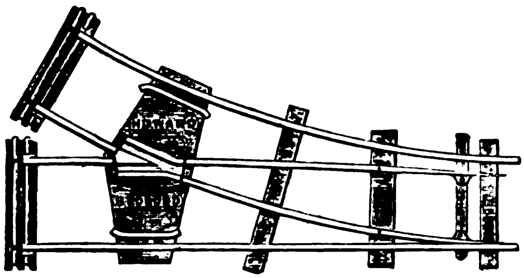


Fig. 289.—Right-hand Points.

The rails and sleepers not being riveted together, stow into small space, whereby cost of freight is reduced to a minimum.

In order to prevent corrosion, the patent sleepers, when at a certain temperature, are coated with an anti-oxidation compound, while for handling and shipment there is no fear of any damage to the sleeper or alteration in shape, nor are there any projections to break off. They are shipped ready for laying, a great consideration where skilled labour is scarce. The expansion of the metal of sleepers equally exposed to high temperature is not sufficient to destroy the accuracy of the gauge. The rails are supplied in lengths up to 21 feet, and of various sections according to the nature of the traffic.

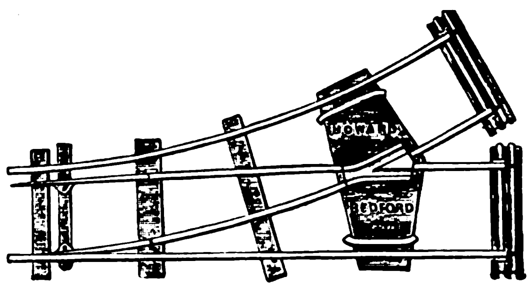


Fig. 290.—Left-hand Points.

If the line is to be used for light locomotives, the jointing sleeper is not used, the ordinary suspended fish-joint being preferable for main lines. The usual gauges for which these sleepers are made are 20, 24, 30 and 36 inches.

The points and crossings for main underground roads or for surface tramways are shown in Figs. 288 to 291. Fig. 288 is a three-way, Fig. 289 a right-hand, and Fig. 290 a left-hand set of points and crossings, while Fig. 291 shows the arrangement of a siding.

The sidings are formed, as shown in the illustration, by a set of points and crossings, either right- or left-hand, a section of curved line, and a number of sections of straight line.

Pass-byes in a straight line are made by a pair of right- and left-hand points and crossings, with two curve pieces and the desired length of line between. On single lines of rail of considerable length these pass-byes are necessary, and if a large number of trams have to pass to and fro they must occur frequently.

For special curves, which must be shaped on the spot, an ordinary rail bender, called a "Jim Crow," is required.

The points are similar to those described for surface railways, but for horse-power lines no switch boxes are required, and frequently the use of levers is positions of the points they are prevented from going too far by studs placed outside the rails, which ensure their proper alignment with the rails forming the continuation of the road. If the branch road leading from a main haulage road is only required for occasional use, the necessity for a check rail and a V crossing is sometimes obviated by the arrangement shown at F G in the same Fig. Here the inside rail of the branch road is not continuous, but is laid with a space reaching from F to G, where it crosses the main road rail. A striding rail of the right length is placed over the main line rail to bridge this space when required. A portion of the metal of an ordinary rail is cut away at the intersection of the two rails, and the striding piece fits into grooves at F G. This method raises the branch road two or three inches above the level of the main road. Sometimes, instead of a striding length, a short rail reaching from G to F, secured to the sleeper by an upright pin with counter-sunk head, is used. This rail may be moved on its central pivot so that its ends form a continuation of the rails laid for either the branch or main road.

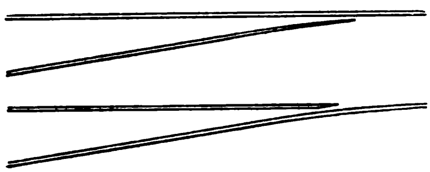


Fig. 293.—Fixed Points at a Pass-bye for Single Tubs.

Wherever lever-worked points occur on main roads an attendant must be placed to set the points in the required direction, and to receive and give signals and perhaps to change the ropes.

The turn-outs or pass-byes for horse traction are sometimes laid with fixed points (see Fig. 293). These are not suitable for journeys of more than one or two tubs, as each empty tub must be pushed over towards the road it is to go on as it approaches the points, or otherwise it is apt to take the wrong road. The points here do not interfere with the outgoing tubs, but at the other end of the

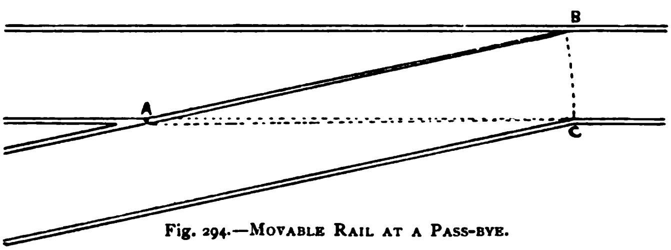


Fig. 294.—Movable Rail at a Pass-bye.

turnout, the same pushing over of the full tubs towards their road is necessary. Movable points are much better for long trains of tubs. If desired they may be counterbalanced.

A rough means of diverting the tubs at a pass-bye laid for a temporary purpose is shown in Fig. 294. It consists of a movable rail, A B, pivoted at A. The end at B can be moved by hand to the point C. The trams take one of two roads in accordance with the position this rail is moved into.

The points, rods, and levers used underground are usually made and fitted by the colliery blacksmith, the plate-layer assisting in the fitting. The crossings are cast, patterns for a right-hand V and diamond crossing, and of a left-hand V and diamond crossing, being kept for the purpose.

It is sometimes impossible to avoid the intersection at right angles of roads in the workings. Here it is impossible, without removing coal or the corners of buildings, to lay a three-way set of points and crossings which require room for the curves to be formed. For small trams the difficulty is met as shown in Fig. 295. The rails for the four roads terminate as shown in the drawing, the space about them having first been laid with wrought- or cast-iron plates. Before the plates are laid the floor is prepared with an even surface, over which planking is carefully laid to which to fix the plates. At the termination of the rails they are opened out a little, and raised guides are fixed curved in the contrary direction. By these means the entrance of the tub is facilitated. The iron plates allow of small tubs being guided over them in any direction, but the flanges of the wheels wear grooves in them sooner or later, and where these plates are laid at the pit bottom or pit top or at a self-acting incline, as they frequently are, the large amount of traffic in these positions soon results in grooves in the plates. To prevent the necessity of removing a whole plate, loose wearing pieces may be introduced which are quickly changed.

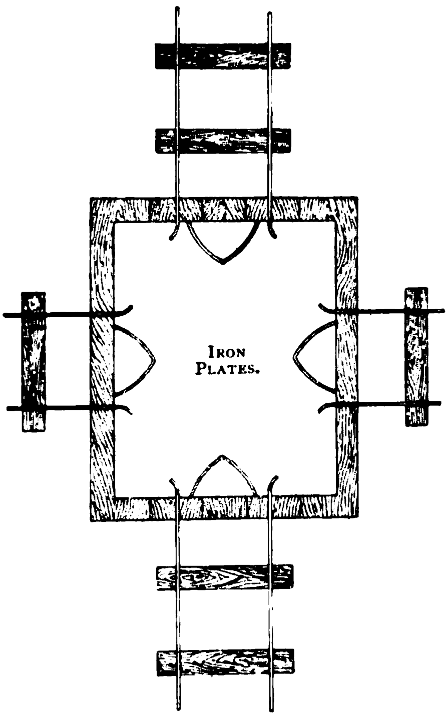


Fig. 295.—IRON PLATES AT JUNCTION OF ROADWAYS.

Heavy tubs are not so easily pushed about on the iron plates, and it may be necessary to use a turn-table. This can be made of any gauge, and when the tub is placed on the line of rail formed on the table top it may be turned on the table which revolves to the position desired. It is in confined positions that turn-tables are necessary. They are only permissible in positions where there is little traffic. A train of trams will pass a curve in less time than a single one can be turned on a table. For that reason on main roads it is necessary to take out sufficient coal, pack-walls or timbers at the sides, and round off corners to allow of the formation of proper curves, rather than to use turn-tables or iron plates, even for horse traction.

It is not practicable for the underground tramways to be laid with the same care and attention a surface railway receives. The rails must be laid in the roads, without the possibility of greatly changing the gradients. The most that can be done is to improve within reasonable limits of cost after making careful surveys and levellings of the roads intended for the main arteries of the colliery. The levellings will show swamps, hillocks, and minor undulations, which when removed will certainly slightly improve the gradients, which, however, cannot be materially altered in this way. The plans aid in showing how to improve or to form the curves necessary for branch roads and for the main road itself, but for the most part the windings of the road have to be followed. The curves on a surface railway are always arcs of circles of a certain radius, except at the extremities, where they join straight lines. The underground sinuosities are mostly made up of a series of short straight lines, at the most pronounced angles of which the rails are roughly curved with a "Jim Crow." Even when it is foreseen that a certain road will be required for an engine-plane, and it is in consequence kept straight for a time by using marks put up by a surveying instrument, for some reason or other, such as the crossing of faults or to alter the gradient, its direction is changed. The turns and undulations nearly always found in underground roads do not allow of the laying of tramways in them on correct principles, and consequently the speed of the trains is reduced to suit such lines as are possible. The sleepers are laid on the floor, which is made even to receive them, and the spaces between are filled with stones obtained in the rippings of roads, &c., and broken up for the purpose. If care is taken that the sleepers rest evenly on the bottom, the whole tramway may be securely carried thus, or with only such displacement as results from the squeezing of the roadway, the effect of which can only be guarded against by constant attention.

The size of tub used will depend upon the height of the seam to be worked, and it may be made of wood, iron, or steel. A very common form is to have oak framing below, the bottom and sides being made of $\frac{3}{4}$ of an inch or an inch oak or elm, the sides being strengthened by straps of iron. A drawbar of iron or steel passes the length from end to end, secured to the framing, and this bar has a hook at one extremity and a coupling chain at the other. If the tub has vertical sides, the wheels, which are flanged, are placed below, and are from 8 to 12 inches in diameter; but when the tub is narrower at the bottom than the top, the wheels may be set outside, and be from 15 to 18 inches in diameter. They may be made to turn on the axle or with it, the latter being the better plan if the roads are straight and the tubs are not run at a high speed, but where there are curves in the road, the loose wheels work better, particularly if a high speed round the curves has to be maintained. The axles are usually about $1\frac{1}{4}$ or $1\frac{1}{2}$ inches in diameter.

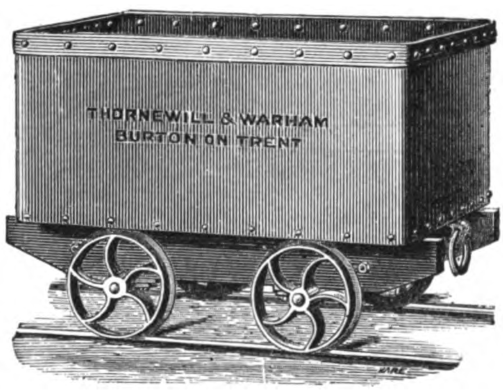


Fig. 296.—Colliery Tub.

Fig. 296 shows an ordinary type of colliery tub with iron or steel body, timber frames, and cast-steel wheels and axles as made by Messrs. Thornewill and Warham, Engineers, of Burton-on-Trent. The tubs are strongly yet lightly made, and of a carrying capacity varying from 5 to 20 cwts. The sides and bottom are connected by an internal angle-iron frame and the top edges are stiffened by a flat iron frame, both being riveted to the body sheets. The wood frame is bolted to the body and provided with cast-iron open bottom chocks to receive the axles secured by bolts. A draw-bar and links for coupling tubs together are fitted to the frame. The wheels are of a special quality of cast-steel and secured to the axles.

Tubs are made by Messrs. Thornewill and Warham of various sizes and types to suit the requirements of the different colliery districts, some being wholly of wood, with iron bands and fittings, others are wholly of iron; some have closed ends, as that in Fig. 296; others have one end open with two iron bars across.

In some districts tail-boards are used on the tubs in order to admit of filling and emptying them easily. Where tubs are subjected to heavy work arising from steep inclinations and undulations, or from passing through ill-kept, muddy, and water-logged roads, the wear and tear are excessive, and necessitate frequent repairs. At collieries where these circumstances prevail, it is found more economical to have tubs made wholly of wood, with iron bands and fittings, being easier to repair than those having iron or steel bodies, which are bulged surface. The smoke passing up the upcast deteriorates the shaft lining, and it is obnoxious in a pit used for the ascent and descent of workmen. Moreover, it is highly objectionable to have large underground fires in a fiery mine, and a source of great anxiety in any. Whatever system of rope haulage is adopted, the engines may be placed either on the surface or underground, but the engine for driving an endless chain should be placed underground. A chain is about three times as heavy as a rope of equal strength, and it adds very much to the power of an engine to have the additional weight in a deep shaft. Another objection to chains being in the shaft is that they are more liable to break than ropes.

Again, where compressed air is used as the motive power, this may be generated above ground and conveyed to the hauling engines underground. The system of haulage may be *direct* with one rope, which the weight of the empty tubs is sufficient to run back, or it may have a double line of rails, the empty train of tubs in that case slightly assisting the full train, or it may be by *tail rope*, which necessitates the use of two ropes, one to pull the loaded tubs out and the other to draw the empty tubs in. Then there is the *endless rope* system, which is capable of two applications, first with the haulage rope below the tubs, and secondly with the haulage rope over or by the side of the tubs. There is also the *endless chain* system.

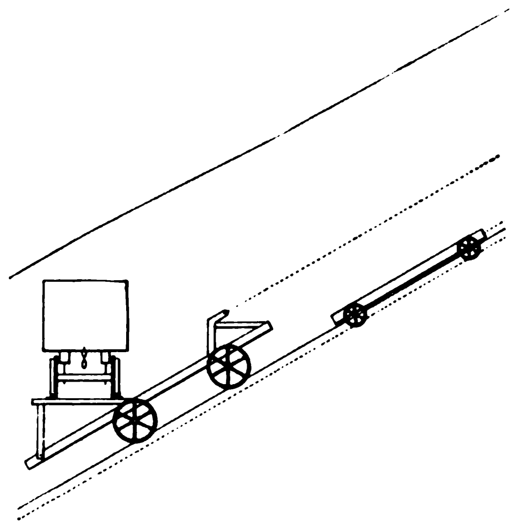


Fig. 308.—Counterbalance Arrangement of Self-acting Inclines in the Steep Measures of Northern France and Belgium.

Direct Haulage.—To apply the direct system it is necessary that the fall from the shaft be more than 1 in 30 to overcome gravitation; it need not be a uniform dip, but no part of the road should have a less inclination than 1 in 28. The engines for hauling the loaded tubs up this incline may be placed vertically or horizontally: single or double engines may do the work: the drum or drums may be placed on the crank shaft, or may be on the second motion with spur wheels. The best form is undoubtedly a pair of horizontal engines, and if the speed required is great, the engine should work direct as in the ordinary coupled winding engines; but if the speed desired is low, the advantage of spur gearing is that it enables a smaller engine to do the work. A decision on this point will be governed somewhat by the speed of piston in the engine. Mr. Percy's rule is to allow a piston speed of 350 or 400 feet per minute for an intermittent haulage as this would be, but for one running continuously it should not exceed 250 feet per minute. A speed of six miles an hour on the average may be safely taken, and much higher, up to 10 or 15 miles an hour, if the roads and tubs are in good repair and properly maintained.

A single road only is necessary for direct haulage, though a double road may be used if desired, and friction rollers should be laid at intervals of 8 or 10 yards throughout its length to carry the rope. The timber on which the rollers rest

No. 1 is a modification of the Tail Rope system. The rope has to be kept constantly tight, and this may be done at any convenient point by passing it round a pulley fixed upon a tram, to which a hanging weight is attached (Fig. 315); or the hanging weight may be replaced by a tram working on a short incline, and loaded sufficiently (see Fig. 316). Motion to the rope at the other end is facilitated by a deep-grooved pulley (see Fig. 317), or by a pair of ordinary large pulleys after taking the rope two or three times round each to cause friction. There are several ways of attaching the trains of tubs to the rope

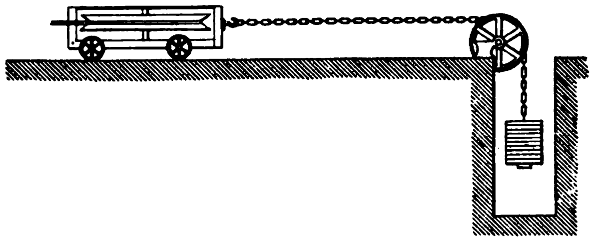


Fig. 315.—Tightening Arrangement for No. 1 Endless Rope Haulage.

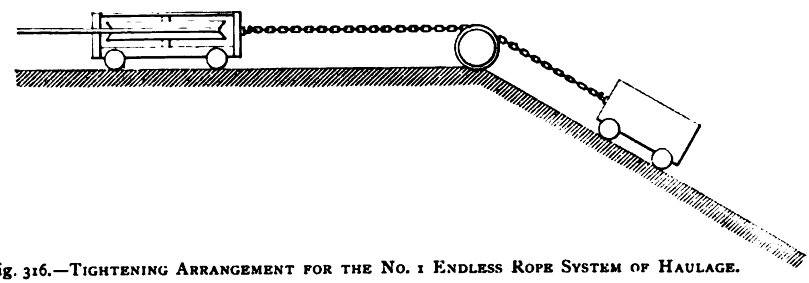


Fig. 316.—Tightening Arrangement for the No. 1 Endless Rope System of Haulage.

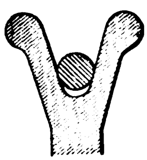


Fig. 317.—Deep-grooved Pulley for Endless Rope Haulage.

(which in the No. 1 system runs under the tubs) by a socket in the rope, and a short attachment chain (Fig. 318), or some form of clamp for gripping the rope.

The mode most commonly employed is that of having a bogie provided with a pair of grips, with which the attendant seizes the rope as required. In this case a train of tubs is taken at once.

A very effective haulage-clip is Hanson's. It is made in two types by Mr. Isaac Hill, George Street Iron and Brass Works, Derby. Figs. 319 and 320 show this clip as adapted for the bogie, whilst 321 and 322 show the form of clip suitable for attaching a single tub or a train of tubs to the rope.

The mode of adapting the clip to bogies, or the present existing bogies, will be chain then only passing half-a-turn round the wheel. The driving-wheels are placed horizontally, and motion conveyed from the engine to them by bevel gearing. The return wheels are ordinary sheaves, round which the chain passes half-a-turn. No tightening arrangement is needed, for the chain rests upon a fork in the tubs, which are always placed singly on the chain at distances varying from 10 to 40 yards apart, and it is better not to have them further than 30 yards apart, so that there may be no danger of the chain touching the floor. The

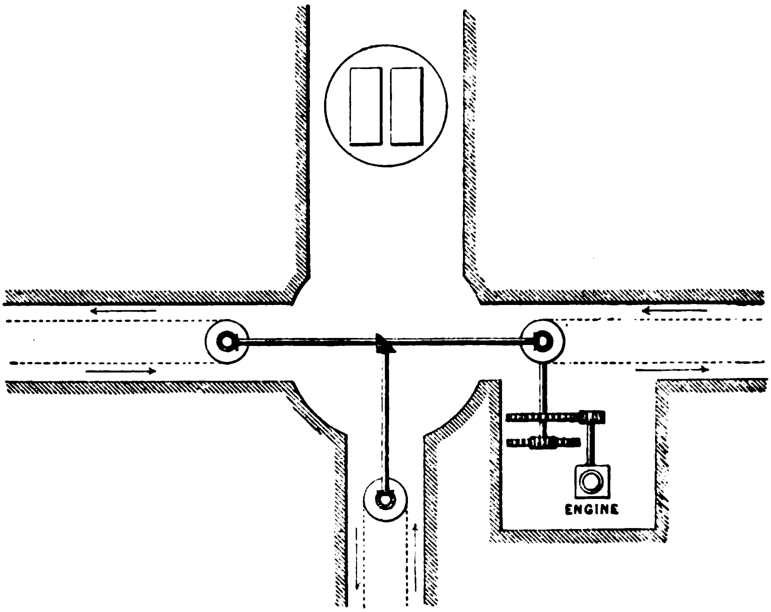


Fig. 328.—Endless Chain Haulage. Arrangement of Bevel Gearing from Hauling Engine to Roads leading from Pit Bottom.

speed at which the chain is driven varies from 1 to 4 miles an hour. Owing to the low speed of the chain, the condition of the rails is not of so much importance as with the Tail Rope and No. 1 Endless Rope, where the speed is greater, but care is required at the termini in the formation of the roads. On the "coming off" side the way is made to rise a few yards from the terminus, in order to obtain such a fall as will enable the tub, on leaving the chain, to acquire an impetus which carries it off the road; on the "going on" side the way is made to dip from the terminus, so that the tub may run to the chain without the aid of manual labour. Fig. 328 shows the arrangement of working roads leading in different directions from the pit bottom.

Every principal chain has its separate pulley and upright shaft, with bevel gearing at the top, and catch-box to put in and out of gear. All of them may thus be at work together, or any of them thrown out of gear at will. The handles to work the catch-boxes may be placed at each wheel, or they may be all placed together in the engine-house or elsewhere. Branch lines form no obstacle to the successful working of the endless chain. There are two ways of working these. A double pulley may be placed opposite the branch similar to that shown in Fig. 327 for working No. 2 Endless Rope branch. The main chain passes round one pulley, and the branch chain takes its power from the other, which has a catch-box arrangement for throwing into and out of gear. Another and perhaps owing to a broken chain. This block would be applicable only under certain conditions of road.

Care should be taken that the tubs are placed on the chain at such intervals as to ensure keeping the chain from touching the ground. No rollers are required for the endless chain; a regular gradient is unnecessary; a heavy dip at one part of the plane may be succeeded by a rise.

A plane may have many changes in its rate and direction of inclination, and where this is so, the energy arising from the counterbalancing effect of the undulating road is not all lost. It is quite possible, where there is a continual load upon the heavy part of a plane in favour of the full tubs, for the plane to self-act, even although some portions of the plane may be flat or dip in favour of the empty tubs. The first cost of fitting the road is heavier than for the Tail rope, as the chain costs more than the ropes, and a double line of rails is necessary. There are, no doubt, circumstances in which each of the different methods will recommend themselves.

SIGNALLING.—Until recently the method of signalling on engine planes was the same as on the self-acting inclined planes, by means of signal wire and hammers and levers.

If the engine was underground, a hammer was fixed in the engine-house. Attached to the hammer was a lever, from the end of which the wire was carried along the engine-plane and supported by iron hooks or staples driven at frequent intervals into the road-posts. If the engine was on the surface, the hammer was placed at a point near the bottom of the shaft convenient to the attendant who received the signals, and by means of another signalling arrangement communicated the signals he received to the engineman. At the terminus of the engine-plane and at any points between its ends from which it was desired to signal, levers were attached to this wire. By pulling the levers the hammer was raised and then (by releasing the lever) allowed to fall on to a flat piece of iron, making a distinct signalling sound.

This method has now, for the most part, given way to the electric signal. In the best form of this method, two wires are carried along the engine-plane throughout its course. The wires are about 6 inches apart, and are supported parallel to each other by insulators fixed to the road-posts or to the sides at intervals of about 20 yards; but this varies according to circumstances.

In the engine-house at the pit bottom, or wherever it is desired to receive the signal, an electric bell is fixed and a battery sufficiently powerful to keep the apparatus well supplied with an electric current. Having a return wire, signals may be given from one end to the other, or from any intermediate point, but can only be received at the electric bell. Through any portions of the plane which are subject to falling water, insulated wire must be used.

At any point on the plane, a signal may be given instantaneously by the "runrider." He has merely to bring the two wires together and rub them with his fingers, or connect the two wires by tapping them with a short iron rod. Contact is at once formed and the bell rings. It is as easy to give a signal at a long as at a short distance from the engine, and less effort is required to ring the bell than in the old wire and hammer system. The first cost for the wire is also less, as a light wire answers the purpose, while in cost of maintenance it compares favourably with the old system. Besides these advantages it is impossible to calculate the value of the convenience in signalling instantaneously from any point.

For convenience in repairing the road timbers, the wires should not be in and $\frac{2,670}{2} = 1,335$ square inches for each cylinder of a pair. $\sqrt{\frac{1,335}{\cdot 7854}} = 41\cdot 22$ as the diameter of the piston, so that a pair of engines will be required on the first motion with $41\frac{1}{4}$-inch cylinders and 30-inch stroke working a 5-foot diameter drum. But the cylinders and stroke are not proportioned to one another, and to find another proportion in which the area of cylinders × stroke will give the same power, proceed as follows :—Take the area of the cylinder at 1,335, and multiply this by the assumed stroke of 30 inches; $1,335 \times 30 = 40,050$. Now let the proportion of the stroke in the size, we shall find, be twice the diameter of piston.

Let x = the diameter in inches we wish to find.

Then $2\,x$ = the length of stroke in inches we wish to find.

Therefore $x^2 \times \cdot 7854 \times 2\,x = 40,050$ in order that the power may be equal to that of the size first ascertained.

$$x^2 \times 2\,x = \frac{40,050}{\cdot 7854}$$

$$2\,x^3 = \frac{40,050}{\cdot 7854}$$

$$x^3 = \frac{20,025}{\cdot 7854}$$

$x = 29\cdot 45$ or, say, 30 inches, as the diameter of the piston, and $30 \times 2 = 60 = 5$ feet as the length of stroke. To find the piston speed, 6 minutes is the time occupied by the train in travelling 3,000 feet, therefore the speed of the train on the average is $\frac{3,000}{6} = 500$ feet per minute. The drum has a circumference of 15·7 feet, therefore $\frac{500}{15\cdot 7} = 31\frac{3}{4}$ revolutions per minute; the piston travels 10 feet for one revolution, therefore $31\frac{3}{4} \times 10 = 317\cdot 5$ feet per minute of piston speed, which is under a fair allowable speed. Now to look at the strength of the rope taken, viz., 7 lbs. per yard. The net load on the rope is 15,354 lbs. or 6·85 tons. By the rule already given to find the circumference of a steel rope, whose working load is 6·85 tons, $C = \sqrt{2\cdot 4 \times 6\cdot 85}$ = a little over 4 inches, and the weight of a 4-inch circumference steel rope is about 7 lbs. per yard.

> *Question* 51.—A pair of hauling engines, 24-inch cylinders and 4-foot stroke, with a boiler pressure of 45 lbs. per square inch, and working on the first motion, by direct haulage, a plane 2,000 yards long, dipping from the shaft 1 in 9. What quantity of coal can be dealt with by such an engine, and what size drum would you use?

Assuming the maximum piston speed of 400 feet per minute, and that the trains run at an average rate of 8 miles an hour on a single line of rails, and that $\frac{2}{3}$ of the boiler pressure is effective, the power on the pistons would be $24^2 \times \cdot 7854 \times 30$ (the effective steam pressure) $\times 2$ (for the two cylinders) $= 27,143$, and deducting $\frac{1}{3}$ for frictional allowances $= 27,143 - 9,048 = 18,095$, travelling at 400 feet per minute. The tubs have to travel 8 miles an hour $= \frac{1,760 \times 3 \times 8}{60} =$ 704 feet per minute. While the piston travels $4 \times 2 = 8$ feet the crank travels $4 \times 3\cdot 14159 = 12\cdot 566$, say $12\frac{1}{2}$ feet, and while the circumference of the drum travels 704 feet, the piston travels 400 feet, and the crank travels $\frac{400}{8} \times 12\cdot 566$ = 628 feet. Therefore, as 628 : 704 : : 12·566 : 14·08 circumference of the drum

$= \frac{14 \cdot 08}{3 \cdot 14159} = 4 \cdot 48$ or $4\frac{1}{2}$ feet as the diameter. The load × 14·08 must equal the power. $\frac{18{,}095 \times 8}{14} = 10{,}340$ lbs., the load with which these engines are capable of dealing. To proportion this load so as to allow for weight of coals, weight of tubs, weight of rope and friction, proceed thus:—The steel rope will probably weigh about 5 lbs. per yard; $2{,}000 \times 5 = 10{,}000$ lbs., and this divided by the inclination $9 = 1{,}111$ lbs., which, deducted from the load, 10,340 lbs., leaves a balance of 9,229 lbs. This × 9, the inclination, = 83,061. Next allow $\frac{1}{28}$ of the gross load, viz., 83,061 + 10,000 lbs. the rope = 93,061 lbs. for friction, $\frac{93{,}061}{28} = 3{,}324$ lbs. Deducting this from the balance of $9{,}229 = 5{,}905$. Multiply this by the inclination $9 = 53{,}145$. The tubs when empty will weigh from $\frac{1}{2}$ to $\frac{1}{3}$ of the load they carry, say $\frac{1}{3}$, then $\frac{53{,}145}{4} = 13{,}286$ lbs. as the weight of the tubs, and $53{,}145 - 13{,}286 = 39{,}859$ lbs. as the weight of coal. The speed of the rope is 704 feet per minute, and the distance being 2,000 yards = 6,000 feet, $\frac{6{,}000}{704} = 8\frac{1}{2}$ minutes each single journey will occupy, and $8\frac{1}{2} \times 2 = 17$ minutes each double journey, and allowing intervals amounting to 8 minutes, that means $\frac{60}{17 + 8} = 2 \cdot 4$ journeys per hour. The load of coal is 39,859 lbs. $\times 2 \cdot 4 = 95{,}661$ lbs. $\frac{95{,}661}{2{,}240} = 42 \cdot 7$ tons per hour. Showing that, with a pair of engines, having 24-inch cylinders, 4-foot stroke, running 400 feet per minute, and drum $4\frac{1}{2}$ feet in diameter, with a boiler pressure of 45 lbs. per square inch, 42·7 tons per hour could be dealt with from a distance of 2,000 yards, the inclination being 1 in 9, but as $4\frac{1}{2}$ feet is rather small, it would be better to use a 6-foot drum.

Question 52.—What size and description of hauling engine would you erect to haul 370 tons of coal per day of $8\frac{1}{2}$ hours by the endless chain, up a plane 1,250 yards long, the average inclination of which is 1 in 27, being a rise for the full tubs, and tubs holding 6 cwt. of coal being used?

Supposing the tubs to be placed at intervals of 15 yards apart, $\frac{1{,}250}{15} = 83$ full and 83 empty tubs on the road. The weight of coal would be $83 \times 6 = 498$ cwt. = 24 tons 18 cwt., and as there are the same number of tubs on each line of rails they will balance each other. The load on the engine will, therefore, be 24 tons 18 cwt., and the friction of the tubs and chain and gearing.

On an inclination of 1 in 27, the actual work done by the hauling engine $= \frac{498}{27}$ = say 18 cwt. The friction of the tubs, assuming they weigh 3 cwt. each, and that the co-efficient of friction is $\frac{1}{28}$, will be $\frac{83 \times 2 \times 3}{28} = 18$ cwt. There would be $1{,}250 \times 2 = 2{,}500$ yards of iron chain, say of $\frac{5}{8}$ inch diameter, weighing 11 lbs. per yard = about 246 cwt. and $\frac{246}{28}$ = nearly 9 cwt. The friction for the coal is $\frac{498}{28} = 18$ cwt. This gives a total load upon the chain of 18 cwt. coal + 18 cwt. friction of coal + 18 cwt. friction of tubs + 9 cwt. friction of chain = 63 cwt., besides the resistance from the driving pulleys round which the chain has to

Suppose each tub to hold 10 cwt., this means 63 tubs, and if each tub weighs 4 cwt. it means $31\frac{1}{4}$ tons of coal, and $63 \times 4 = 252$ cwts. or 12 tons 12 cwts. of tubs = say 44 tons as the gross load. But the average inclination being 1 in 30, $\frac{44}{30} = 1\frac{1}{2}$ tons nearly as the net load. The ropes to some extent balance each other, but as it is usual to make the main rope stronger than the tail rope take this into account. Supposing a $2\frac{1}{4}$ lb. per yard steel rope to be used for the main and a $1\frac{1}{2}$ lb. per yard for the tail rope, then $2\frac{1}{4} - 1\frac{1}{2} = \frac{3}{4} \times 1{,}100 = 825$ lbs. as the weight of the rope, the net load of which would be $\frac{825}{30} = 28$ lbs. nearly. The net load exclusive of friction is $1\frac{1}{2}$ tons = 3,360 lbs. representing the tubs and coal + 28 lbs. representing the rope = 3,388 lbs. acting at the circumference of a drum whose circumference is $6 \times 3{\cdot}1416 = 18{\cdot}849$ feet. But add to this the friction. The ropes are $1{,}100 \times 2\frac{1}{4} + 1{,}100 \times 1\frac{1}{2} = 1{,}100 \times 3\frac{3}{4} = 4{,}125$ lbs., the tubs 252 cwts. or 28,224 lbs., the coal $31\frac{1}{4}$ tons or 70,000 lbs. $4{,}125 + 28{,}224 + 70{,}000 = 102{,}349$ lbs. total gross load. $\frac{102{,}349}{28} = 3{,}655$ lbs., the allowance for friction which added to $3{,}388 = 7{,}043$ lbs. the actual load upon a circumference of 18·849. $7{,}043 \times 18{\cdot}849 = 132{,}754$ the moment of load. A 4-foot stroke and an effective steam pressure of 30 lbs. $= \frac{132{,}754}{4 \times 2 \times 30} = 553$, the theoretical required piston area in square inches. Add $\frac{1}{2}$ for general resistance, making the required piston area 830 square inches for one cylinder. $\frac{830}{2} = 415$ square inches for each cylinder of a pair. $\sqrt{\frac{415}{{\cdot}7854}} = 22{\cdot}98$, or say 23 inches as the size of the pair of engines required, with a 4-foot stroke which is pretty well in proportion and 6-foot drums. Now to test the piston speed. The train travels 3,300 feet in 5 minutes $= \frac{3{,}300}{5} = 660$ feet per minute. The drums have a circumference of 18·849 feet, therefore $\frac{660}{18{\cdot}849} = 35$ revolutions per minute, and $35 \times 8 = 280$ feet per minute of piston speed, which is well under a fair allowable speed. The net load on the rope is 3,388 lbs. or 1·6 ton $\sqrt{2{\cdot}4 \times 1{\cdot}6} = 2$ inches nearly circumference, and the weight of a 2-inch circumference steel wire-rope is $4\frac{1}{4}$ lbs. per fathom or $2\frac{1}{8}$ lbs. per yard, so that the rope chosen weighing $2\frac{1}{4}$ lbs. per yard is quite strong enough for the purpose.

Question 54.—A train of 10 tubs, ascend an incline, each tub (with coal) weighs 1 ton with a rise of $4\frac{1}{2}$ inches per yard, what is the power required and the strain on the rope?

Assume that the friction of the tubs is $\frac{1}{28}$th of their weight. A rise of $4\frac{1}{2}$ inches per yard $= \frac{36}{4\frac{1}{2}} = 1$ in 8, 10 tubs $\times 2{,}240 = 22{,}400$. $\frac{22{,}400}{28} =$ 800 lbs. necessary to overcome friction, and $\frac{22{,}400}{8} = 2{,}800$ lbs. necessary to overcome gravity, which would be the power without considering friction; the amount of strain on the rope would be therefore $800 + 2{,}800 = 3{,}600$ lbs. = 1 ton 12 cwts. 0 qr. 16 lbs. It often happens that $\frac{1}{70}$th of the weight is taken for the friction, if that were adopted it would mean $\frac{22{,}400}{70} = 320$ lbs. for friction. 2,800 lbs. for gravity as before = 3,120 lbs. total = 1 ton 7 cwts. 3 qrs. 12 lbs., but the former allowance is preferable as being safer.

The force due to friction would be $\frac{70}{70} = 1$ cwt., hence the working load of the rope would be $31\cdot1 + 1 = 32\cdot1$ cwt. or $32\frac{1}{10}$ cwt.

Question 59.—At what gradient would the full tubs hold the empties in suspension, the set to consist of 8 full tubs weighing 11 cwt. each and 8 empties weighing 3 cwt. each, friction $\frac{1}{70}$th?

Equilibrium will ensue when the sum of the friction of the two loads is equal to the difference of their gravity on the incline. Supposing in the question asked that the weight of each load respectively includes the weight and friction of rope, friction of rollers, &c., connected with it; then—

Let 1 in x denote the gradient;

$$\text{then } \frac{11 \times 8}{x} - \frac{3 \times 8}{x} = \frac{(11 \times 8) + (3 \times 8)}{70};$$

$$\therefore \frac{88}{x} - \frac{24}{x} = \frac{112}{70}$$

$$\text{and } \frac{64}{x} = 1\cdot6$$

$$64 = 1\cdot6\,x$$

$$x = \frac{64}{1\cdot6} = 40.$$

Therefore the gradient would be 1 in 40. Put as a formula for the above rule, $(F - E)\,R = (F + E)\,G$, where F stands for the full loads, E for the empty ones. R the denominator of the fraction of the friction, the numerator of which is always 1, G the denominator of the fraction of the gradient, the numerator of which is always 1.

Question 60.—What ought to be the gradient of a horse road when the loaded train consists of six tubs 18 cwt. each, and six empties of $4\frac{1}{2}$ cwt. each, friction $\frac{1}{70}$th, so that the resistance may be equal both ways?

In this case resistance is equal when the friction of the full load, less its gravitation, is equal to the friction of the empty load, plus its gravitation.

Let 1 in x denote the gradient;

$$\text{then } \frac{108}{70} - \frac{108}{x} = \frac{27}{70} + \frac{27}{x}$$

$$\therefore \frac{27}{x} + \frac{108}{x} = \frac{108}{70} - \frac{27}{70}$$

$$\frac{135}{x} = \frac{81}{70};$$

$$x = 116\tfrac{2}{3},$$

the gradient therefore is 1 in $116\frac{2}{3}$. Express this rule by a formula, using the same expressions as in the last example, as follows—

$$\frac{F}{R} - \frac{F}{G} = \frac{E}{R} + \frac{E}{G} \therefore (F + E)\,R = (F - E)\,G.$$

Question 61.—What are the usual methods of determining the size of the high and low-pressure cylinders of compound engines, and how is the horse-power of such engines calculated?

It is necessary, in the first place, to determine the rate of expansion in compound engines, then if A = area of low-pressure cylinder, a = area of high-pressure cylinder, E = rate of expansion $= \frac{A\,L}{a\,l}$, L = length of stroke in feet,

$\frac{53,760 + 79,200}{28} = 4,749$ lbs. the allowance for friction. $9,900 + 4,749 = 14,649$ lbs. as the actual load upon a circumference of 18·849. $14,649 \times 18·849 = 276,119$ the moment of load. Assuming a 5-foot stroke and 30 lbs. effective steam pressure $\frac{276,119}{10 \times 30} = 920·4$ theoretical piston area, to which add $\frac{1}{2}$, $920·4 + 460·2 =$ say, $1,381 = 691$ for one cylinder and $\sqrt{\frac{691}{·7854}} = 29·66$, or say 30 inch cylinders. The size of the engines is not affected by the gearing, this would only affect the piston speed maintained in doing the work. To consider the possibility of gearing the engines, the drums are to be 6 feet in diameter and have a circumference of 18·849 feet. The speed of the ropes is $\frac{4,500}{13} = 346$ feet per minute, and therefore the drums revolve $\frac{346}{18·849} = 18·4$ times per minute. The piston speed must not exceed 400 feet per minute, and as the stroke is 5 feet, $\frac{400}{5 \times 2} = 40$ revolutions of the engine. The proportion of gearing must therefore not exceed 18·4 to 40 or say 2 to 1, and it would be better not to use gearing at all with 6-foot drums, in which case the piston speed would be $5 \times 2 \times 18·4 = 184$, thus allowing a margin for occasional quicker speed following delays through accident.

Question 64.—How do you proceed to calculate the friction of tubs?

By allowing the tubs to run down an incline with a regular gradient, taking the length of the incline and noting the time occupied by the tubs in traversing it. The gradient must, of course, be ascertained if not known, and the experiment may be tried on either full or empty tubs.

Resistance due to friction $= \frac{H}{D} - \frac{D}{T^2 \times 16\frac{1}{12}}$ where H = height of plane in feet, D = length of plane in feet, T = time in seconds taken by tub to traverse the distance D.

Supposing a tub traversed a plane 150 feet long and having a fall between the two ends of 5·74 feet in 20 seconds, the resistance due to friction is found thus:—
$\frac{5·74}{150} - \frac{150}{20^2 \times 16\frac{1}{12}} = ·01495$, so that if the loaded tub weighed 1,804 lbs., $1,804 \times ·01495 = 26·97$ lbs. and $\frac{26·97}{1,804} = \frac{1}{66·9}$ say $\frac{1}{70}$th as the co-efficient of friction.

If an empty tub traversed the same plane in 23 seconds and the tub weighed 688 lbs., then $\frac{5·74}{150} - \frac{150}{23^2 \times 16\frac{1}{12}} = ·02063$ and $·02063 \times 688 = 14·2$ lbs. and $\frac{14.2}{688} = \frac{1}{48·45}$ say $\frac{1}{48}$th as the co-efficient of friction.

Another way of finding the co-efficient of friction is by placing the tub on a perfectly level piece of the road laid in the ordinary way, and attaching one end of a cord to the tub, the other with a weight at the end is passed over a pulley, and whatever weight is found sufficient to keep the tub in motion without accelerating its speed is taken as the amount of friction, and this weight divided by the weight of the tub operated on gives the co-efficient of friction.

Another method consists in starting a tub down a piece of road which becomes gradually flatter and observing where the tub comes to rest. The average gradient between the starting and resting points shows the ratio of the friction to

CHAPTER X.

DRAINAGE.

Winding Water up Shafts—Lifting and Forcing Pumps—Making Pipe Joints Watertight—Balance Bobs—The Windbore—Clack-piece—Fish-piece—The Working Barrel—Action of the Pumps—Water Speed in Pipes—Construction and Method of securing Pumps—Preserving Pipes from the Action of Mineral Water—Pump Rods: Their Material; Method of Joining: Steadying; Safety Catches—Attachment of Bucket to Spears—Hanging Clack and Bucket Doors—General Arrangement of Lifting and Forcing Sets of Pumps—Determination of Weight Necessary for Balance Bobs—Use and Action of the Air Vessel—Bunton and Plank Brattices to form compartment in Shaft for Pumps—Arrangement for a Sinking Set of Pumps—Sliding Suction for Sinking Set—Messrs. Thornewill and Warham's Details of Pump Work—The "Deane" Sinking Pumps—The Cornish Pumping Engine—The Cornish Double-beat Valve—Davey's Differential Valve Gear for Cornish Engine—Davey's Compound Differential Pumping Engine—Relative Advantages of placing Pumping Engines Above and Underground—Steam Pumps—The Compound Differential Engine as arranged underground—The Worthington Pumping Engine—Compressed Air for underground Pumping Engines—Hydraulic Pumps—Moore's Hydraulic Mine Pump—Wire Rope Systems of Pumping—Electrical Pumping Plant at the Trafalgar Colliery—Syphons for Drainage of Underground Roadways—Memoranda—Powers of Engines for given work—The Pulsometer, its Action and Use—Pulsometer arrangement for Draining Underground Workings—Calculation of Contents of Water Barrels.

THE water which finds its way into the underground workings must be removed. In the case of an adit driven at a proper rise and having no workings to the dip, the water will run out without trouble; but in the case of a pit with workings far enough beneath the surface, the water is removed either by pumps placed in the shaft, or by tanks placed in the cage, to be used when the pit has done drawing coals for the day. Unless the quantity is very small indeed, it is found an advantage to have pumps, for with only 20 galls. per minute to be removed it would take more than 6 hours winding, lifting one ton at a time at the rate of 20 tons per hour, giving 3 minutes to fill, wind the cage, and empty. There are two kinds of pumps used—the *lifting* and the *forcing*. In the lifting pumps, a bucket works in the working barrel, the part of the pump below the bucket is called the "suction," and above it the lift. The suction should not exceed from 20 to 24 feet in height. The height of the lift, limited by the strength of the material and the weight of the spears, does not usually exceed 50 fathoms. Spears or pump-rods connect the bucket working in the barrel with the beam of the engine, and as they are inside the pipes conveying the water up the shaft, they reduce the area of these for the water, and so increase the friction. Lifting pumps are not suitable to great depths on account of the wear and tear to the leather rings forming the packing of the bucket, necessitating frequent change, and costing a considerable amount for repairs. In the forcing or plunger pumps the plunger pole, or ram, works through a stuffing box into a plunger case of bored cast-iron, and at every down stroke the water is forced upwards through an upper clack into the column of pipes above, and these pipes in the plunger pumps are quite clear for the ascent of the water.

The advantage of the forcing method is that there is less wear and tear, the hemp packing of the stuffing box is preferable to the leathers of the bucket, as giving less friction and being more durable. It is not necessary to remove the

Reference.

t, is the clack in its seat in the lifting set, and *s* is the clack-door.
r, is the bucket working in the working barrel *o'*.
o, is the rising main of the lifting set, and *u* the windbore.
n, is a wooden trough by means of which the water is delivered into the cistern *v*, from which the forcing set takes its water.
f, is the set-off and iron straps from the main rods *c*.
In the lifting set that part of the pipe between the bottom of the working barrel and the top of the clack-piece is usually slightly reduced in size, and this allows of a drop valve with bow being dropped down into position after the bucket has been drawn up through the rising main, if it should ever become necessary.
g, is the ram of the forcing set, and *e* its set-off and iron straps by means of which it is secured to the main rods *c*.
b, are spears working through guides *a a*, to ensure the plunger working truly in its case; the spears *b* need only be long enough for the engine to make its stroke.
l, is the H piece between the plunger and clack-pieces.
m and *k* are the two clacks in their seats, *p* is the windbore receiving water from the cistern *v*, and *d* shows the rising main of the forcing set.

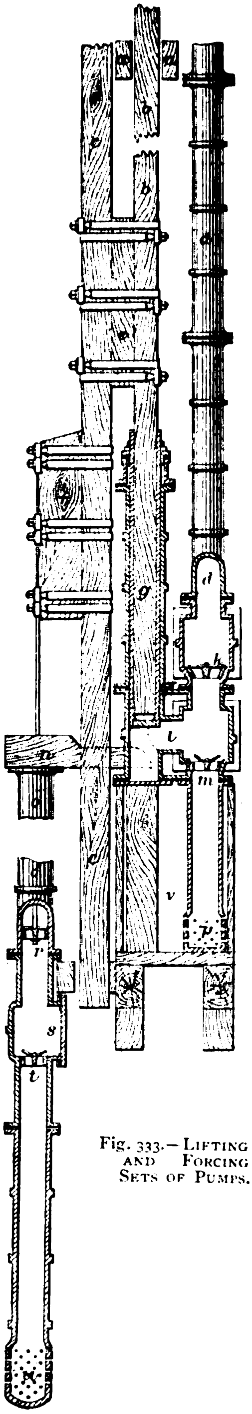
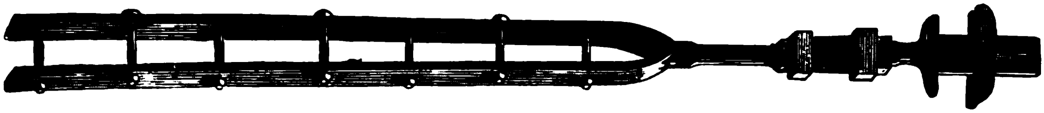


Fig. 333.—Lifting and Forcing Sets of Pumps.

bottom clack opens, and the water from the cistern follows its course, the water in the column of pipes by its weight at the same time closing the upper valve. The diameter of pipes which form the rising main and the thickness of metal in the pipes depend upon the quantity of water to be raised, and the height it has to be lifted. These must be calculated. In order to reduce the shocks to which all pumps are liable the velocity of the water in the pipes should not exceed 240 feet per minute, and it will be better to limit it to 200 feet per minute. The pipes should be cast of a uniform thickness, and have brackets under the flanges, as well as a belt round the pipe at the socket end so as to better resist the strain in screwing up the joints. All the pipes should be faced, and have just sufficient spigot to keep the ring in position. They are usually made 9 feet long. To preserve the pipes from the action of the mineral water (if such is being raised), they are lined with a thin casing of wood. The pumps are kept steady by collars placed across from the buntons to the side of the shaft at each alternate pipe, so that these collars would be 18 feet apart. The pump rods or spears are usually made of Memel or pitch pine, square in section, and must be as sound and free from knots and faults as possible. They should be of uniform lengths, so that a spare rod will fit anywhere, and may be from 30 to 45 feet long. The lengths are put together by scarfed joints (Fig. 334), and secured by stout wrought-iron plates and bolts. Sometimes these plates are placed on two sides only, but often a plate is placed on each of the four sides. Where the plates are single there should be clinch-bolts a little above the other



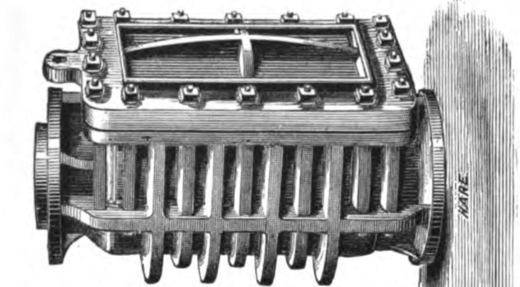


Fig. 343.

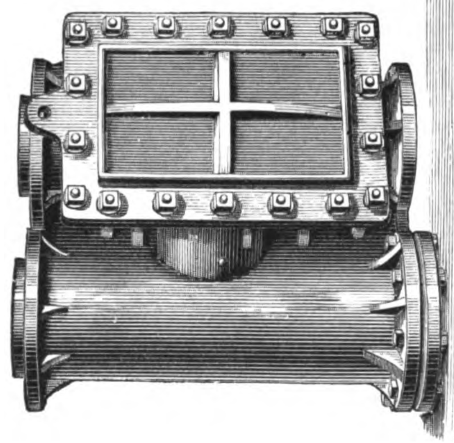


Fig. 342.

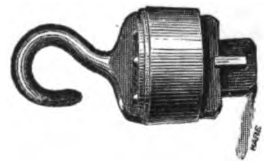


Fig. 347.

Fig. 346.

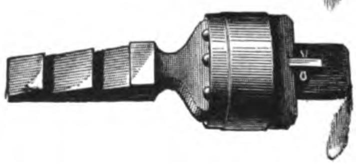


Fig. 344.

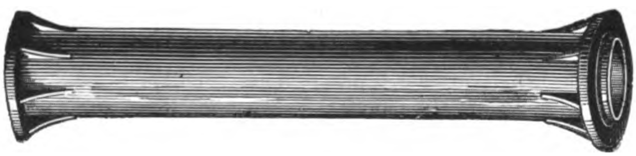


Fig. 341.

Messrs. Thornewill and Warham's Details of Pump-Work

Notches are cut out of the stringing planks every 3 feet apart for the purpose of receiving other buntons, which are placed horizontally from the one stringing plank to the other across the shaft. These horizontal buntons are nailed to the stringing planks, and a cleading of fir boards from 1 to 2 inches thick is nailed vertically to the horizontal buntons. The fir boards are either planed smoothly at their edges to ensure a close fit, or if one compartment of the shaft is to be used as a downcast and the other as an upcast, to prevent leakage, these boards may have a groove ploughed in their edges, and a slip of wood which fits in the groove is inserted as the brattice is progressing.

In the plank brattice, no horizontal buntons are used, and the stringing planks, instead of being fixed singly, one on each side of the shaft as in the last case, are placed in pairs on either side of the shaft, a space of 3 inches being left between them. Memel planks 3 inches thick are then guided down edgeways through the space formed on the opposite sides of the shaft by the stringing planks to form the brattice. The brattice planks are kept in position by having their edges planed smooth and by using iron dowels, or the joints may be made true as in the manner described for bunton brattice. The plank brattice makes a stronger and more permanent division of the shaft than the bunton brattice.

Pumps for a sinking pit are somewhat different from the permanent arrangement. Either the whole set of pumps or the bottom part must be hung in the pit so as to follow the progress of the sinking. A lifting set is invariably used in the bottom of a sinking pit; and as the sinking progresses over 40 fathoms, a permanent forcing set may be placed at that level, and the lifting set used again for another stage, supplementary spears to work the bucket being attached to the main spears. A method of hanging the pumps in a sinking pit is by two ground spears, fixed one on each side of the set by iron collars. At the top of each of the ground spears is one of a pair of 5- or 7-fold blocks called ground blocks, the other being placed on buntons at the top of the pit. Through these blocks a pair of ropes are rove, the bottom ends being connected to the ground spears and the surface ends of each being taken to a ground crab. The ropes are called ground ropes. The pumps are steadied in their position by means of temporary buntons or collarings. The top pipe or pump is called a "hogger." It is bell-mouthed on the top, and just below at the side is provided with a flexible hose to accommodate itself to the varying height of the column. As the sinking proceeds, the pumps are lowered by the ground crabs, and when the hogger pump is down nearly to the delivery drift, it is taken off by means of the main crab, and lifted over the top of the spears. To allow of this being done readily, instead of the spears being attached to the engine beam, they are clamped to the front of a piece of wood called a Y. A length of pump is then added to the column, a length of spears being added as required, and the hogger pump replaced.

To avoid hanging the whole column, an arrangement, whereby the suction and three following pipes only are suspended, and all other parts are fixed by buntons, is often used. This is called a "sliding suction," that part being made telescopic. When it is necessary to add a pipe, it is put in by breaking the joint above the bucket door and lowering the parts below this to admit of the new pipe. The engine must not be driven faster than is necessary to keep down the water, or too much air will be drawn into the pumps to the injury of the working parts. If it is desirable at times to lower the water below the level of the upper holes of the windbore, these should be carefully plugged first with soft wood.

Figs. 340 to 349 show details of pump-work made by Messrs. Thornewill and Warham, Engineers, of Burton-on-Trent.

Fig. 340 is a sliding windbore used in connection with a bucket set when sinking. The snore-piece is made very strong to prevent its being broken by pieces of rock striking it when the sinkers are blasting, and it has a hand-hole fitted with the Cornish as well as to other engines, thus doing away with the tappet gear. A perspective view of it is shown in Fig. 357. A lever (*a*) called the main lever, gives motion to the valves through a rod (*b*). The motion of the engine is given to the outer end of the lever through the rod (*c*), by means of a lever of the first order, the long end of which is attached to the plug rod or any moving part of the engine, where it gets the motion of the piston on a reduced scale; the other

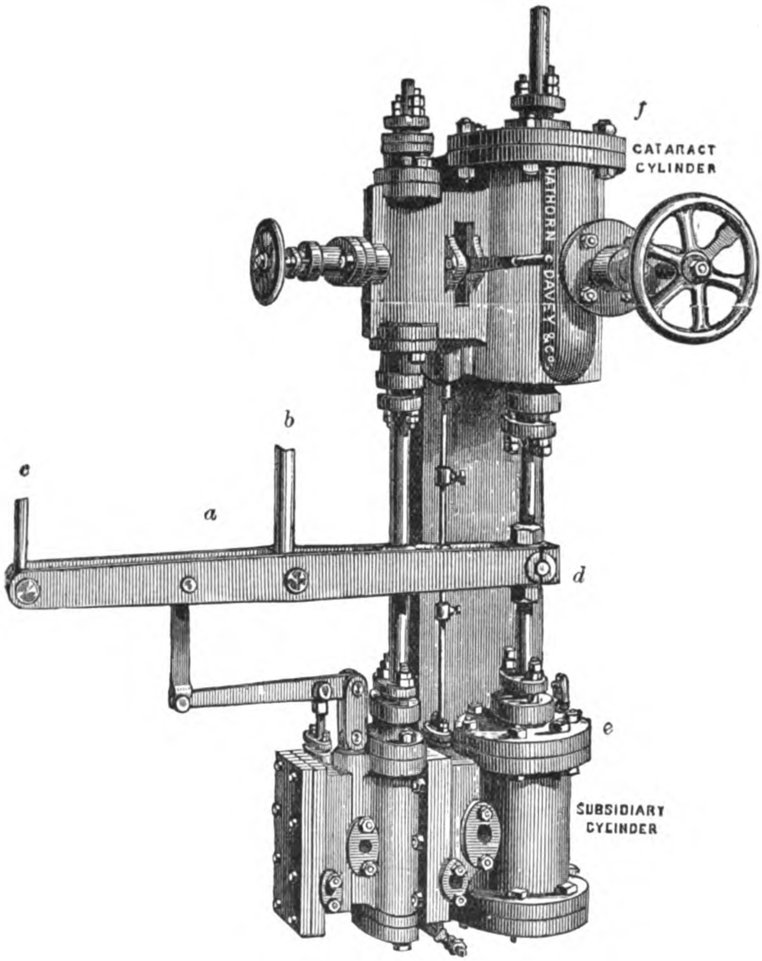
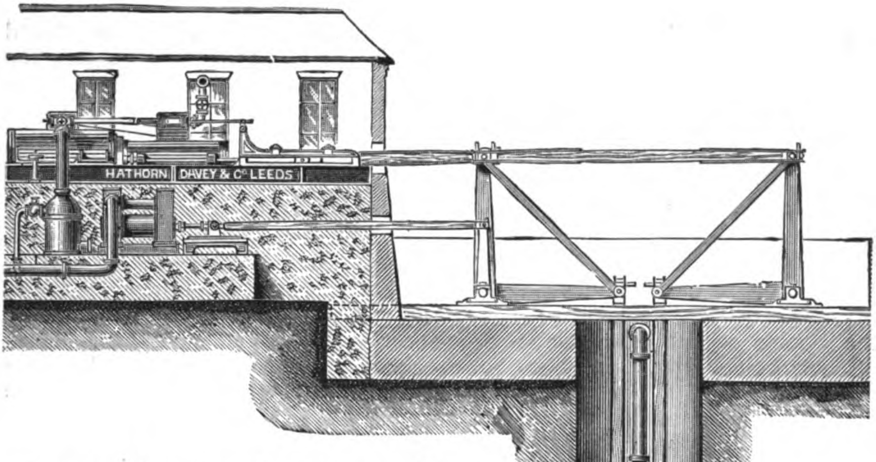


Fig. 357.—Davey's Differential Valve Gear as applied to work the Cornish Engine.

end (*d*) deriving its motion from a subsidiary cylinder (*e*), and being controlled by means of the cataract (*f*). The cylinder has a slide valve, which is worked by means of a tappet arm on the rod of the piston of a secondary cylinder; the motion of the secondary piston is also controlled by a secondary cataract. The slide valve is, however, free to move with the motion of a hand lever.

It will be seen that there are two hand wheels and a lever attached to the cataracts. The function of the large wheel is to regulate the speed of the engine during the stroke, the small wheel is for regulating the pause between the



Compound engines, having two cylinders, admit of higher degrees of expansion, without excessive speed of piston, than is possible with a single cylinder. The relative sizes of cylinders require to be carefully proportioned to suit the pressure at which it is proposed to work. This class of engine is becoming more and more in favour, and in marine engines, where economy in fuel is of such importance, three and even four cylinders have been employed, the results being highly satisfactory.

Davey's Compound Differential Pumping Engine, shown in Fig. 358, and manufactured by Messrs. Hathorn, Davey & Co., Leeds, owes its name to the differential arrangement of the valve gear. The engine is placed horizontally, is double acting, condensing, and worked expansively. The two cylinders are placed in a line. Their relative diameters depend on the initial pressure of the steam.

The back end of the high-pressure cylinder forms the front cover for the low-pressure cylinder. To meet this arrangement a single piston-rod works in the high-pressure cylinder and two in the low-pressure, one on each side of the piston, and these work through tubes on the outside of the high-pressure cylinder-jackets made for them. All the piston-rods work on to one cross-head, and motion is given to the pump rods or spears by means of two quadrants or L pieces, working two sets of pumps.

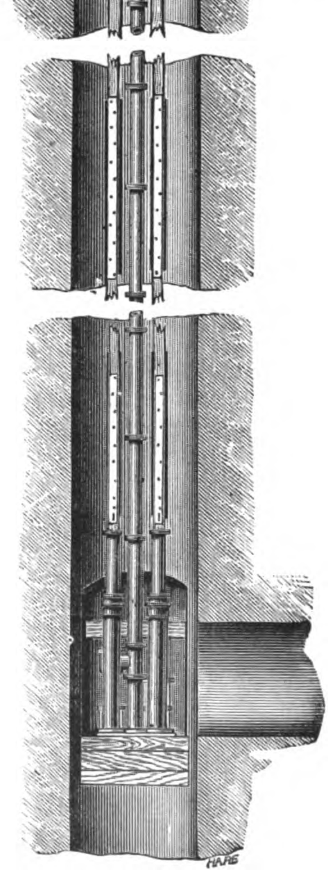


Fig. 358.—Davey's Compound Differential Pumping Engine.

The engine does not require expensive foundations. In one arrangement the low-pressure engine piston has a rod passing through its back cylinder cover

The engine has 33- and 60-inch cylinders with a 10-foot stroke, and it works two 20-inch plunger lifts, each pumping 400 feet high in one lift.

A larger engine of this description has been supplied by Messrs. Hathorn, Davey & Co., to the South Staffordshire Mines Drainage Commissioners. It has a 44-inch high and a 76-inch low pressure cylinder, the stroke being 10 feet, with air-pump and surface condenser, fitted with a large number of 1-inch gun-metal tubes. The condensed water is cleared from grease, and is then used to feed the boilers. The engines work two 19-inch plungers, with 10-foot stroke placed at a depth of 464 feet, forcing the water up an 18-inch column to the surface. The buckets and clacks are of gun-metal and of the Cornish type, with double beats. At each stroke of the plungers 245 gallons of water are raised, and the engine is capable of raising 2,000,000 gallons in twenty-four hours.

The advantages of pumping-engines being placed on the surface are:—a minimum of loss in the steam-supply, the boilers being close to the engines; better supervision and greater facilities for repairs; if the mine should be flooded, and the water rise above the pumps, the engine can work on uninterruptedly; and further, in cases where the heaviest feeders are met with and collected in the shaft, the lifts may be reduced in size from the top lift downwards. The water, perhaps, is pumped out of a shaft by means of a sinking set. The engine obtained for the purpose of working the pumps during sinking may be made to work the pumps permanently.

Pumping-engines are sometimes placed underground. A pit may be sunk comparatively dry, or the water may have been wound out by barrels or "kibbles" during the sinking, and afterwards large quantities of water may be made in the workings. On the other hand, a great advantage of the engine being placed underground is the absence of the cumbersome pump rods or spears, which are both costly in the first instance and in the wear and tear of working. Moreover, the spears limit the length of the column. Another advantage of placing the engine underground is derived from the fact that a much smaller engine will do the work there than on the surface, as the pumps may be always double-acting, and pump-rods being dispensed with, may be worked at a higher speed than engines on the surface. Lodge-rooms and cisterns in the shaft are also not required. In the usual form of pump worked by an engine on the surface the water is only in motion half the time, but with the engine placed underground it is continuously so, and smaller pipes will suffice, although they may require to be made of thicker metal in a long column. A disadvantage in placing the pumping-engine underground is that a disarrangement or accident to the pumps is more difficult to deal with, and also that the engine is liable to be drowned whilst standing. Steam may be conveyed from the surface down the shaft to the engine, or the boilers may be placed underground, but as stated when dealing with steam haulage, it is better to have the boilers on the surface. All large engines for underground pumping and their pumps, and most of the smaller engines or steam pumps, are placed horizontally, an arrangement which takes up little room and gives great compactness. The piston of the engine and the plunger of the pump are generally on one rod. The pumps underground should be double-acting, and it is well to have a pair of engines either of which is capable of working the pumps. The engines may have condensing arrangements or may exhaust into the upcast shaft. If there is no separate condenser, the exhaust steam can be led into the suction water pipe, which will not only get rid of the steam, but assist the engine by forming a partial vacuum. The Compound Differential engines give excellent results when placed underground. They have been made of all sizes of cylinder up to 60 inches in diameter and 8 feet stroke. The engine has frequently been placed, where circumstances have permitted, at a position in the pit above the workings, ing-engine is already working at its maximum power the hydraulic system cannot be applied.

The usual arrangement for hydraulic pumps is to have the power-piston and the ram on one rod, one behind the other. The valves are specially made and are generally worked by a tappet arrangement. The surplus pressure on the piston over the ram gives its motion; the area of the piston × lbs. pressure of head is in excess of the area of the ram × lbs. pressure of its head + friction. The speed of ram is from 18 to 30 feet per minute. Friction is an important point in hydraulic pumps. The loss of head in forcing water through long pipes is inversely as the fifth power of their diameters. For instance, with a 2-inch and a 4-inch pipe of the same length, in order to force the same quantity of water through them, the difference of loss of head would be as 1,024 : 32; or, say 165 gallons per minute had to be discharged at a distance of 400 yards, the 2-inch pipe would require 180 feet of head, while the 4-inch pipe would only require 5·625 feet of head to overcome their friction. In proportion to the relative costs of the pipes, the size of the supply pipe, and more especially the discharge-pipe should be large. An objection to hydraulic pumps arises from the fact, that whether the pump has much or little work to do it takes the same quantity of water to do it. The water after doing its work in the hydraulic pump is discharged into the delivery and returned along with that pumped to the sump.

A very ingenious method of raising water from mines is by means of Moore's hydraulic mine-pump. An ordinary horizontal steam-engine placed on the surface works a double-acting water-ram, also on the surface. A strong wrought-iron tube connects the two ends of the water-ram case, and also from either end of this case a power pipe is carried down the shaft to the bottom, or to any point in the workings where it is convenient to fix the hydraulic pump.

The hydraulic pump consists of a double-plunger pump, having connecting rods projecting through glands at each end, and these are made a suitable length to form the plungers of two hydraulic rams, placed at either end of the main pump as shown in Fig. 361. The two power pipes after being carried down the shaft are connected to the outer end of each hydraulic ram. The main pump is double acting, and has two suction valves and connections to a single pipe from the sump or cistern and also two outlet valves and connections to the rising main. Protection against shocks caused by the stoppage of the moving power column, or by irregularities between the motion of the engine rams and the underground hydraulic rams, is obtained by a connection between the two power pipes, and by providing a relief-valve, which, when opened, allows the water to pass from one column to the other. The hydraulic ram gives motion to a small bell crank at the end of each stroke, which works the relief-valve. The reciprocating action of the engine rams on the surface is transmitted direct to the underground hydraulic rams, the water forming a rod which receives motion from the engine rams at one end, and transmits it to the hydraulic rams at the other, there being no valves whatever between. The water in the power pipes should be quite clean when first placed there, and is not afterwards changed. Care must be taken, however, to maintain the column of water solid, so as to completely fill the ram cases and the power pipes; and to provide against leakages, small inlet valves and pipes, connected to a cistern, are placed at a higher level. If the pump is a small one, the steam-engine on the surface may be direct-acting; the piston rod passing out through the back cylinder cover, and there connected to the water-ram. With large pumps, a small engine running at a high speed may be geared to the pump, as the water in the power pipes must not have a high velocity. The engine should have a heavy fly-wheel on the crank shaft. The power columns are worked at a high pressure—about 1,000 lb. to the square inch. The area of the rams and power pipes should be so proportioned that the rams travel at a speed of about 80 ft. per minute, and the speed of the water in the power pipes

The application of electricity to underground pumping is practical and in some instances may be economical. At the Trafalgar Colliery, Forest of Dean, Mr. Frank Brain has erected pumping plant which is working satisfactorily and economically. The plant consists of a single 16-inch cylinder engine with a 12-inch stroke working with about 35 lbs. steam pressure on the surface. The power is applied to a generator placed on the surface near the shaft, by means of a belt carried on two pulleys, one being on the crank shaft of the engine, the other on the generator. The pump and electro-motor (the latter being a machine for converting the energy of electric currents into the energy of mechanical motion) are underground at a distance of 1,650 yards from the bottom of the shafts, and the water is raised through a vertical height of 300 feet to the pit bottom, along slant roads.

The pump is a double 9-inch plunger with a 10-inch stroke and fitted with gear running six to one. The spur pinion is keyed on the same shaft as a 64-inch pulley which receives motion from a 14-inch pulley on the motor shaft by means of a leather link belt. When the motor runs 650 revolutions per minute, the pump makes 25 revolutions.

The electric current is conveyed to the motor from the generator on the surface by means of a copper cable 2,000 yards long, wrapped with compounded tape, and taken down the shaft in wooden boxes. It is not afterwards enclosed but carried along the side of the underground roads, being supported on earthenware insulators at intervals of about 10 yards. An old pit rope is used as a return cable. It is about 4 inches in circumference and secured to the road posts by staples. A small insulated copper wire, connected to a battery of eight No. 3 Leclanche cells, connects the engine-house on the surface with the pump-house underground, and through this is registered, upon a bell placed for the purpose in the former, each stroke of the pump. The same wire serves as a telephone line, so that conversation can be carried on between anyone in the engine-house with the man in charge of the pump underground. The water pumped is about 114 gallons per minute.

Syphons may be beneficially applied for conveying water in mines under certain circumstances. The principle of the syphon depends upon the fact that the atmospheric pressure will sustain a column of water 34 feet high. In its simplest form it is merely a tube bent like the letter U inverted, and having one leg longer than the other. If the short leg be inserted in the water to be drawn off and the air extracted from the inside of the tube, the atmospheric pressure acting on the surface of the liquid forces it up the tube. The liquid will consequently flow through the longer leg until the level of the liquid falls below the short end of the tube, or, if the liquid be water, until the surface falls to 34 feet below the highest point of the tube.

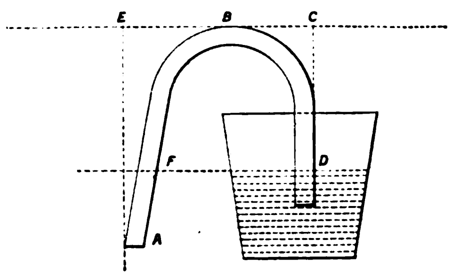


Fig. 362.—THE SYPHON.

Two circumstances limit the application of the syphon. Its highest point must be less than 34 feet above the surface of the water to be run off, and the delivery end of the pipe must be lower than the plane of the surface of the water to be removed. Suppose, in Fig. 362, A B D to be a syphon filled with water. The force acting on the vessel at D is the pressure of the atmosphere and the water is driver in the direction D B by that force, less the weight of a column of water whose into the syphon, which fills it in a few minutes, the air blowing off at valve, E. Whenever the air is all expelled from the pipes the water overflows at box, D; then let go the lever, L, when the weight, H, will take down levers *f* and *g*, which shuts the tap, F, and opens the large tap, G; the syphon will now be working, and will do so while the pipes get water at A.

The following useful memoranda relate to water and pumps :—

From Molesworth.

1 Cubic foot of water = 62·4 lbs. = 6·2355 gallons.
1 Cubic inch of water = ·036 lbs.
1 Imperial gallon of water = 10 lbs. = 0·16 cubic foot = 277·274 cubic inches.
1 Cwt. of water = 1·8 cubic foot = 11·2 gallons.
1 Ton of water = 35·9 cubic feet = 224 gallons.

Pressure of Water per square inch at Different Heads.

P = Pressure in lbs. per square inch.
H = Head of water in feet.
$P = H \times \cdot 4333.$
$H = P \times 2\cdot 31.$
Pressure per square foot = H × 62·4.
Cubic feet of water × ·557 = Cwt. approximately.
" × ·028 = Tons "
1 Cubic foot of sea water = 64·14 lbs.
Weight of sea water = weight of fresh water × 1·028.

Delivery of Water in Pipes.

D = Diameter of pipe in inches.
H = Head of water in feet.
L = Length of pipe in feet.
W = Cubic feet of water discharged per minute.

$$W = 4\cdot 72 \sqrt{\frac{D^5 H}{L}}$$

$$D = \cdot 538 \sqrt[5]{\frac{L \times W^2}{H}}$$

Hawkesley's formula for the delivery of water in pipes is,
G = Number of gallons delivered per hour.
L = Length of pipe in yards.
H = Head of water in feet.
D = Diameter of pipe in inches.

$$D = \frac{1}{15} \sqrt[5]{\frac{G^2 L}{H}}$$

$$G = \sqrt{\frac{(15\ D)^5 H}{L}}$$

Molesworth's Rule for the Weight of Pipes.

D = Outside diameter of pipe in inches.
d = Inside diameter.

Question 69.—What size of pumping-engine would you erect underground to raise 30,000 gallons of water per hour from a depth of 300 yards?

Assuming that the piston and pump are double-acting, and that the piston will work at 250 feet per minute, which is a safe speed, and have an effective steam-pressure of 30 lbs. per square inch, $\frac{30{,}000}{60} = 500$ gallons per minute, and as the pump works at 250 feet per minute, $\frac{500}{250} = 2$ gallons for each foot the pump works. $2 \times 277{\cdot}274 = 554{\cdot}548$ cubic inches in each foot of the pump. $\frac{554{\cdot}548}{12} = 46{\cdot}212$ square inches area, allow say $12{\cdot}788$ square inches for the pump-rod of about 4 inches diameter $= 59$ and $\sqrt{\frac{59}{{\cdot}7854}} = 8{\cdot}667$; say, to allow for leakages 10 inches as the diameter of pumps necessary. The pressure per square inch of water on a 300-yard column will be $300 \times 3 \times {\cdot}4333 = 390$ lbs. and $46 \times 390 = 17{,}940$ lbs. total pressure on the pump. $17{,}940 \div 30$ lbs. the steam pressure $= 598$ square inches, and adding $\frac{1}{2}$ for frictional allowances, $598 + 299 = 897$ square inches area, and $\sqrt{\frac{897}{{\cdot}7854}} = 33{\cdot}8$ inches diameter of the cylinder required for the pumping-engine; the stroke might be made 6 feet, and it would then work $\frac{250}{6 \times 2} = 20\frac{5}{6}$ revolutions per minute. The stroke of the pump should always be made as long as practical considerations will admit of.

Question 70.—What quantity of water could you raise and from what depth with an Underground Pumping-Engine, having a 30-inch cylinder and 5-foot stroke, double acting, working a 12-inch pump also double acting, the effective steam pressure on the piston being 30 lbs.?

Assume a piston speed of 250 feet per minute, or $\frac{250}{5 \times 2} = 25$ revolutions per minute. The pump-rod will be the same size as the piston-rod, and the piston-rod would be about $\frac{1}{8}$th the diameter of the piston, or $\frac{30}{8} =$ say 4 inches or $12\frac{1}{2}$ square inches area. Deduct this from the area of the pump, thus $(12^2 \times {\cdot}7854) - 12\frac{1}{2} = 100{\cdot}5$ as the effective area. The 30-inch cylinder has an area of $706{\cdot}86$, from which deduct the third for frictional allowances $= 706{\cdot}86 - 235{\cdot}62 = 471{\cdot}24$, say 471 square inches area on which the steam pressure operates; $471 \times 30 = 14{,}130$ available effective power. The effective area of pump being $100{\cdot}5$, $\frac{14{,}130}{100{\cdot}5} = 140{\cdot}5$ lbs. pressure which the pump can support and $\frac{140{\cdot}5}{{\cdot}4333} = 324$ feet height of column. And to find the quantity $100{\cdot}5 \times 12 = 1{,}206$ cubic inches in each foot $\frac{1{,}206}{277{\cdot}274} = 4{\cdot}4$ gallons per foot. $4{\cdot}4 \times 250 = 1{,}100$ gallons per minute. $1{,}100 \times 60 = 66{,}000$ gallons per hour, the quantity of water this engine would raise to a height of 324 feet.

In working out these two examples the method given by Mr. Percy, in his *Mechanical Engineering of Collieries*, has been adopted.

Question 71.—The feeders of water at a colliery are lifted at the rate of 3,000 tons in 12 hours by an engine going 12 strokes per minute, length of stroke 5 feet 6 inches. What diameter of pump will be required, and what is the feeder per minute in gallons?

To find the feeder per minute, which is dealing with the last part of the question first, $3{,}000 \times 224$ (the number of gallons in a ton) $= 672{,}000$ the number of gallons lifted in 12 hours, therefore $\frac{672{,}000}{12 \times 60} = 933{\cdot}\dot{3}$ gallons per minute, which is the feeder. Assuming the pumps to be single-acting and adopting Molesworth's rule as already given, $D = \sqrt{\frac{933{\cdot}\dot{3}}{{\cdot}034 \times 5{\cdot}5 \times 12}} = 20{\cdot}4$; or the same result may be arrived at as follows, $D = \sqrt{\frac{933{\cdot}\dot{3} \times 277{\cdot}274}{5{\cdot}5 \times 12 \times 12 \times {\cdot}7854}} = 20{\cdot}4$ as before ($277{\cdot}274$ being the number of cubic inches in a gallon). This gives the net diameter of the pump plunger, and it is usual to increase the area of the plunger $\frac{1}{4}$th to allow for leakage, &c. Therefore $20{\cdot}4^2 \times {\cdot}7854 = 326{\cdot}74$ area of plunger and $326{\cdot}74 + \frac{326{\cdot}74}{4} = 408{\cdot}4$ which is the area the pumps should be, and $\sqrt{\frac{408{\cdot}4}{{\cdot}7854}} = 22{\cdot}804$ or say 23 inches, which is the diameter the plunger should be to pump 3,000 tons of water in 12 hours by an engine going 12 strokes per minute with a 66-inch stroke.

Question 72.—Describe the general arrangements, diameter of cylinder, length of stroke of pump, and pressure of steam by which 2,000 tons of water may be raised from a depth of 200 fathoms in 24 hours.

$\frac{2{,}000 \times 224}{24 \times 60} = 311$ gallons to be pumped per minute, but this would allow no margin for the engine to rest through accident, and as the depth is great, entailing many working parts, it would not be prudent to assume that the engine can go on continuously, and the colliery may have no storage arrangements. Assume that the engine works 16 hours out of the 24, then $\frac{2{,}000 \times 224}{16 \times 60} = 467$ gallons to be pumped per minute. It would be better to choose an engine placed on the surface to deal with this water, because it would be necessary to use large pumps of 18 or 19 inches diameter. If an engine were placed underground and one column of pumps used, the thickness of metal in the bottom part of the column would be excessive. For 19-inch pumps in a 200-fathom pit, it must be ${\cdot}000144 \times 200 \times 6 \times 19 = 3{\cdot}283$ inches, a thickness of metal for pumps never adopted and not practicable, and even for a 12-inch set it would require to be over 2 inches, a size not often exceeded for pumps.

The most approved engine to do the work would be Davey's Compound Differential Pumping Engine, but, assuming that a Cornish single-acting engine with beam overhead be adopted and that it works with a 10-foot stroke, it may be driven comfortably and steadily at 5 strokes a minute. To find the size of pumps then $\sqrt{\frac{467 \times 277{\cdot}274}{10 \times 12 \times 5 \times {\cdot}7854}} = 16{\cdot}57$ diameter, or allowing $\frac{1}{4}$th increase of area for leakage, it means $18{\cdot}57$, or say 19 inches as the diameter of the working barrel and plungers. In arranging the sets in the shaft divide them into 5 sets of 40 fathoms each; and, for reasons already given, the bottom one should be a lifting set, the others being all forcing sets.

To find the horse-power of the engine, assuming that there is an effective steam pressure on the piston of 30lbs. $\frac{467 \times 10 \times 200 \times 6}{33{,}000} = 169{\cdot}82$ or say 170 horse-power, but to which must be added 80 per cent., 20 per cent. to overcome friction and 60 per cent. more for contingencies $170 + \frac{170 \times 80}{100} = 306$ as the horse-power of the size of the spears it would be found impracticable to adopt that plan. Great wear would result to a bucket of one long lift, requiring constant changing and repairing; broken spears, especially near the bottom of the lift, would be a source of great annoyance and expense, and the arrangement would not give facility for counterbalancing the engine.

Question 76.—A pumping-engine goes 14 strokes per minute for 13 hours a day, working two 18-inch sets with a 49-inch stroke. What is the feeder per minute?

$\frac{18^2 \times \cdot 7854 \times 49}{277 \cdot 274} = 45$ gallons per stroke for each set, 90 gallons per stroke for the two, and $90 \times 14 = 1{,}260$ gallons as the feeder pumped per minute; but from the question it would appear that this quantity pumped per minute for 13 hours represents what is made in the pit during the 24 hours, and if so the feeder in the pit would be $\frac{1{,}260 \times 60 \times 13}{24 \times 60} = 683$ gallons per minute.

Question 77.—Describe the pulsometer.

The following description of the pulsometer is taken partly from the *Colliery Guardian* of Sept. 29th, 1876, and partly supplied by the Pulsometer Engineering Co., Nine Elms Ironworks, London.

This pump may be said to raise water by the direct pressure of steam upon its surface, and then to turn this same steam to account in forming the vacuum necessary for effecting the suction for the next lift. This is accomplished without the intervention of any piston or plunger. The invention is, in fact, but a perfection of the principle of Thomas Savery in 1698, and by him carried out so clumsily that the idea seems to have been abandoned until now; and engineers have been content in the meantime to interpose between the power and its work a complicated set of working parts, adding greatly to the first cost of the pump, introducing a heavy item for repairs and maintenance, and diminishing the duty on account of friction.

The pulsometer as now constructed, consists of a hollow casting, called the body, composed of two pear-shaped water-chambers, with their necks joining above, between which is the air-vessel, while the discharge chamber is a kind of offshoot from the lower portion of the body. The steam admission-valve is bolted on at the top, at the junction of the ends of the water-chambers; and the suction pipe is similarly bolted on at the bottom. The construction will, however, be better understood on reference to the drawing, Fig. 365, which is a vertical section through the pulsometer. A A are the two water-chambers surmounted by a separate casting containing the steam ball-valve I, fitted so as to pass alternately between the seats formed in the junction, and connected at top with the steam pipe K. The water-chambers terminate at bottom in a suction-chamber, to which the suction pipe is bolted, while the suction valves E E, with their seats, are arranged at an angle of 45 degrees between them. Covers for closing the spaces are left in the casting for the insertion of the valves, and also for the removal of any obstructions. The air-vessel for rendering the flow constant is situate between the necks of the water chambers, and is in direct communication with the suction.

A discharge chamber common to the two water-chambers A A, and leading to the discharge pipe D, is provided, and this contains one or two valves, according to the purpose to be fulfilled by the pump, which are shown in dotted lines in the figure. Small air-cocks are screwed into the cylinders and air-chamber, for use as described hereafter.

The pump being filled with water, either by pouring water through the plug hole in the chamber or by drawing the charge, according to the printed directions, is ready for work. Steam being admitted by slightly opening the stop-valve, which allows it to enter the steam-pipe K, it passes down that side of the steam neck which is left open to it by the position of the steam ball, and presses upon the small surface of water in the chamber which is exposed to it, depressing it without any agitation, and consequently, with but very slight condensation, and driving it through the discharge opening and valve into the rising-main.

The moment that the level of the water is as low as the horizontal orifice which leads to the discharge, the steam blows through with a certain amount of violence, and being brought into intimate contact with the water in the pipes leading to the discharge chamber, an instantaneous condensation takes place, and a vacuum is in consequence so rapidly formed in the just emptied chamber, that the steam ball is pulled over into the seat opposite to that which it had occupied during the emptying of the chamber, closing its upper orifice and preventing the further admission of steam, thus allowing the vacuum to be completed; water rushes in immediately through the suction pipe, lifting the inlet valve E, and rapidly filling the chamber A again. Matters are now in exactly the same state in the second chamber as they were at the commencement in the first chamber, and the same results ensue. The change is so rapid that, even without an air vessel on the delivery, but little pause is visible in the flow of water, and the stream is, under favourable circumstances, very nearly continuous. The air-cocks are introduced to prevent the too rapid filling of the chambers on low lifts, and for other purposes, and a very little practice will enable any unskilled workman or boy so to set them by the small nut that the best results may be obtained. The action of the steam ball is certain, and no matter how long the pump may have been standing, it will start as soon as dry steam is admitted.

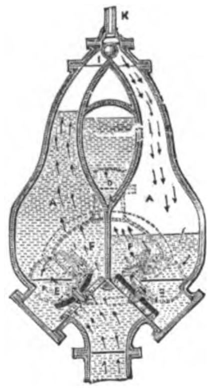


Fig. 365.—THE PULSOMETER.

The smallness of the quantity of steam used in raising the water and effecting the condensation, is proved by the very slight increase of temperature acquired by the water raised.

It will be noticed that great simplicity of form and fewness of working parts are attained in the pulsometer; there are, in fact, only 5 moving parts, the steam-admission valve, the two suction or foot valves, and the two delivery valves. These, again, are made as simple and efficient as possible. The steam-ball valve, when once made spherical, wears itself and its seats true, as it turns in its bed at every pulsation. The foot and delivery valves may be of the same character, though larger, with suitable guards, but for heavy work the flap-valve is found preferable. It consists of a cast-metal shoe, planed on its lower surface, and having a bored recess into which is fitted a seat made of hard wood boiled in oil, the end of the grain being the part exposed to wear. This surface is turned true, and the wood is fastened in the shoe by a bevelled brass ring secured with pins. The flap works in a bored bearing, and is kept in its place by a guard cast on the cover of the chamber, or by a small cover. Its play is regulated by a stop on the chamber cover, and noise is prevented by the wood lining of the horn of the valve. In some instances the flap valves are furnished with wooden faces. India-rubber is also used for some purposes, either in the form of a flat disc or the hollow valve designed by Perreaux.

In the majority of pumps, grid valves are employed, as shown in Fig. 365.

times as heavy as hydrogen: for nitrogen this number is 14. These numbers give the comparative weights of oxygen and nitrogen.

The specific gravity of a compound gas like carbon dioxide, referred to hydrogen as unit, is calculated by simply halving the total weight of the constituents of its formula. Thus the weight represented by CO_2 is $12 + 32$ or 44; and this gas is therefore $\frac{44}{2}$ or 22 times as heavy as hydrogen.

Hence the relative weights of hydrogen, oxygen, nitrogen and carbon dioxide are 1 : 16 : 14 : 22. Since air has a density of 14·5 on this scale, it is only necessary to divide the above numbers by 14·5 in order to obtain the density of these gases in relation to air as unit.

It will thus be seen that the use of a correct chemical name furnishes a knowledge of the composition of substances, while chemical formulæ, supplemented by a reference to a table of atomic weights, enable the composition by weight of substances to be calculated; and in the case of a gas they enable the specific gravity to be calculated with the greatest ease.

The general character and composition of a few important gases is described below.

ATMOSPHERIC AIR.—Density = 1, or referred to hydrogen = 14·43.

Air consists principally of the elementary gases nitrogen and oxygen, but it always contains in addition some carbonic acid gas. Water vapour is invariably present in air, but its proportion varies widely according to circumstances.

Leaving the proportion of moisture and carbonic acid out of consideration, the average composition of fresh air by volume is represented by the following percentage statement:—

Composition of dry, pure, fresh Air.

By measure.		*By weight.*	
Nitrogen	79·1	Nitrogen	76 23
Oxygen	20·9	Oxygen	23·77
	100·0		100·00

Taking a rough average for the proportion of moisture, and entering the usual and fairly constant proportion of carbonic acid in fresh air, the percentage composition will stand thus:—

Composition of fresh Air.

By measure.	
Nitrogen	77·95
Oxygen	20·61
Moisture	1·40
Carbonic Acid	0·04
	100·00

In this statement of composition all occasional constituents are omitted. The minor constituent, discovered recently and named Argon, is also not inserted, since its proportion certainly does not exceed 1 per cent. of the atmosphere; and its small proportion and its wholly negative and indifferent character cause it to be probably of little importance in connection with the changes hereafter referred to.

The uniformity in composition of fresh air in open places is strikingly constant. The only constituent which varies in quantity to any appreciable and important

Oxygen is now prepared in large quantities from atmospheric air by the Brin process. It can be purchased compressed in strong steel cylinders; and it is therefore available at a moderate cost.

NITROGEN.—Chemical symbol, N. Atomic weight and density = 14. Specific gravity = 0·965 (if air = 1).

This elementary gas is present as the major constituent of air, which it nearly approaches in density. Nitrogen is unable to support combustion, and cannot maintain life when it is breathed. It is not, however, a poisonous gas, and if its proportion in air becomes unduly large, its presence is injurious mainly by excluding some of the oxygen.

Nitrogen is present as a constituent in many liquid and solid explosives, and the expansion which causes the explosion is in such cases due to the separation from the explosive of a considerable volume of nitrogen gas.

HYDROGEN.—Chemical symbol, H. Atomic weight and density = 1. Specific gravity = 0·069 (air = 1).

This elementary gas is the lightest gas known, hence if the densities of other gases are referred to that of hydrogen as unit, these densities are necessarily larger than 1.

Hydrogen occurs in nature only as a constituent of certain compound substances, of which water is the most common and abundant.

When this gas is kindled, it burns vigorously in the air with a very pale reddish-purple flame. This flame is intensely hot, and is very difficult to extinguish by being blown upon. The flame maintains itself in air containing large proportions of extinctive gases, owing partly to the small proportion of oxygen necessary for its combustion, and partly to the high temperature of the flame. The sole product of the combustion of hydrogen is steam.

The high temperature of the hydrogen flame burning in air is further augmented if the flame burns in air enriched in oxygen, and the temperature becomes extremely high when the flame burns in pure oxygen. The oxy-hydrogen blowpipe is an apparatus arranged to furnish this flame of hydrogen fed with oxygen; the flame of this blowpipe was originally used chiefly for heating a small cylinder of quicklime to produce the lime-light, but is now employed in many important metallurgical processes. Hydrogen explodes violently if it is mixed with half its measure of oxygen and the mixture is fired.

Hydrogen can now be cheaply bought compressed in strong steel cylinders, and is therefore available for useful applications, one of which will be described below.

METHANE, OR MARSH GAS.—Chemical symbol, CH_4. Density = 8. Specific gravity = 0·55 (air = 1).

This compound gas was formerly known as "light carburetted hydrogen," a name intended to denote its lightness and the fact that it is a compound of carbon and hydrogen. Its composition is now denoted by terming it a *hydrocarbon*, a name which is applied to all substances which contain hydrogen and carbon as their sole constituents. The name "marsh gas" was applied to the gas when it was obtained by stirring the mud of stagnant pools and marshes.

Methane is the inflammable constituent of *fire-damp*. Some samples of fire-damp consist almost entirely of this gas, whilst others contain it in mixture with carbonic acid and with nitrogen.

The name "gas" is frequently applied to methane in the coal-mine: it is sometimes wrongly termed hydrogen by the miner, the name being probably a curtailment of the old chemical name for this gas.

Methane is readily kindled; it burns with a somewhat luminous flame in the

10 parts, after-damp contains 7 of nitrogen, 1 of carbonic acid gas, and 2 of steam. Directly after the explosion the steam condenses, and there is then left out of $8\frac{1}{2}$ parts, about $7\frac{1}{2}$ of nitrogen and 1 of carbonic acid gas. Breathing after-damp soon causes death, and many who escape the force of a fire-damp explosion in mines fall victims to the deadly after-damp.

If a mixture is exploded which contains more than 1 of fire-damp in 8 or 9 of air, its force is less than that of the most destructive proportions of fire-damp and air, as a certain amount of carbonic oxide is formed, which, if it could have burnt, would have increased the temperature and force of the explosion. Its presence after the explosion renders the after-damp more deadly. Again, if an explosion occurs in which the fire-damp forms less than 1 in 8 or 9 of air, its force must be less than that of the most destructive proportions of fire-damp and air, but in this case the after-damp will be rather less deadly than that resulting from the most violently-explosive mixture, as a part of the oxygen in the air remains unchanged. Whatever the proportion of fire-damp to air, the after-damp left from its explosion is unfit to breathe.

When ignited, the flame temperature of fire-damp and air is extremely high, and where there is a large volume of the explosive mixture present, the temperature of the portion being consumed increases the volume of the rest. From the centre or seat of explosion a great pressure is caused by the flames and heated gases, which proceed in every direction, driving the air away with great force; this pressure is exhausted at a distance near to or far from the seat of the explosion, according to its violence. As it progresses the condensation of steam in the after-damp and the cooling of the gases reduces its volume and pressure until it descends sufficiently to cause a backward movement, and a partial retreat of the ignited mass towards the seat of the explosion frequently ensues.

The ravages committed by these destructive blasts are familiar enough to explorers after explosions. As the conflagrations rush along the roadways, they for the most part take a course opposite to that of the intake air, with occasional "kick-backs," or slight splits at the junctions of roads, and as the flames are fed by the fresh air, the blast, if supplied by sufficient fire-damp, or fire-damp and coal-dust, it may be reaches the bottom of the downcast shaft, and exhausts itself in the shaft, disarranging or perhaps blowing out nearly all the shaft fittings with so much violence and noise as to cause the utmost terror to those employed on the pit top. In its destructive progress everything presenting an impediment, unless strong enough to resist the blast, is hurled to one side or overthrown; doors, air-crossings, trams, horses, men, the timbers for securing the roadways, &c., usually offer no obstacle to the fury of the explosion. The road timbers being knocked down the roof falls in, and the ventilation is arrested. In other parts of the roadways the timbers are considerably charred and deflected from an upright position, their altered state and appearance pointing almost as certainly as a finger-post in the direction of the blast. Too often the evidence of such mute objects is all that is to be obtained, for those who escape the violence of the explosion are poisoned by the after-damp, and not one is spared to throw light on the calamity.

Wet roadways naturally prevent accumulations of dust and the progress of an explosion, for it is in dry and dusty mines that the worst occur. The violence of the blast in these is arrested through any portions of the roads which are wet, however the dampness is caused.

Carbonic Acid, or Carbon Dioxide.—Chemical symbol, CO_2. Density $= 22$. Specific gravity $= 1 \cdot 5$ (air $= 1$).

This compound gas is the product of the combustion of the elementary substance carbon, and it is formed when any substance containing carbon is burnt with a sufficient supply of air. The gas is also formed during respiration. It escapes in some quantity from certain kinds of coal, and is also present in the soil.

The gas cannot be inhaled if it is unmixed with air, or if it only contains a small amount of air; hence it is not inappropriately termed "choke damp," since the attempt to inhale produces a sensation of choking. Carbonic acid is an important product of an explosion of fire-damp, and is present invariably in large proportion in the "after-damp" produced when "gas" is fired or exploded in the coal-mine.

Carbonic acid gas issuing from the coal or soil tends by its weight to fall in the air and to collect on the "floor" of the workings, or in the "sump" or well, but the process of diffusion causes it to slowly mingle with the air. When carbonic acid issues from the lungs, or is produced by a flame, however, it is rendered light by its high temperature, and tends to rise in the air.

Atmospheric air, as has been already stated, contains usually about 0·04 per cent. of carbon dioxide. This proportion has no effect upon the respirability of the air; but if the proportion of the gas increases it will at last reach an amount which renders the air unfit to support life. The extreme limit has been stated to be about 15 per cent., but even less proportions than this in the air cause drowsiness when the air is breathed, and should not be inhaled for any length of time.

Pure carbon dioxide gas at once extinguishes flame, and the gas retains this extinctive power even when it is mixed with a considerable proportion of air. Air containing 15 per cent. of carbonic acid extinguishes at once the flame of candle, oil, and alcohol; coal-gas flames require about 33 per cent. of carbonic acid for their extinction, and a hydrogen flame is not extinguished until the proportion reaches 58 per cent. Hence the hydrogen flame is able to maintain itself in air containing four times the quantity of carbonic acid which proves extinctive to a candle, oil, or spirit flame, and nearly twice the amount which extinguishes a coal-gas flame.

Carbonic Oxide, or Carbon Monoxide.—Chemical symbol, CO. Density, 14. Specific gravity = ·965 (air = 1).

This gas is composed of the same constituents as carbon dioxide, but these constituents are present in different proportions. It does not occur in coal, and is only produced by artificial processes. It is formed when any substance containing carbon is incompletely burnt; also when carbon dioxide gas is acted upon by red-hot carbon.

It is therefore generally considered that one of the conditions essential to the formation of this gas in the coal-mine, is the burning or explosion of fire-damp with a quantity of oxygen insufficient to burn it completely into carbon dioxide. Another cause of the formation of the gas is the presence of coal-dust, which becomes heated by the fire-damp flame and changes carbon dioxide into carbonic oxide. The smothered combustion of coal in the mine or of a gob-fire also furnishes the gas.

This gas is also produced in the coal mine by the firing of blasting-powder, and of nitro-cotton. It is an important constituent of "water-gas."

Carbonic oxide is combustible in air; it burns with a blue flame, and becomes explosive when it is mixed with air and fired. But the chief danger arises from its powerful, poisonous action when it is inhaled, even in small proportions, probably less than 1 per cent.; it is rapidly fatal when breathed. Those who have been poisoned by the gas retain the appearance of life and health, and the colour of the skin is not changed.

The gas has a faint sickly smell, and when inhaled produces a tight feeling in the brain and giddiness. It shows a distinct "cap" over the hydrogen flame when less than 0·25 per cent. is present in the air, and may therefore be detected in proportions which are not at once fatal. The "cap" test, however, can only risk from the employment of blasting-powder may be reduced by employing other explosives, which give no carbonic oxide or inflammable gas and which do not produce so lengthy and durable a flame.

EXPLOSIVES.

The explosive substances used for blasting and bringing down the coal are solid or semi-solid substances, which owe their power to being able to generate instantaneously a considerable volume of gas, and to produce at the same time a high temperature. The production of the gas alone would lead to great increase in volume of the explosive, or to production of great pressure if the explosion occurred in a closed space; but the high temperature imparted to this gas at the moment of its production further adds very considerably to its volume and to the pressure which it exerts if confined. The gases usually evolved by explosives are carbon dioxide, nitrogen, and steam.

Explosives are caused to explode by two distinct methods:

(1) *By firing with heat*, so as to cause the properly prepared mixture to catch fire and burn rapidly; of such explosives, gunpowder and blasting-powder are examples. These are usually fired by a combustible fuse.

(2) *By shock or detonation*, the explosive being caused to undergo its explosive change by being subjected to the shock of a suitable substance being exploded or detonated in contact with it: dynamite and blasting-gelatine are instances of this class. They are exploded by so-called detonators, which are themselves caused to explode by electricity.

The applicability of an explosive for use in the coal-mine must be judged from several standards, of which the following are the most important:

1. Its safety from being exploded whilst undergoing transport and handling.
2. Its incompetence to fire "gas" or coal-dust in the air of the coal-mine.
3. Its non-production, when exploded, of any amount of gas which is poisonous or inflammable, such as carbonic oxide.
4. The non-emission of burning particles which could fire gas or coal-dust.

Some of the chief explosives which are in use or have been proposed for use in the coal mine are described in the following pages. They are classified according to their method of being exploded. Certain modern explosives are omitted or shortly treated of on account of their unfitness for mining purposes, because they do not comply with all the above conditions. It may be at once stated that the only explosives which satisfy the third condition are nitro-glycerine, which is used in the form of dynamite, and the so-called Sprengel explosives.

I. *First Class of Explosives.*

Mining powder, Blasting-powder and Bobbin powder.—These explosives have until recently been most largely used in the coal-mine. They are fired by a fuse which in itself is a source of danger. These explosives are practically gunpowder, modified in composition so as to lessen the cost and to increase the volume of gas produced when it is fired, while the temperature produced by the firing is lowered. The average percentage composition of these powders in England is as follows:

Potassium nitrate	65
Sulphur	20
Charcoal	15

Gelignite	Nitro-glycerine	56·5
	Nitro-cotton	3·5
	Wood meal	8·0
	Sodium nitrate	32·0

Since nitro-glycerine is a liquid substance, none of the above mixtures are hard solids. They present special danger owing to liquid nitro-glycerine being apt to exude from them. They are further sensitive to shock, more especially when they are frozen. In the frozen condition they therefore present danger to which the ordinary explosive is not subject. The necessary thawing of these substances is also attended with serious danger.

Nitro-glycerine is a powerful explosive, and presents the advantage over many other explosives, that it contains more than enough oxygen to completely oxidise its carbon, hence no carbon monoxide is produced when it is exploded. It is converted for use into a semi-solid form by mixing it with solid powders. The powder used in making dynamite does not add to the explosive force, but in each of the other mixtures the powder which is added is itself explosive.

2. Another kind of explosive falling under this second class is that composed of nitro-cotton powders, such as Tonite, which is a mixture of nitro-cotton and barium nitrate.

Nitro-cotton itself, when detonated, furnishes carbon monoxide, owing to the oxygen which it contains being too small in amount to convert its carbon into carbon dioxide.

III. *Third Class of Explosives.*

This class of explosives can only be exploded by a special detonator composed of fulminate of mercury or of nitro-glycerine. These explosives accordingly obviate risks from explosion by heat or by spark, or by any ordinary shock. They include the following:—

Roburite	Ammonium nitrate	86
	Chlorinated dinitrobenzol	14
Bellite	Ammonium nitrate	80
	Dinitrobenzol	20
Ammonite	Ammonium nitrate	88
	Nitro-naphthalene	12
Securite	Ammonium nitrate	80
	Dinitrobenzol	17
	Ammonium oxalate	3

These explosives produce a lower temperature when they are detonated than other explosives do; and, although the temperature is high enough to kindle fire-damp, the kindling does not occur on account of the short duration of the flame. It is found that fire-damp must be exposed to flame for several seconds in order that it may be kindled with certainty. This is not the case with carbon monoxide. These explosives produce no carbon monoxide or other inflammable or poisonous gas, and no smoke when they are detonated; and since they contain no substance which remains solid after detonation, there is no risk of red-hot solid particles being projected into fire-damp or fine coal-dust and causing fire or explosion. Further, since they are exploded by a detonator, there is no danger of a burning fuse being projected and causing damage for a similar reason.

The ammonium nitrate (NH_4NO_3), employed in the preparation of these explosives, presents several advantages over the potassium nitrate (KNO_3) and the potassium chlorate ($KClO_3$) used in the preparation of certain other mining powders. It is completely converted into nitrogen gas and steam during detonation, and at the same time furnishes some oxygen for the combustion of the other constituents of the explosive. Further, it absorbs a considerable amount of heat during its detonation and thus tends to lower the temperature when the explosive is in use. This absorption of heat makes it impossible to kindle these explosives and cause them to burn in a large mass, and renders necessary a very powerful detonation to explode them.

That these explosives require a high temperature for their kindling seems to be established by the fact that when a mixture of gunpowder and roburite is ignited the pieces of roburite are scattered without change.

A similar set of safe high explosives are in use in France in which ammonium nitrate is mixed with dynamite, with blasting gelatine, with nitro-cotton, and with trinitronaphthalene.

In connection with these comparative statements concerning explosives, the following tabulated information will be of interest.

PERCENTAGE OF INFLAMMABLE GAS PRODUCED BY VARIOUS EXPLOSIVES.

Explosive.		Carbon Monoxide.	Hydrogen and Methane.	Total percentage of Combustible Gas.
Powder . .	Gunpowder	10·5	3·1	13·6
	Blasting-powder . . .	33·7	7·9	41·6
Sprengel explosives .	Roburite	Nil.	Nil.	Nil.
	Ammonite	Nil.	Nil.	Nil.
Nitro-glycerine explosives .	Nitro-glycerine	Nil.	Nil.	Nil.
	Gelignite	7	Nil.	7
	Carbonite	15	26	41
	Blasting gelatine . . .	32·5	8·6	41·1
Nitro-cotton explosives . .	Tonite	8	Nil.	8

TEMPERATURE PRODUCED BY FIRING VARIOUS EXPLOSIVES.

	Degrees Centigrade.
Blasting-gelatine	3,090
Nitro-glycerine	3,200
Dynamite (20 p.c. silica)	2,940
Gun-cotton (11-nitro)	2,650
Tonite	2,648
Picric acid	2,560
Roburite	2,100
*Ammonium nitrate	1,130

* Ammonium nitrate cannot be completely detonated by itself, but it is completely detonated in the explosive mixtures which contain it.

THE VENTILATION OF MINES.

Air is an *elastic* body. If it or any gas (as they have the same physical properties) be confined in a vessel, it exerts a pressure against the sides altogether apart from its weight, and the pressure is exerted on the upper part of the vessel as well as the lower. If the volume occupied by it be in any way diminished. that is, if the same quantity is made to occupy a smaller space, the pressure will be increased, or if the space occupied remain the same, and the temperature be raised, the pressure will also be increased.

Air is *compressible*. Boyle and Mariotte's law of compression is:—The temperature remaining the same, the volume of a given quantity of gas varies inversely as the pressure which it bears. The law showing the relation between the temperature and the volume of any gas was discovered by Charles, and may be stated as follows:—If any gas be allowed to expand freely under a constant pressure, its increase of volume when raised from 32° F. to 212° F. will be equal to 0·366 of its original volume, and this law of increase holds true in the same proportion for intermediate temperatures. From this it follows that as the difference between 212 and 32 is 180, then $\frac{1}{180}$th of ·366, or about $\frac{1}{492}$, (strictly $\frac{1}{491 \cdot 8}$ or $\cdot 0020\dot{3}$) is the expansion for each degree, and this fraction is taken as the co-efficient of expansion. In other words, a gas expands $\frac{1}{492}$ of its volume at 32°, or $\frac{1}{460}$ of its volume at 0° for each degree that it is raised above that point. Supposing such a question as the following be given:—A quantity of gas is measured at a temperature of 70° and is found to occupy 400 cubic inches, what is its volume at 60°? To find the proportion between the space a gas occupies at 60° and 70° of Fahrenheit's thermometer, 492 cubic inches at 32° occupy 520 at 60° and 530 at 70°. The volumes therefore for any other quantity must be in this proportion, and therefore 400 cubic inches at 70° will occupy a volume at 60° in proportion as 530 is to 520. As 530 : 520 : : 400 : 392·4 cubic inches; or say that $1 + \frac{70 - 60}{460 + 60}$ of the volume at 60° = 400 at 70°, and therefore $400 \div 1\frac{10}{520} = 392 \cdot 4$. But this is assuming the pressure to be the same in each case; if not, a correction must be made for it. For instance, if in the above example the barometer stood at 30 inches when the 400 cubic inches of gas were at a temperature of 70° but at 29 inches when at 60°, the correction for the difference in pressure would be made thus:—As 29 : 30 : : 392·4 : 406; or it may be expressed thus, $400 \times \frac{30 \times (60 + 460)}{29 \times (70 + 460)} = 406$. Or, expressing these rules as formulæ, if v be the volume of any given weight of elastic fluid under any pressure and at 32° F., the volume v_1 which it will occupy under the same pressure and at any other temperature t of F. will be $v_1 = v + v \times \cdot 0020\dot{3}\ (t - 32)$.

This will be true if the ratio of the relative volumes be put u and u_1 instead of the ratio of the absolute volumes v and v_1, thus: $\frac{u}{u_1} = \frac{1 + \cdot 0020\dot{3}\ (t - 32)}{1 + \cdot 0020\dot{3}\ (t_1 - 32)}$.

Applying this to the question given,

$$\frac{u}{400} = \frac{1 + \cdot 0020\dot{3}\ (60 - 32)}{1 + \cdot 0020\dot{3}\ (70 - 32)} \therefore \frac{u}{400} = \cdot 9814,$$

and $u = \cdot 9814 \times 400 = 392 \cdot 5$.

The formula to find the relative volumes when both temperature and pressure change at the same time is,

$u = 1 \times \frac{p_1}{p} \times \frac{1 + \cdot 0020\dot{3}\ (t - 32)}{1 + \cdot 0020\dot{3}\ (t_1 - 32)}$, and applying this to the question in its second form,

$$u = 400 \times \frac{30}{29} \times \frac{1 + \cdot 0020\dot{3}\ (60 - 32)}{1 + \cdot 0020\dot{3}\ (70 - 32)},$$

$$u = 413 \cdot 8 \times \cdot 9814 = 406.$$

may be built to protect the sides and roof from fire, and provide a passage for the circulation of air.

If it is necessary to use the return airway for purposes other than ventilation, some other place must be selected for the furnace and a connection made with the return airway. The furnace is usually from 5 to 10 feet wide and the fire-bars 6 feet long, the space above being from 3 to 5 feet to the arch and below about 4 feet. A good plan is to let the breadth equal the diameter of the upcast shaft and to let the sectional area of the furnace drift be not less than the sectional area of the upcast shaft. The length of a furnace should not exceed 8 or 9 feet, and its breadth should not exceed 10 or 12 feet, because of the difficulty to feed and attend to the fire, and it is much better to have two furnaces than one large one. Figs. 369, 370, and 371 show in plan and section a double furnace with side arches.

All the work in contact with the flames of the furnace must be of fire-brick. The arches over the furnaces may be built with a thickness of fire-brick next the furnaces and have a blank space of a few inches interposed between the fire-brick and the ordinary brickwork beyond. The fire-bricks should be cemented together by ground fire-clay so as to effectually stand the heat from the furnace.

A well-constructed furnace will yield about 6,000 cubic feet per minute for each foot in breadth of fire-bars.

With a double furnace each grate of which is 8 feet wide, there would be therefore about $16 \times 6{,}000 = 96{,}000$ cubic feet per minute passing over it. But the work yielded by a furnace is affected considerably by the depth from the surface at which it is placed. In all cases the quantity of air will be as the square root of the difference between the temperatures of the downcast and upcast shafts and also as the square root of the depth from the surface. All other things being equal, the same furnace which is placed at a depth of 200 fathoms will produce double the quantity of air that it would yield at 50 fathoms. The horse-power ot a furnace is calculated from the ascertained weight of the air in the upcast and downcast shafts, which will be more particularly referred to later on. Sometimes the fires of underground steam-boilers act as furnaces assisted by the heat of the exhaust steam from the engines. Where the return air is liable to be charged with explosive gas, it is often made to enter the upcast shaft by a dumb drift, the point at which such drift enters the shaft being not less than 8 fathoms above the end of the furnace drift, so as to ensure that the return air, if inflammable, shall not be ignited by the furnace. By the Mines Act, 1887, the return air, unless it be so diluted as not to be inflammable, must be carried off clear of the fire by means of a dumb drift or airway. The dumb drift should have an inclination of not less than 1 in 6, or the smoke is liable to flow back from the running of the cages or the opening of doors. If all the return airways to the shaft are charged with inflammable gas, none of them will be available to supply the furnace with air, and in that case it must be fed with fresh air from the downcast. The fire should be kept thin and coal thrown on frequently, so that the air may pass freely through the burning fuel.

The *waterfall* is an expedient for producing ventilation. It may be caused by allowing the pump cisterns to run over, or pipes may be laid for the purpose, the water being scattered and not falling in one stream. It is not a very efficient means of ventilation, nor a very economical one, as the water has to be pumped again, unless under the exceptional circumstances of the mine having an adit by which the water would run level free to the surface. However, it is a very ready way to obtain air under exceptional circumstances, such as after an explosion.

The *steam jet* is another of the artificial methods of ventilation. It consists of steam, which may be brought down the shaft, being allowed to issue in small jets

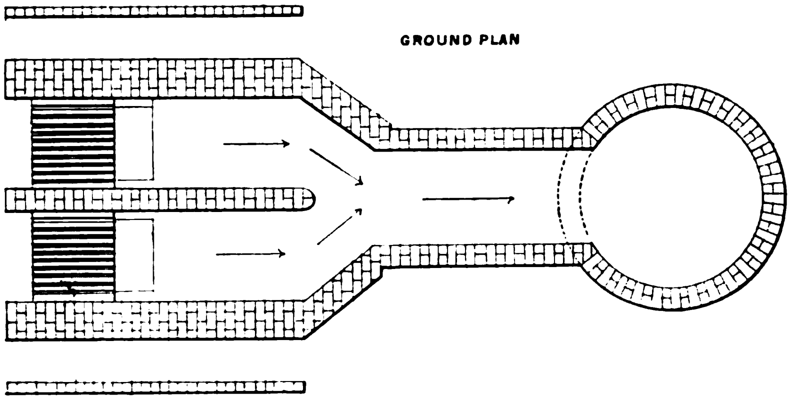


Fig. 369.

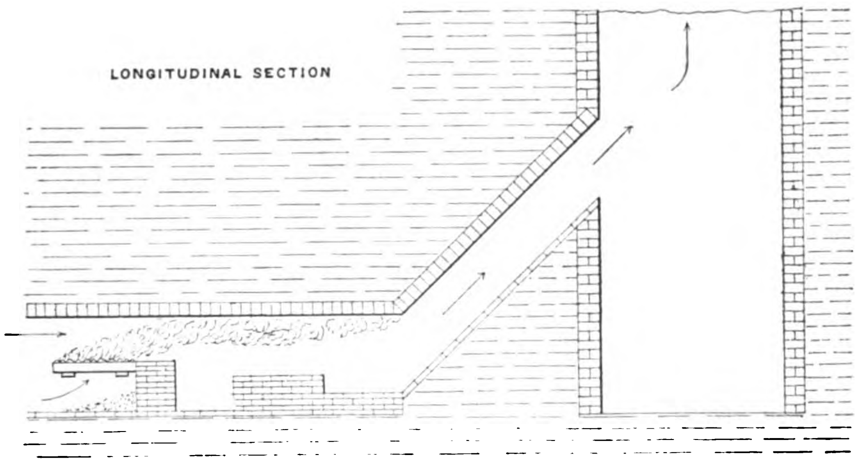


Fig. 370.

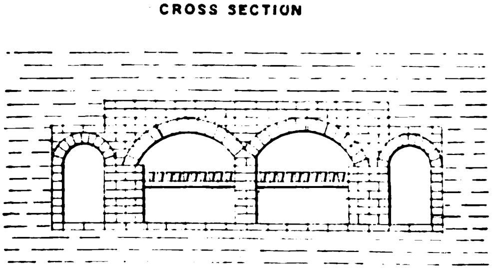


Fig. 371.

DOUBLE VENTILATING FURNACE.

from $\frac{1}{16}$th to $\frac{1}{8}$th inch in diameter directed upwards and placed in concentric circles round the bottom of the upcast shaft. It is not nearly so efficient as the furnace, and except in cases of emergency is not much resorted to.

MECHANICAL VENTILATION, OR THAT CAUSED BY THE USE OF MACHINERY.— This machinery may be divided into two classes, viz., the varying capacity or displacement machines, and the centrifugal.

Dealing first with the *Displacement* machines, the oldest form of mechanical ventilator is the air-pump, worked by a steam-engine.

The *Struvé* is one of this kind, shown in Fig. 372. It consists of two close-

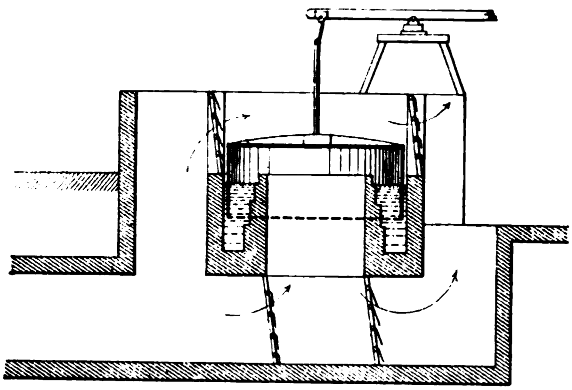


Fig. 372.—THE STRUVÉ VENTILATOR.

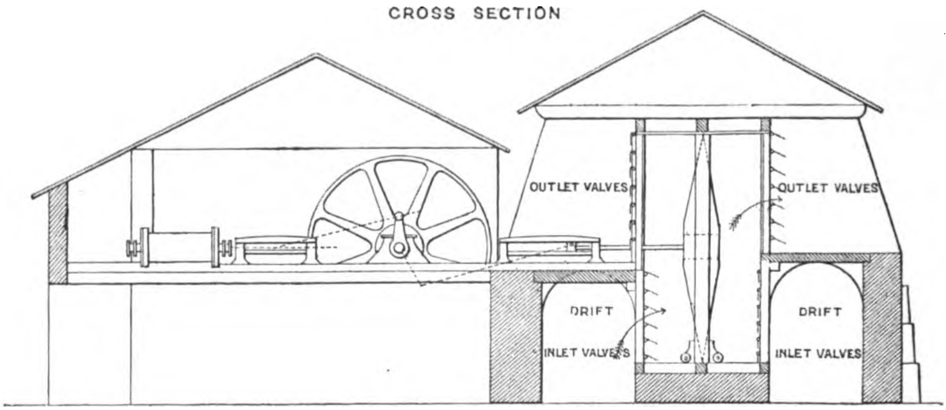


Fig. 373.—NIXON'S VENTILATOR.

topped airometers or pistons made of sheet-iron in the form of a gasometer, worked alternately, by means of a beam, up and down in a ring of water formed between the brickwork. Only one of the airometers is shown in Fig. 372, the other, similar to it, being placed at the other end of the beam. At the top and bottom of the walls of the chambers (of which there are two to each piston) are placed flap-valves, hung upon vertical gratings in the wood framing, and these valves are so arranged that in making the up-stroke as in making the down-stroke outlet being regulated to suit the volume of air under various circumstances. The most effective opening of the adjustable shutter is ascertained and fixed at any particular colliery experimentally. The percentage of useful effect given in the

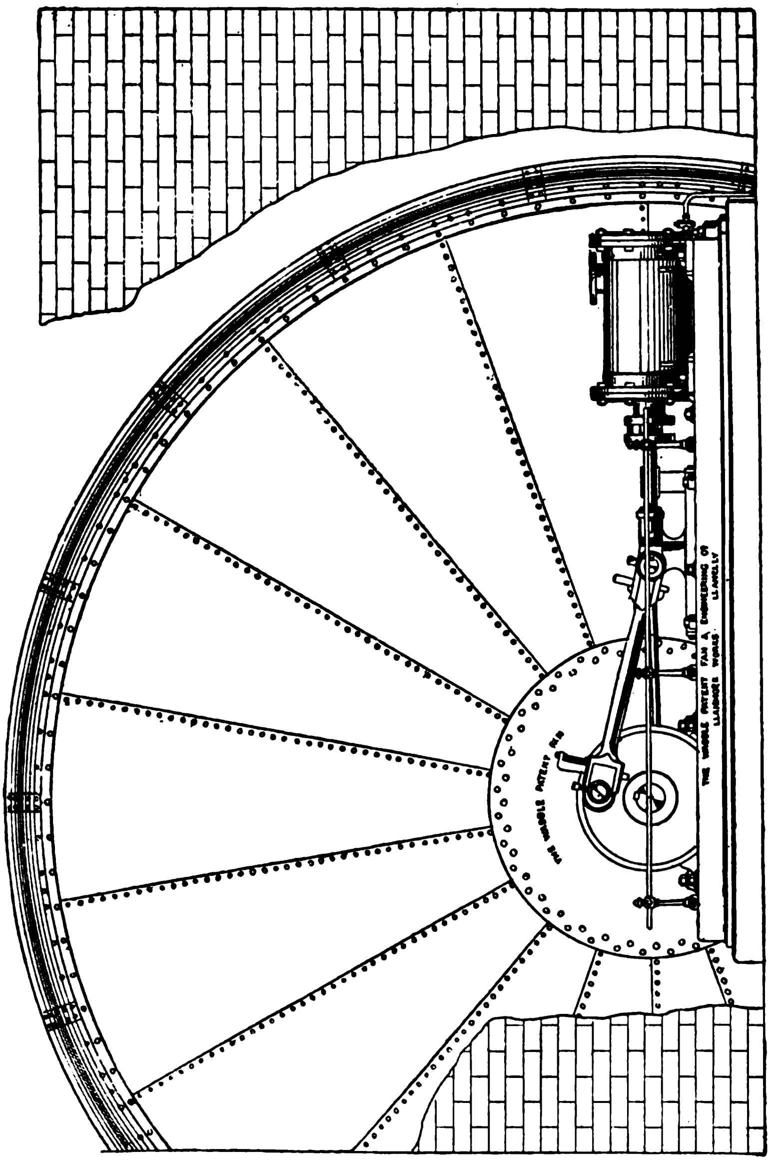


Fig. 384.—The Waddle Ventilating Fan.

same Report as quoted before, is 40 for the Guibal ventilator at Hilda Colliery, South Shields, but as 52·95 for that at Pemberton Colliery, Wigan.

The *Waddle* (Figs. 383 and 384) is an open-running fan, because it delivers the air all round the circumference into the atmosphere. For this reason its width is reduced at the periphery, and it is therefore very narrow in proportion to its diameter. The air is received on one side only. The blades are inclined backwards, these and the casing are all in one revolving piece which works completely free of vibration; when well built it forms an extremely compact, rigid ventilator. A high velocity of periphery is obtained by a moderate number

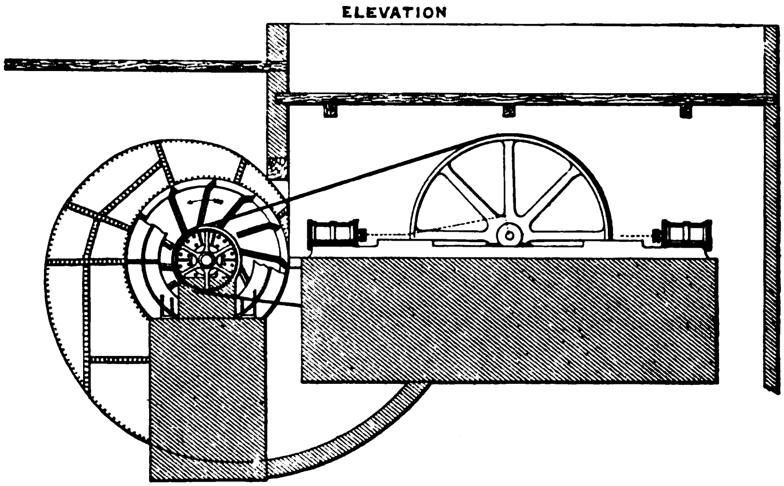


Fig. 385.

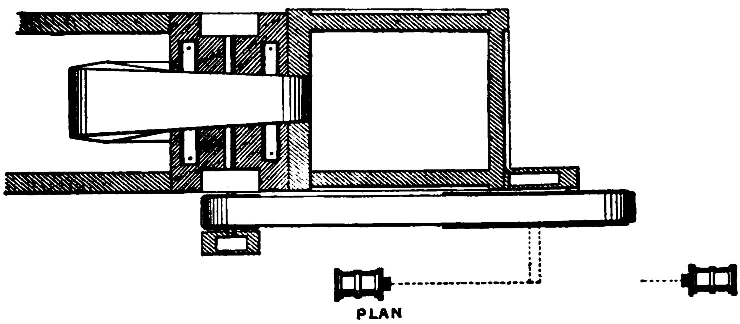


Fig. 386.

THE SCHIELE FAN.

of revolutions. The Report of the North of England Mining Institute Committee gives a useful effect of 52·79 for this fan at Celynen Colliery, South Wales.

A 30-foot one at Cwmaman Colliery—when tested in October, 1888, by the combined committees of the North of England, the Chesterfield, and the South Wales Institutes of Mining Engineers—gave a useful effect of 55·8 per cent. and circulated 134,394 cubic feet of air per minute with a water-gauge of 5·53 inches. Mr. Hugh Waddle, the inventor and patentee of the improved fan (Figs. 383 and 384) made by the Waddle Patent Fan and Engineering Co., Llanmore Works.

These small fans make from 100 to 300 revolutions per minute, and a fan-engine should not have a greater piston speed than 250 feet per minute. The most effective engines work expansively, and are condensing, and compound engines are the most economical in working.

As regards the economy or efficiency of furnaces, much depends on the depth of the upcast shaft, but with fans the depth is not an element requiring consideration. Besides being in every case a much safer and steadier means of producing ventilation, fans undoubtedly give better results in shallow mines. No doubt a furnace can be advantageously applied in a deep, dry shaft, the bottom of which is considerably below the bottom of the downcast. If, however, the mine be fiery, as most deep mines are, a fan should be used to produce the ventilation, and in arranging the upcast and downcast pits the air should be made to descend that sunk to the lowest level, and by what is called "ascentional ventilation" pass into each district. This is the most natural, and also the safest, means of ventilating, for the currents, on becoming warmer by their passage through the mine, are more easily carried up-hill than down, and, moreover, the gas is much more readily carried away with ascentional ventilation.

As the air, if allowed to follow its own course, would go by the easiest and shortest route, from the downcast to the upcast, the different currents must be guided by stoppings and doors into the various divisions of the mine. With a downcast 15 feet in diameter, giving an area of 176·715, there might be five separate currents branching off, each 35 feet in sectional area, without any increase in velocity. The benefits of thus splitting the air may be briefly stated. The same ventilative power will produce a larger volume of air by splitting than by carrying one body or current of air from the downcast round the workings to the upcast. Each district is supplied with fresh air, which comes from the downcast and returns to the upcast quite independently of the other currents, so that each is thus rendered purer and pleasanter for the workmen.

But as it almost invariably happens that the lengths of these air-currents are unequal, the air if allowed to take its own course would not flow in equal quantities into each division, or in such other proportion as from the circumstances of each district may be desirable. If left to itself the shortest air-course would probably get the largest share of the air; but there are other circumstances, such as sectional area and rise or dip workings, which would affect the quantities flowing into each district. To balance these splits, if they are very unequal, regulators are fixed in such a way that we can enlarge or diminish the aperture through which the air passes at pleasure. Usually this is done by means of a sliding door which moves horizontally in a groove in the wooden framing. Sometimes the door slides vertically, but whichever way it moves it should be kept locked in its proper position, by a properly-appointed person. It does not matter whether these regulators are placed in the intake or return airway, so far as regards restricting the passage for the air, but as they would form obstructions in the intakes which are used generally for haulage, they are placed in the return air-ways. The amount of opening for the passage of air is determined by the circumstances of the mine, and the relative quantities desired for each district.

Care must be taken both in the construction and erection of a regulator, a skilful workman being employed to fix it. The frame may be square or rectangular, and must be well morticed together. The kind and size of timber used depend upon the size of opening and its position, but must always be of sufficient strength. The grooves, whether formed at the sides or at the top and bottom, must fit the sliding shutter accurately, and yet allow of its being moved smoothly and easily. The boards used for the door must be a proper thickness and tongued and grooved in order to ensure tightness at the joints. The wood used in the construction must be well seasoned to prevent subsequent warping.

Fig. 388 shows a regulator formed in an ordinary underground door which is hinged at the side and provided with a latch. It is hung to ensure its self-closing, and, if intended only for occasional use, may, at other times, be kept locked. E is the door-frame having a little brickwork built over and at the sides to ensure a close fit and prevent the passage of air. A is the sliding shutter moving horizontally in the slots B. C is the door opening on the hinges D. The shutter is kept securely in its proper position to form the desired opening by means of the chain and padlock shown, or by some other equally satisfactory fastening. There should be in use only one padlock key kept by the man having charge of the ventilating arrangements.

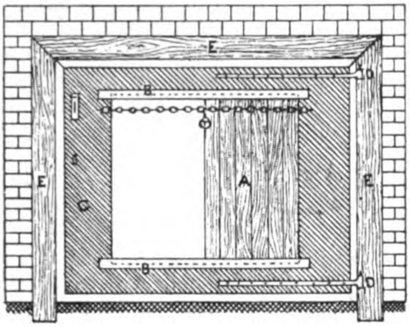


Fig. 388.—Regulator or Scale Door.

Where the door C is not required in the return airway a smaller door-frame may be used. This is fixed in surrounding brickwork built to reduce the size of the air-way, which, however, must not be too small for the formation of the largest air passage likely to be wanted. The amount of opening then left by the locked sliding shutter must be sufficient for a man to pass through to allow of all parts of the return airway being available as a travelling road.

However carefully regulators and doors are erected, they require constant supervision and re-adjustment, for the weight of the superincumbent strata squeezes and displaces them. If neglected, breaches in the masonry and timber, sooner or later, appear, and then the air finds its way through crevices and large openings behind the framing and through the open spaces in the broken or badly-fitting door. Where the inclination of the strata is great, there is constantly a weight on the high side of a level course road, tending to push a door or timbers placed in it towards the low side. This is more noticeable in deep collieries.

Where it is necessary to have regulators placed in airways traversed by horses, it is usual to hang sheets of brattice cloth from the roof across the road. This is a very convenient arrangement, as the air passes underneath the sheet which forms no impediment to the horses, but yields to their pressure on either side and falls into place by its own weight when the horses and trams have passed.

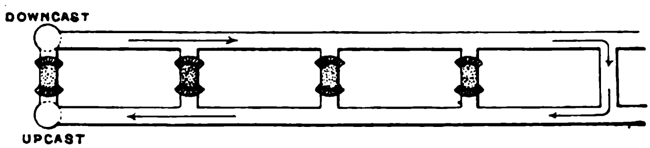


Fig. 389.—Permanent Stoppings in Fiery Mines.

The effect of placing regulators in the airways is fully dealt with later.

The stoppings used to prevent the air from being diverted from the desired course are variously constructed. For a non-fiery mine it may be found sufficient to build a brick or stone wall across the mouth of the road leading out of that intended for the main air-course and to back it up with rubbish to prevent leakage. In the case of fiery mines, however, these stoppings cannot be too strongly built, as they may possibly have to resist the effect of an explosion, and it must be remembered that a blown-out stopping allows the intake air to return through it to the upcast shaft. The best stoppings have two walls 15 or 20 feet apart, each with a curved outline, the convex sides of which are presented outwards and the space between the walls filled closely with stone or rubbish (see Fig. 389).

If only stoppings were built strong enough to resist the force of an explosion of fire-damp, the air-currents, unless stopped by falls, would still flow into the workings after the explosion, and many lives might thus be saved which would otherwise be lost through the after-damp. Occasionally it is necessary to have a travelling road through some of these bye-roads, and it is then impossible to have the way bricked up altogether. Doors of course must be fixed in such cases, but as it is impossible for these to resist the effects of an explosion, they should be avoided if possible. Where they must be put, they should have considerable attention paid to their design. If only required for travelling through, the frame should be set in masonry and the doors hinged from the top and open towards the intake side. This will ensure their falling to, despite of any want of thought on the part of the person passing through, and if instead of being placed vertically they form an angle of 70° or 80° with the floor, they will remain closed, the force of the intake air helping to keep them firmly in that position. Where the road has to be used both for travelling and the passing of tubs, a different kind of door is required so as in opening to allow the horse and tubs to pass. The framing of such doors should not be set quite upright, but sufficiently inclined for the door to fall and close by its own weight. A boy is stationed near to open it as required. Two similarly-constructed doors or more should be used between all main intake and return airways, and wherever the escape of air during the passing of the tubs is important, they should be placed at such intervals apart as to allow of the train or set of tubs and the horse to be between them, to prevent the two doors from being open at the same time. In fixing these doors care should be taken to make them airtight at the different joints and spaces behind the framing. Safety-doors are sometimes fixed in the roof, so that in the event of an explosion blowing the ordinary doors away, these safety-doors may fall into use or be dropped as soon as possible, and so take the place of the others.

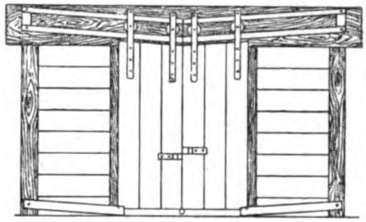


Fig. 390.

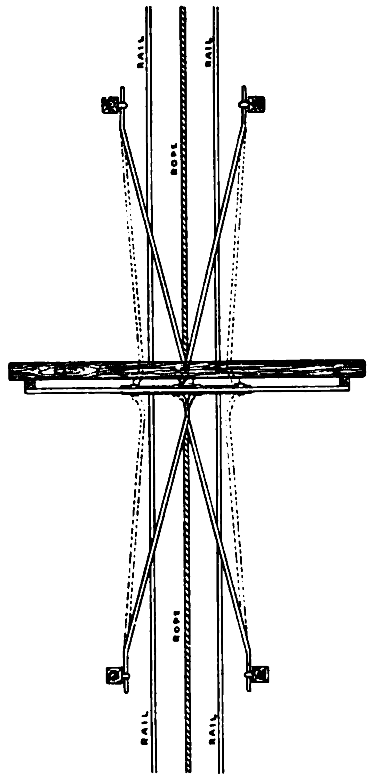


Fig. 391.

Figs. 390 and 391.—Elevation and Plan showing a Self-acting Ventilating Door used on a Main and Tail Rope Engine Plane at Hetton Colliery.

Figs. 390 and 391 show the elevation and plan of a self-acting ventilating door used on a Main and Tail rope engine plane at Hetton Colliery, in the county of Durham, a description of which appears in *The Iron and Coal Trades Review*

$\frac{6000}{30} = 200$ feet per minute for the latter quantity. There is no inconvenience in such a velocity, but if the same quantity of air were taken along an extremely small road the velocity would be unpleasant to those traversing it. A velocity of from 100 feet to 300 feet per minute is safe, convenient, and economical. At some collieries the main airways are swept by current velocities of 1,200 feet per minute. If an ordinary Davy lamp be used at the colliery 300 feet per minute is the extreme velocity the air should travel at in the return air-courses, where the air is mixed with fire-damp to a dangerous degree, because the flame of an ordinary Davy lamp will pass the gauze in a current of 6 feet per second, and it would not be safe to use such a lamp in currents approaching the explosive point. In shafts, in the main intake and return airways, the velocity cannot in practice be kept so low as in the further parts, because the former are necessarily limited in number. From 900 to 1,000 feet per minute is a fair velocity for the air to travel at in the shafts, and generally if it is required to pass from 150,000 to 200,000 cubic feet per minute, the diameter of the shaft should be not less than 14 feet.

There is a difference of opinion as to whether the upcast or downcast should be the larger. Owing to the expansion of the air, the volume ascending the upcast would require more area than the downcast, if the same velocity were maintained in the two shafts. But there is no absolute necessity for this, and the velocity of the air may be much higher in the upcast shaft. Irrespective of areas, it is more important that the upcast shaft be on the rise side of the downcast, but in the case of two shafts of equal depth whose surface levels are the same, a slight advantage will be derived from making the shaft whose area is least the downcast, so that the velocity of the air may be the same in each shaft. Other considerations may, however, suggest a different arrangement; for instance if a furnace were used, the larger shaft being wet, and the smaller one dry, the circumstances point to the advisability of making the larger shaft the downcast. Again, where a furnace is used, there may be wire ropes or tubbing that the smoke of a furnace would injure in the one shaft and not in the other, so that in making a decision as to which shaft shall be the upcast, the particular circumstances of the case must be well considered.

Experiments have been made with fans worked by compressed air placed underground at long distances from the bottom of the shaft. The object of these fans is to assist the main or general ventilating power. Where there are far-off splits otherwise difficult to ventilate, these auxiliary fans give good results and increase the total quantity of air circulating.

The usual method of ascertaining the quantities of air in mines is by the use of the *Anemometer*—an instrument for measuring the velocity of the air. The wind gives the vanes of the anemometer a speed proportional to that of the current, and the number of revolutions is registered upon the face of a dial fixed on the central part of the instrument. Biram's anemometer is the one in general use, and in it each revolution of the first pointer corresponds to one hundred feet in the linear motion of the air. To get the number of cubic feet passing per minute, multiply the velocity per minute, or in other words the recorded revolutions per minute by the sectional area of the airway. A slight correction should, however, be made owing to the friction of the anemometer. This correction is supplied by the maker along with the instrument.

Another method of air measuring formerly practised in mines worked with naked lights, but now seldom used, consists in exploding a small quantity of gunpowder and noting the time the smoke takes to traverse a certain distance, from which the velocity of the air is ascertained. The method of holding the anemometer is worthy of remark, as different results may be obtained in the same airway owing to the difference in the velocity of the current at different parts of the airway. It should be held at arm's length in front of the body, the vanes should be kept square with the current of air, and the anemometer should be slowly moved uniformly over the whole area of the airway from a point near the floor to a point near the roof. Several trials should be made in the same place and the average result taken. In large airways and fan drifts, where the greatest accuracy may be required in testing the efficiency of fans, the most correct method is to fix fine wires or strings across the road from side to side and from floor to roof, at regular distances apart, the one set of strings being placed at right angles to the other, so as to divide the airway into a number of even divisions, and then to note the revolutions of the anemometer during 1, 2, or 3 minutes at each division. The mean of these are taken. But the operation should be repeated in cases where great accuracy is desired, and if the mean result differs materially from the first mean result, it will be necessary to go through the operation again, and perhaps again.

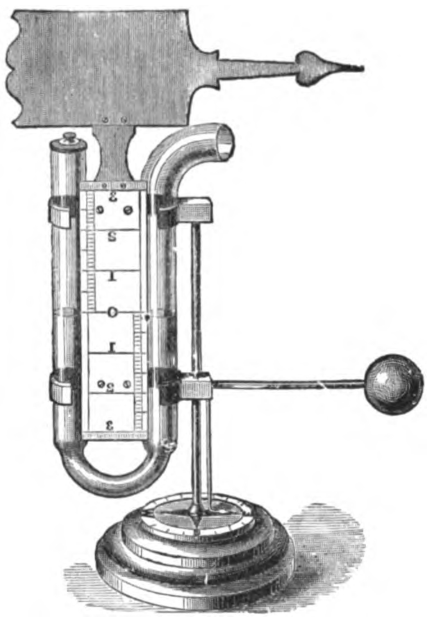


Fig. 393.—Lind's Anemometer.

Wolf's Anemometer.—Wolf is said to have invented the first anemometer. It has four sails, like those of a wind-mill, which turn on a horizontal axis; this axis is connected by wheel-work with another in which is inserted one end of a bar carrying a weight. The wind pressing upon the sails causes this bar to turn in a vertical plane; and, when it is in such a position that the weight on it counterbalances the pressure of the wind, the angle which it makes with a vertical line passing through the axis affords a measure of the wind's force.

Wolf's anemometer is no longer in use, as it has been superseded by later forms.

Lind's Anemometer.—The anemometer or wind-gauge invented by Dr. Lind is shown in Fig. 393. It consists of two glass tubes about 8 or 9 inches long and about $\frac{1}{4}$th of an inch in diameter. As now made they are placed vertically, and connected at the base by a small bent glass tube only $\frac{1}{10}$th of an inch in diameter to check oscillations of the water caused by sudden gusts of wind. The upper end of one tube is bent to receive the wind blowing horizontally into it, while the upper end of the other is covered by a brass cap with a hole in it. The hole allows water in the tube to be under the influence of atmospheric pressure while the cap shields the surface from the action of gusts of wind. The instrument is upheld by two brass clips, one near the top and the other near the bottom; the clips terminate on one side in square ends. The square end of the lower clip is pierced right through by a small circular opening, and the square end of the upper one is partly bored out by a similar sized opening. By this means the instrument can at any time be placed on the steel spindle, which is screwed vertically into the stand, or taken off for part-filling. A scale of inches and tenths is fixed between the two legs of the glass tube, the zero point of which is about half way up the tubes. The scale is graduated above and below its zero line, the upper divisional lines being cut on the opposite side to that of the lower markings so as to suit the water displacement in the two limbs. Before using the instrument water is poured in until the tubes are half full and the scale is then adjusted with its zero line level with the water in the tubes. The stand must be placed perfectly level to ensure the same height of water in the two tubes before it is disturbed by the wind. The whole instrument being free to turn easily about the upright spindle, the mouth of the bent tube is made to face the wind by the application of a weather-cock. The water is then forced down the one tube and correspondingly up the other. The difference between the heights of the surfaces of the water in the gauge will be the height of a column of water, whose pressure is equal to the force or momentum of the wind, blowing against an equal base.

The stand has the cardinal points marked and as the direction of the wind changes, the instrument responds, carrying with it the index-pointer, which moves round the stand dial-plate and thus shows the direction of the wind. The projecting handle terminating in the knob on the right hand side of the illustration is for lifting the instrument on and off the stand, and for balancing it on the spindle.

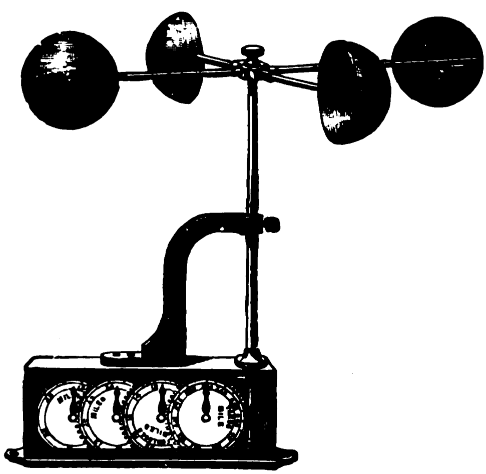


Fig. 394.—Robinson's Anemometer.

Robinson's Anemometer.—This has been much used at meteorological stations. It was invented by Dr. Robinson, of Armagh, and has been modified and improved by Mr. Casella. It consists of four equi-distant hemispherical cups upon which a passing current of air acts, see Fig. 394. The cups are fixed to four folding horizontal arms which at their other extremity are attached to a vertical shaft or axis. The wind presses forcibly on the concave surface of a cup when its diametrical plane faces it, and feebly on the convex surface of the cup on the opposite arm which is at the same time presented to it. The difference in the pressure of the wind on the two cups causes the arms and the vertical shaft to revolve. Each cup in turn receives the full force of the wind, and the speed attained by the cups amounts to one-third of the wind's velocity. A thread is cut at the lower end of the vertical shaft, into which a toothed wheel fits. The revolving shaft causes this wheel to move on its axis, and, by means of gearing, this in turn gives motion to the other wheels, which carry suitable indices showing the space traversed by the wind for any timed period the graduations are capable of registering.

Robinson's anemometer folds up and is carried in a small case. It is very useful for registering the velocity of the wind in the open, but is not so well adapted for use in mines.

The *Biram* anemometer has been manufactured by Messrs. John Davis & Son, of Derby, since the year 1847, and is adapted to indicate the velocity of air currents in mines, sewers, or any confined passage within certain limits. It is made in 6-inch, 4-inch, and 3-inch sizes, as shown in Fig. 395, and in a 12-inch size with a different arrangement of the vanes. In the illustration the ten vanes are set at an angle of 45° to the axis, like the sails of a wind-mill. There is a central dial-plate, the circle on which is graduated up to 100, and round which they should be tested from time to time. For that purpose they may, on payment of a fee, be verified at Kew Observatory.

The *Thermometer* is a measurer of temperature, mercury being used for ordinary temperatures. It depends for its action on the fact that all bodies with the rise and fall of their temperatures expand and contract. It consists of a glass tube closed at the top, with a bulb at its bottom end, and having mercury placed in it. A scale of degrees is fixed to the tube. As used in mines the thermometer registers the temperature of the air, and we are able to measure the difference of temperature between the air in the downcast and upcast shafts or at any desired point in the workings.

Thermometers are graduated according to three scales, viz., Fahrenheit's, which is that commonly used in England; the Centigrade scale, which is that generally used in the scientific world; and Reaumur's scale, which is that taking its name from a French philosopher, who constructed his thermometer with alcohol of such a strength that 1,000 parts at the freezing point of water became 1,080 parts at its boiling point.

On Fahrenheit's thermometer 32° indicates the freezing point and 212° the boiling point of water, and the space between these two fixed points is divided into 180 even divisions; these even divisions are produced above and below 32° and 212°.

In the Centigrade thermometer 0° indicates the freezing point and 100° the boiling point of water, the space between these two points being divided into 100 even divisions. In Reaumur's 0° indicates the freezing and 80° the boiling point of water, the space between these two points being divided into 80 even divisions. It is plain therefore that—

$$180^\circ \text{ Fah.} = 100 \text{ Cent.} = 80^\circ \text{ Reaum.}$$

and therefore
$$1^\circ \text{ Fah.} = \tfrac{5}{9} \text{ Cent.} = \tfrac{4}{9} \text{ Reaum.}$$

To transfer Fahrenheit degrees to the other scales we must first subtract 32°, in order that the number of degrees from the freezing point may be ascertained. These multiplied by $\frac{5}{9}$ths will give the equivalent number of Centigrade, and by $\frac{4}{9}$ths the equivalent number of Reaumur degrees.

To reduce Centigrade and Reaumur degrees to the Fahrenheit scale, multiply by $\frac{9}{5}$ and $\frac{9}{4}$ respectively and add 32°.

If the temperature be below the zero in any of the scales, a minus sign (—) is placed before the number thus: -5° Fah. means 37° below freezing point.

The following examples may be tested:—

Fah.		Cent.		Reaum.
190°	=	$87{\cdot}\dot{7}^\circ$	=	$70{\cdot}\dot{2}^\circ$
155°	=	$68{\cdot}\dot{3}^\circ$	=	$54{\cdot}\dot{6}^\circ$
128°	=	$53{\cdot}\dot{3}^\circ$	=	$42{\cdot}\dot{6}^\circ$
3°	=	$-16{\cdot}\dot{1}^\circ$	=	$-12{\cdot}\dot{8}^\circ$
-15°	=	$-26{\cdot}\dot{1}^\circ$	=	$-20{\cdot}\dot{8}^\circ$
-40°	=	$-40{\cdot}^\circ$	=	$-32{\cdot}^\circ$

It must be borne in mind that a thermometer does not give the absolute expansion of the mercury, but the difference between the expansion of the mercury and that of the glass. Mercury expands about 7 times more than glass.

Mercury, or quicksilver, is a metal having the remarkable property of being fluid at ordinary temperatures. Mercury boils at 660° Fahr., giving off vapour of specific gravity 6·976, and it may therefore be distilled. It always forms one of the metals of an amalgam, and is much used in the process of amalgamation and decomposition in order to separate gold and silver from their ores. Mercury is between the two fixed points is divided into 100 equal parts, which are produced above and below as far as required. For Fahrenheit the same space is divided into 180°, the freezing point being 32°. For Reaumur the division is into 80°, the freezing point being 0° as in the Centigrade.

The experiments are made when the barometer is at 29·95 inches. A mercury thermometer cannot be graduated lower than −40° on Fahrenheit's scale, because the mercury then congeals. It may be graduated upwards to 600° Fahr., but not higher, to be trustworthy, for mercury boils at 660° Fahr.

If the vacuum in the tube of a thermometer be good, on the instrument being inverted, the mercury strikes against the top of the tube with a clear ringing sound. If, after being in use some time, it is tested by placing it in melting ice, and the mercury then does not stand at the freezing point mark, the error is

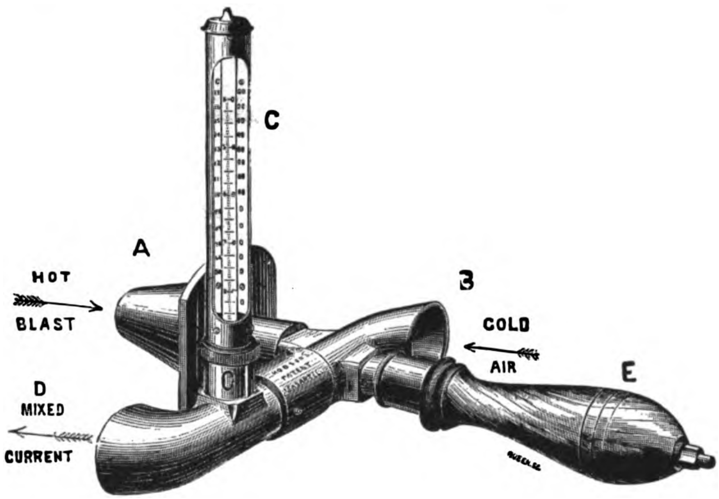


Fig. 407.—Hobson's Patent Hot Blast Pyrometer.

called "the displacement of zero." It arises from the curious fact that sometimes bulbs do not perfectly contract for two or three years after blowing. In the very best instruments the bulbs are kept empty for that length of time before being filled. A thick glass bulb is less likely to change than a thin one.

The three thermometer scales referred to are shown in diagram form in Fig. 406. It is drawn to scale, so that a rough comparison of the three sets of figures corresponding to a height of the mercury may be seen at a glance within certain limits.

As it is almost impossible to freeze the liquid, alcohol is used in the manufacture of thermometers to register very low temperatures. There is no limit to the downward graduation of such thermometers, but their upward range is much more limited than that of mercurial thermometers.

High temperatures are measured by pyrometers. These are instruments for measuring all gradations of temperature above those which can be directly indicated by the mercurial thermometer. Many pyrometers depend for their action upon the uniformity of the expansion and contraction of metal.

In Hobson's patent hot blast pyrometer, made by Mr. J. Casartelli of Manchester, mercury is used, and although this is incapable of directly indicating temperatures above 600° Fahr., it is made to do so by the temperature of the hot blast being first reduced to a known degree. Fig. 407 shows the instrument. By means of is cooled. The evaporation will increase in proportion to the dryness of the air, and the reading indicated on the scale of the wet-bulb thermometer will then be considerably lower than that on the dry. The difference between these readings will be less and less the more humidity there is in the air, and will become zero when the air is completely saturated. Hence the difference of the readings on the scales of the two thermometers is a measure of the dryness of the air. From published tables, it may be ascertained from the two thermometer indications, the relative humidity, 100° being saturation.

Watery vapour is never altogether absent from the air we breathe, and on the other hand, the air is very rarely saturated. The amount of watery vapour which can be contained in a given space of air, depends chiefly on the temperature. Directly the temperature of saturated air is raised, it becomes capable of holding more watery vapour, and is therefore no longer at the point of saturation. If on the other hand its temperature be lowered, it can no longer retain all its vapour in solution, and a portion passes from the gaseous to the liquid form, and will be deposited as dew or rain. After the sun has set, many bodies which were warmed by its rays, radiate their heat until they become cooler than the air

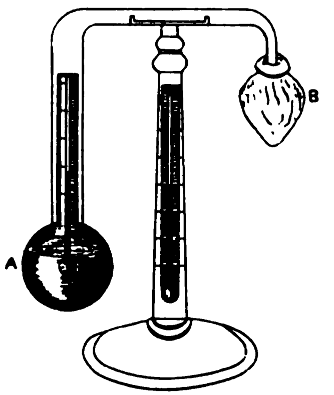


Fig. 411.—Daniell's Hygrometer.

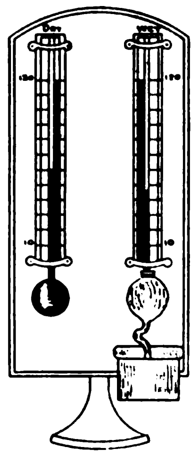


Fig. 412.—Hygrometer in Ordinary Use.

around them. A clear and calm night, free from wind, is favourable to rapid radiation; moisture condenses on the ground, the condensation being most abundant on those bodies which radiated their heat most. Dew is thus formed by the cooling of the air in contact with the ground: rain is produced by the cooling of a mass of air below its dew-point.

The temperature of air as it descends a mine may be raised or lowered, according to the season of the year, and the depth of the shaft. Workings of moderate depth will be warmer in winter and cooler in summer than the air at the surface, the temperature of the flowing air being closely assimilated to that of the galleries soon after leaving the shaft. As the air travels onward, its temperature is raised by workmen, horses, and lights in the mine. The exact state of the air is obtained at different points of the mine by means of the hygrometer. The Royal Commissioners on Coal in the United Kingdom, give several observations of this kind in their Report of 1871.

The following, taken from their Summary for the County of Durham, may be of interest as showing variations in the hygrometrical condition of the air at different collieries, having shafts of varying depth.

SUMMARY OF HYGROMETRIC OBSERVATIONS IN COAL MINES IN THE COUNTY OF DURHAM, ALL OBSERVATIONS HAVING BEEN MADE IN A WORKING FACE.

Name of Mine.	Depth in Feet.	Distance from Down-cast Shaft in Yards.	Dry Bulb.	Wet Bulb.	Relative Humidity, 100° being Saturation.	Remarks.
Jane Pit, Eppleton Colliery	1,395	4,332	73·5°	73·5°	100·	
	1,395	4,440	74·	74·	100·	
	1,395	4,560	74·5	74·5	100·	
Caroline Pit, Eppleton Colliery	838	2,560	64·	64·	100·	
	1,040	3,364	65·5	65·	97·2	
	1,012	3,328	65·5	65·	97·2	
	1,030	3,365	65·	65·	100·	
LadyPit, Elemore Colliery	970	1,866	67·5	67·5	100·	
	930	2,660	66·	66·	100·	
Wharton Colliery	88	3,454	65·	64·5	97·2	
	900	4,246	64·5	64·	97·2	
	924	3,696	68·	67·	94·7	
	1,254	1,826	71·75	70·	91·	
Monkwearmouth Colliery	1,646	3,256	81·25	79·5	92·	
	1,640	3,216	82·25	81·25	95·6	
Ryhope Colliery	1,560	2,762	73·	71·	90·2	Under the sea.
Murton Colliery	1,374	4,532	70·	69·5	97·4	
Monkwearmouth, 2nd observations	1,646	3,256	81·	78·	86·8	These observations were taken by Mr. L. Wood and Mr. Dickinson conjointly.
	1,640	3,216	81·	78·	86·8	
Monkwearmouth Colliery, 3rd observations	1,646	3,256	81·	74·	70·4	It is the practice in this colliery to water the roads to keep down the dust, but this practice had been suspended for 11 days immediately preceding this 3rd set of observations.
	1,640	3,216	81·5	77·	80·6	
Seaham Colliery	1,995	2,200	78·	67·	60·3	

Watering the roads affects the air as seen by comparing the different observations at the Monkwearmouth Colliery with each other.

The *Barometer* is an instrument used for measuring the pressure of the air. If a glass tube a yard long and closed at one end be filled with mercury and inverted with the finger placed over the open end until that end be placed in a vessel containing mercury and then removed, a part of the mercury will run out, but the tube remains filled to a height of about 30 inches above the surface of the mercury in the vessel. That is, the ordinary pressure of the atmosphere is sufficient to balance a column of mercury 30 inches high. But the pressure of the atmosphere varies in this country between 28 and 31 inches of mercury. The ordinary barometer has a scale and a sliding vernier fixed to it, by means of which it taken place in the barometer. For this purpose a wheel-barometer may be used, the float of which gives motion to a well-balanced lever. The shorter end of the lever is attached to the float, and the longer sometimes carries a pointer which is caused by clockwork to puncture at regular intervals a ruled card slowly moved under the point. A line joining the punctures shows in diagram form the changes in the height of the column.

Instead of a wheel barometer, Messrs. John Davis & Son supply a large and powerful aneroid and an eight-day clock. Between the aneroid and clock a vertically placed cylinder having a paper attached to it ruled to coincide with the barometer scale, revolves. Responsive to the action which takes place in the aneroid, a pencil moves up and down, and every hour is made to mark the paper by mechanism connected with the clock. A weekly barometric chart obtained from one of their self-recording aneroids, reduced to sea-level, is communicated and published in the *Colliery Guardian* by Messrs. John Davis & Son.

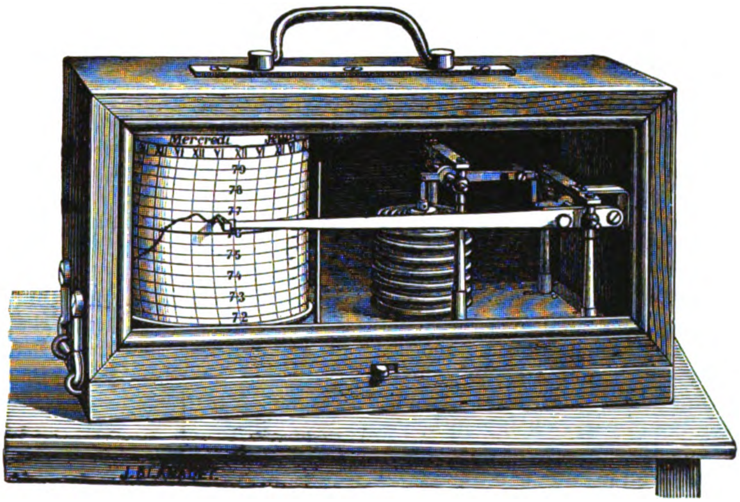


Fig. 418.—Self-recording Aneroid Barometer.

Fig. 418 is an illustration of a Self-Recording Aneroid Barometer or Barograph. The thin metal box of corrugated form is the vacuum chamber which is the prime mover as in the ordinary aneroid, but is of greater height. Its corrugations give the box a certain amount of elasticity which yet is sufficiently rigid not to collapse with the pressure of the atmosphere when the chamber is exhausted. The small movement of the corrugated surface is greatly increased as it is communicated by well-designed and delicately constructed levers to the pen which is held up to the paper-covered drum by the gentle pressure of the lever. These levers are compensated for temperature. The object of multiplying the movements of the vacuum box is to make variations in the atmospheric pressure more apparent in the line traced by the pen at one end of the long lever, which line will be more or less undulating in accordance with the rapidity and range of fluctuations in the atmospheric pressure.

A properly ruled paper chart, supplied with the instrument, is wound round the drum, which is revolved by clock-work. The drum must be of sufficient height to take a 4 or 5-inch diagram card, which allows for movements of the pen over the variations in height of the mercury column in this country. The men, who would then be in a position to caution the workmen of repeated falls in the barometer.

It is not suggested that in efficiently ventilated mines where proper precautions are taken changes in the atmospheric pressure are in themselves sufficient to produce an explosion. The ventilating current passing along any face ought, under normal conditions of gas emission, to be ample to carry the gas harmlessly away even when affected by a gradual reduction of pressure. The Royal Commissioners on Accidents in Mines, speaking on this question, say: "Some remarkable instances have appeared to support the view that explosions accompany unusual depressions or specially rapid falls in the barometer, but the evidence which has been adduced to show that this view is of general application seems to us most imperfect and untrustworthy. Indeed, the absence of a general connection between colliery disasters and barometric changes is practically established by the tables compiled by some of your Majesty's Inspectors of Mines, by the Northern Institute of Engineers, and recently by Mr. Thomas Embleton in communications made to the Midland Institute."

Collieries having only feeble currents of air are of course much more seriously affected by a falling barometer, as gas comes off from the goaves much more freely with the reduced pressure then indicated. There is no excuse for neglecting the ventilation, whatever the state of the barometer, but the changes in the weather should be carefully watched, although gas in the mine is more sensitive to atmospheric disturbances than mercury, and abnormal issues are known to have preceded a fall of the barometer.

Even with a continuously high barometer an explosion may readily be brought about by neglecting the usual precautions, such as the opening of doors, or the derangement of brattice or air pipes, which in fiery mines would quickly cause gas to accumulate.

When the barometer is at 28 inches, the pressure of the air per square foot is about 1,979 lbs., and when at 31 inches it is about 2,191 lbs. To find the theoretical quantity of gas that would be given off from each 1,000 cubic feet of space in the gas-charged goaves of a fiery mine due to a fall of the barometer from 31 to 28 inches :—$31-28=3$ inches difference. Then, as $31 : 3 :: 1{,}000 : 96{\cdot}77$ cubic feet. So that if the goaves could be measured and were found to contain 10,000 cubic feet, there would be $96{\cdot}77 \times 10 = 967{\cdot}7$ cubic feet of fire-damp given off by them. The cubic contents of goaves cannot be measured with any degree of certainty, however, and any estimate of goaf contents must be based on the superficial area over which the goaf extends and an assumption of a certain percentage of it being more or less open. The amount of goaf present in some old collieries necessitates the greater caution in observing and noting the changes of the barometer.

Atkinson, in his *General Principles of Ventilation*, says :—"In ordinary states of the weather mercury is about 10,800 times as heavy as the same volume of air near the surface of the earth, and hence about 900 feet of ascent or descent makes a change of 1 inch of mercury in the height of the barometer." Again: "The air at the surface of the earth is generally pressed by the whole of the air above it, to an extent measured by 29·922 inches of mercury (reckoned at the density due to melting ice 32°), as shown by our common barometers; a pressure equal to 2,116·4 lbs. per square foot. To give this pressure we should require the air of the atmosphere to be 26,216 feet high, if it was all as heavy as the air at the earth's surface."

In forming a rule to meet the fluctuations of the barometer, take this 26,216 feet as being the height of the atmosphere at sea level, which gives its appreciable weight or pressure on the earth's surface.

Then for pits on the datum line of sea level will be obtained the following rule to find the height of the mercurial column corresponding to shaft depths :—

$I = \frac{D \times B}{26,216}$, where I = inches of mercury due to the shaft depth; D, depth of shaft in feet; and B, height of barometer at the pit top. The barometer reading at the pit bottom will then be the reading at the top + I, or in case where the height of the barometer is given at the top and bottom of a pit whose depth it is desired to know, and where I represents the difference of the two barometer readings, $D = \frac{26,216 \times I}{B}$.

Supposing a question like the following has to be answered. The barometer at the top of a shaft is 30·2 inches, the thermometer is 65° F., the depth of the shaft is 1,100 feet, and the thermometer stands at 75° F. at the pit bottom, say what is the difference in the pressure of the air at the top and bottom of the shaft, and the difference in the reading of the barometer.

Here $I = \frac{1,100 \times 30{\cdot}2}{26,216} = 1{\cdot}267$. Therefore the reading of the barometer at the pit bottom is $30{\cdot}2 + 1{\cdot}267 = 31{\cdot}467$. To get the weight of a cubic foot of air at the shaft top by Atkinson's formula, $\frac{1{\cdot}3253 \times 30{\cdot}2}{459+65} = {\cdot}07638\,14$ lb. Similarly, $\frac{1{\cdot}3253 \times 31{\cdot}467}{459+75} = {\cdot}078096$ lb. as the weight of a cubic foot of air at the shaft bottom.

Hence, $\cdot078096 - \cdot0763814 = \cdot0017146$ lb. difference in the weight per cubic foot of the air at bottom and top of the shaft.

The *Water-gauge* is a very simple instrument, and consists of a glass tube bent like the letter U, both ends of the tube being open. A little water is placed in the bend of the tube, which forms the bottom part of it. A sliding scale of inches and decimals of an inch is attached to it. At the top of one arm of the tube is placed a nose-piece, by means of which it is passed through a door, and, by so doing, one side of the tube is placed in contact with the air on one side of the door, and the other is exposed to the influence of the atmosphere or air current at the other. Where a difference of atmospheric pressure exists, such as would be between the intake and return currents of air near the shaft, or between a fan drift and the outside air on the surface, the water is depressed in one side of the tube and raised in the other. The scale of inches and decimals shows the difference of level in the tubes. The instrument is used thus to show the force of the air current generated, whether by furnace or fan.

Fig. 419 shows Daglish's water-gauge for use in mines as made by Messrs. John Davis & Son, of Derby. It is fixed by small brackets to a mahogany back for safe carriage, and has a small thumb-screw at the bottom by means of which the scale can be adjusted; a level to ensure its being inserted in an upright position when making an observation; a 6-inch scale divided into inches and tenths; and a brass cap with branch pipe at one end for insertion where required. This form of water-gauge is also made with longer or shorter scales to suit requirements.

Davis' pocket water-gauge is only 6 inches in length, that of the tube being $4\frac{1}{2}$ inches and the scale 3 inches. This leaves a travelling space of $1\frac{1}{2}$ inches for the adjustment of the scale. The gauge is made with a level and with or without a thermometer, the card to which the instrument is attached forming the back of the case in which it is carried. This instrument is very handy, but does not admit of readings higher than 3 inches. Other lengths of tubes and scales are, however, made in the pocket form to suit a greater range if desired.

In some instances it is extremely difficult to obtain accurate readings of the ordinary water-gauge. The variations in pressure which are constantly going on cause a vibratory motion of the water in the glass tube, and this is particularly noticeable where there is a displacement ventilator or with any mechanical ventilator where the upcast shaft is used for winding coal. The running of the cages and the continual opening and shutting of the doors across the pit-top cause fluctuations in the water. Under such circumstances more or less successful attempts are made to watch the oscillations in both legs simultaneously so as to adjust the scale with its zero mark at the mean of the fluctuation in one leg while reading the scale at the mean of the extreme movements in the other. Such difficulties have naturally led to improvements in the instrument. The idea of contracting the bend in the glass tube so as to reduce the passage for water between the tubes is due to Mr. John Daglish.

Fig. 419.

THE DAGLISH WATER-GAUGE.

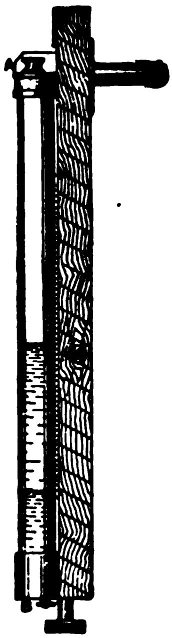


Fig. 420.

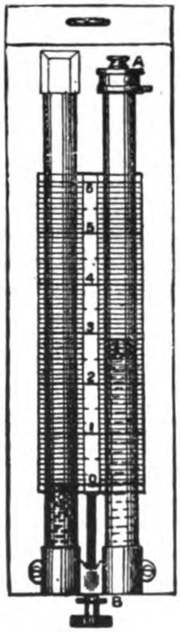


Fig. 421.

THE DAVIS IMPROVED WATER GAUGE.

An improved form of water-gauge made by Messrs. John Davis & Son is shown in Figs. 420 and 421.

It consists of two parallel glass tubes with their lower extremities sealed into a hollow brass pedestal through which there is a connection or water passage between the tubes. In the centre of the brass pedestal is a tap B pierced by a small hole, making a passage for the water through it when the tap is in a certain position. This tap takes the place of the reduced bend in the Daglish form of water-gauge. There is difficulty in obtaining a continuous U-shaped glass tube with a small hole or contraction of uniform size in the bend, but there is no difficulty in making or retaining the hole in the brass tap of a standard uniform size. The restricted passage for the water through the hole in the tap prevents the oscillation due to varying pressure being so marked as in the Daglish form of water-gauge. The throttling of the water at the tap prevents quick vibratory motions whilst it still affords a sufficiently free connection between the two columns of water.

Two or more persons frequently differ in the readings obtained from the

the total weight in the downcast. For the upcast, $\frac{1\cdot3253 \times 30}{459+150} = \cdot06529$ lb., the weight of a cubic foot and multiplied by 300 and by 113 = 2,213·34 lbs. as the total weight in the upcast. 2,596·74 — 2,213·34 = 383·4 lbs. as the difference in weight in the two shafts. Therefore to find the motive column, as 2,596·74 : 300 :: 383·4 : 44·3, as given before. If the quantity of air circulating were 150,000 cubic feet per minute, proceed to find the horse-power exercised by the furnace thus:—The weight of a cubic foot of air in the downcast being ·0766 lb. at the bottom of the pit, there would be a pressure of $\cdot0766 \times 300 = 22\cdot98$ lbs. due to its mere weight. In the upcast the weight would be $\cdot06529 \times 300 = 19\cdot587$ lbs. and $22\cdot98 - 19\cdot587 = 3\cdot393$ lbs. as the difference of pressure on each square foot of area. Therefore $\frac{150,000 \times 3\cdot393}{33,000} = 15\cdot4227$ horse-power.

Some authorities use the formula $M = D \times \frac{T - t}{t + 459}$ which gives the length of motive column in feet of air of the temperature in the upcast, and this formula will give the same pressure in pounds per square foot as $M = D \times \frac{T - t}{T + 459}$ which gives the length of motive column in feet of air of the temperature of the air in the downcast, but the former gives also the theoretical velocity, viz., $v = \sqrt{2\,g\,M}$, with which the air would escape from the upcast neglecting friction. Applying this formula to the foregoing example $M = 300 \times \frac{150 - 60}{60 + 459} = 52\cdot02$. Or, taking the total weight as before, worked out in the downcast at 2,596·74 lbs., and in the upcast at 2,213·34 lbs., with a difference between the two of 383·4, the motive column in feet of air of the temperature of the upcast may be found by proportion thus:—As 2,213·34 : 300 :: 383·4 : 51·97, as before, except a slight difference owing to the loss of decimal places.

The simplest way of getting the difference of pressure on each square foot of area between the upcast and downcast is by using the water-gauge. If this had been tested at the separation doors between the upcast and downcast shafts in the case just given, it would have read about ·65″, because $5\cdot2 \times \cdot65 = 3\cdot38$ lbs., as will be understood by reference to the remarks on the water-gauge.

This head or motive column may be easily converted into inches of water, as shown in the water-gauge. The motive column is always expressed in feet, and as the weight of a foot of air at 60° F. is 0·0766 lb., and that of an inch of water is 0·036 lb., this gives a pressure of $\cdot036 \times 144 = 5\cdot184$, but, as before stated, usually taken at 5·2 lbs. per square foot; therefore multiply this motive column by ·0766 ($= \cdot0766 \times 44\cdot3 = 3\cdot393$ lbs.), the pressure of a foot of air, and divide the product by 5·2 lbs., the pressure of an inch of water, to find the indication of the water-gauge. $\frac{3\cdot393}{5\cdot2} = \cdot652$. Or, what is the same thing, divide the motive column at once by 68 to find the water-gauge, because $\frac{5\cdot2}{\cdot0766} = 68$ nearly. Taking then the motive column at 44·3 feet in the case being considered, the water-gauge would read $\frac{44\cdot3}{68} = \cdot6514$.

Or, if the motive column had been expressed in feet of air of the temperature of the air in the upcast, a foot of air at 150° F. as shown, weighs ·06529 lb., and if the motive column of 52·02 feet be multiplied by ·06529 lb., the pressure of a foot of the air, and the product be divided by 5·2, thus $\frac{52\cdot02 \times \cdot06529}{5\cdot2}$, the result is the water-gauge = ·653, which is almost the same as before, proving that the pressure in pounds per square foot is the same whichever formula is used.

To find the horse power of ventilation then multiply the pressure per square foot (which may be ascertained by multiplying the water-gauge in inches by 5·2) by the cubic feet of air passing per minute, and divide by 33,000.

The ventilating pressure is chiefly required to overcome the resistance due to friction and obstruction, that required to put the air in motion being very slight. If that pressure be expressed as a head of air or motive column, the velocity due to the pressure will be equal to that which a body would acquire when it had fallen through a height equal to the head. A column of air 1 square foot in section and 13·09 feet high weighs 1 lb., and will therefore exert a pressure of 1 lb. to the square foot. This pressure produces a velocity in the air current equal to that which would be attained by a falling body through a height of 13·09 feet. This is usually expressed by the well-known formula for gravity, $V^2 = h\ 2\ g$, whence $V = \sqrt{h \times 2\ g}$ in which V is the velocity in feet a second, h, the height or space in feet fallen through, and g, the velocity in feet acquired by a falling body at the end of one second of time, the value of which is 32·2. Thus $V = \sqrt{h \times 64{\cdot}4} = 8{\cdot}02\sqrt{h}$, and therefore $h = \frac{V^2}{64{\cdot}4}$.

If the air current has a velocity of 4 feet per second, the head required to produce this velocity (omitting all consideration of friction) would be calculated thus, $h = \frac{4^2}{64{\cdot}4} = {\cdot}2484$ foot, or expressed in inches of water-gauge $\frac{{\cdot}2484}{68} =$ ·00365. Strictly speaking, this value should be added to that found for frictional and other resistances in mines, but when it is noted how very small is the pressure required to produce the velocity, and also the fact that the resistances are only estimated approximately, it is not surprising that the pressure to produce the velocity is in practice neglected, and the calculations rendered easier by such neglect.

The economy of a fan is often judged of by what is called its useful effect. This simply means the proportion that the power of ventilation bears to the horse-power exercised by the engine in driving the fan. Thus, suppose the horse-power of a fan-engine to be 100, and the horse-power of ventilation to be 50, we say that the useful effect of the fan is 50 per cent. Where it is desired to work out questions as to the useful effect of ventilating fans, it is necessary to be very exact in all data, or the results are misleading. Thus, the air must be very carefully and accurately measured at a point near the fan inlet, as explained under remarks on the anemometer. If it is desired to calculate the useful effect on the volume of intake air, a correction will have to be made for pressure and temperature, as the volume of air in the fan drift will be increased as compared with its state at the intake owing to different barometer and thermometer readings. That is, supposing in the ventilator drift the barometer reads 30 inches and the temperature shows 70° F., whilst the readings are 31 inches and 40° F. respectively at the intake airway. Then for every 1,000 cubic feet per minute in the intakes, the volume it would occupy in the ventilator drift will be found by the formula already given thus: $1{,}000 \times \frac{31 \times (70 + 460)}{30 \times (40 + 460)} = 1{,}095{\cdot}3$ cubic feet. Then, as 1,095·3 : 1,000 :: quantity in fan-drift : intake quantity.

Again, very accurate diagrams must be taken with the steam indicator (of which more will be said hereafter) on both sides of the piston, and these should be made simultaneously with the air measurements, and in calculating the horse-power of the engine allowance must be made for the area of the piston rod, whether on one or both sides of the piston, as the case may be. Any natural ventilation operating with or against the fan should be carefully ascertained and allowed for. As it is evident that *all* the horse-power of the engine is not used for driving the fan, but a part is required to overcome its own resistances, these should be ascertained by disconnecting the engine from the fan, and diagrams taken of it when running at the same speed as when working the fan. Suppose it be required to find the useful effect of a fan when there are 200,000 cubic feet of air passing per minute with 2 inches of water-gauge. The fan is worked by an engine having a 28-inch cylinder and $4\frac{1}{2}$-foot stroke, there being a piston rod of 4 inches' diameter on either side of the piston, the effective pressure of steam on the piston is 30 lbs., and it has a speed of 270 feet per minute resulting from 30 revolutions. The horse-power to work the engine without the fan has been ascertained to be 18.

The horse-power of the fan is $\frac{200{,}000 \times 5{\cdot}2 \times 2}{33{,}000} = 63$. That of the engine is $28^2 \times {\cdot}7854 = 615{\cdot}753$ area of piston; $4^2 \times {\cdot}7854 = 12{\cdot}566$ area of piston-rod; $615{\cdot}753 - 12{\cdot}566 =$ say 603 effective area of piston $\frac{603 \times 30 \times 270}{33{,}000} = 148$ horse-power of fan engine. As it requires 18 horse-power to work the engine itself, $148 - 18 = 130$ as the useful horse-power of the engine. Therefore, as $130 : 63 :: 100 : 48{\cdot}5$ per cent., which is the useful effect of the fan.

As to the amount of air exhausted by fans of given dimensions, it is almost impossible to say what that might be; very much would depend upon the condition and size of the airways, and these vary, so that the same sized fan of the same make gives different results at different collieries, sometimes being assisted by the natural ventilation and sometimes not.

A centrifugal fan, properly proportioned, and employed merely in displacing air, that is, under no drag, should deliver (at a velocity equal to the tips of its blades) a stream of air having a sectional area equal to the breadth of its blades at their outer ends, multiplied by the circumference of the circle described by those ends.

Also, the greatest water-gauge which any centrifugal fan can afford is dependent upon the speed at which the tips of the blades can safely be driven. Theoretically the depression of water-gauge, due to the velocity of the periphery of a perfect fan is equal to twice the height of column necessary to create such velocity in a falling body.

Take the case of a fan 24 feet in diameter, and allow it to run 64 revolutions per minute, it will be seen from the law $h = \frac{V^2}{64{\cdot}4}$ when $V =$ the speed of the tips of vanes in feet per second, that theoretically the greatest water-gauge it could afford would be $2{\cdot}954$ inches, thus $h = \frac{(24 \times 3{\cdot}1416 \times \frac{64}{60})^2}{64{\cdot}4} = 100{\cdot}43$; twice this head $= 200{\cdot}86$, and $\frac{200{\cdot}86}{68} = 2{\cdot}954$ inches of water-gauge, taking the atmospheric air at a temperature of 60° F. and at 30 inches barometric pressure.

The theoretical water-gauge was calculated on the old system by the formula $h = \frac{v^2}{2g}$, but M. Daniel Murgue's rule proceeds by the formula $h = \frac{v^2}{g}$ which gives exactly double the result. On the continent Murgue's rule is universal for the theoretical water-gauge due to the peripheral velocity, and the same standard should be adopted everywhere.

Experiments seem to show that with a Guibal fan a certain amount of benefit is derived from the shutter and chimney, and that the water-gauge as actually taken during those experiments gave a slight excess over the theoretical water-gauge.

made in the straight portion preceding the curve. Here the value of the co-efficient of the loss of pressure increased to 0·00062, so that the projection of the current of air against the concave side of the unbroken curve, had the effect of nearly doubling the friction in the straight gallery. Two other experiments were made in galleries, one of intermediate, and one of small area, which, although sinuous, had a general straight direction (see Fig. 422B). In the larger of these, the mean value of the co-efficient of friction was 0·00052 and in the smaller, which was only 22·9 square feet area, the co-efficient increased to 0·00055. Three experiments were made in timbered galleries, two of which were in normal airways, having the usual sinuosities of galleries driven in the seam and not set out by marks, one being an old gallery, and the other a newly-repaired one. The mean

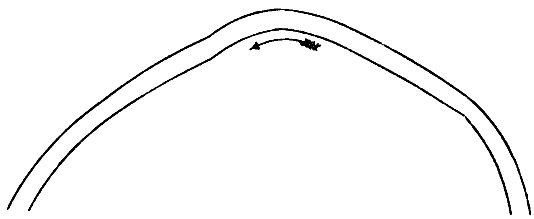


Fig. 422A.—Illustrating the Effect of a Curve in Increasing the Co-efficient of the Loss of Pressure.

value of the co-efficient of friction obtained was 0·00158. A final experiment was made in a gallery of small area, the curvatures in the airway being greater than in the preceding cases, and the surface irregularities between the props relatively of greater importance. In this instance the co-efficient of friction reached the highest point obtained, viz., 0·00241.

From these experiments it is plain that the values of the co-efficient of friction are not only different in different types of galleries, but are influenced by curves in the airways, the sectional area, and their inclination. In galleries of small area, for each of the three types, there is an increase in the co-efficient of friction.

It will be seen then, that there is difficulty in fixing a mean value for the co-efficient of friction for the whole of the galleries in a mine. In conducting

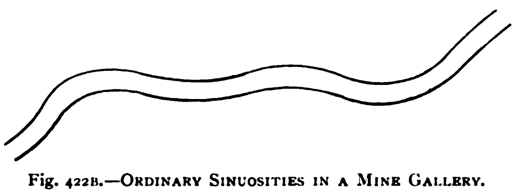


Fig. 422B.—Ordinary Sinuosities in a Mine Gallery.

any experiment upon the friction of air in mine galleries, there is great difficulty in obtaining an adequate length of gallery possessing a continuous regularity of area and uniformity of outline. Results obtained from experiments in short sections of airways, probably differ from those made in long ones, where the presence of refuge holes, stenton ends, and other openings interrupt the regularity of air-flow and increase the amount of resistance. However desirable it may be to have a correct co-efficient of friction for air-currents, it is not possible to fix one which can be applicable to all mines, but that of 0·00417 inch of water-gauge adopted by the late Mr. J. J. Atkinson, has hitherto been generally accepted. It has therefore been thought advisable to base all calculations which appear in this work, on these figures; if they are wrong, the error is on the safe side, and this is of importance when considering the friction of the mine, with a view to the erection of a ventilating fan.

THE MEASUREMENT OF VENTILATING PRESSURE.

If an ordinary water-gauge be fixed on a stand in a mine gallery (see Fig. 422c), having one limb connected by means of an open-ended pipe with the point A, and the other having fixed to it a longer pipe-connection also with an open end at

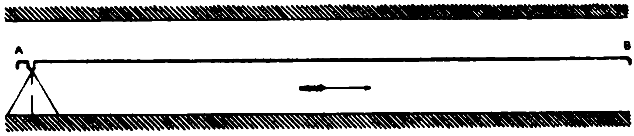


Fig. 422c.—Ascertaining the Difference of Pressures between the points A and B in an Underground Passage.

the point B, more or less distant from A, the stagnant air in the tubes will transmit to the water-gauge the pressures existing in the open ends at the two points. Whatever modification of effect is exercised on the pressure by one orifice is also exerted by the other, so that the difference of level of the water in the gauge, is a

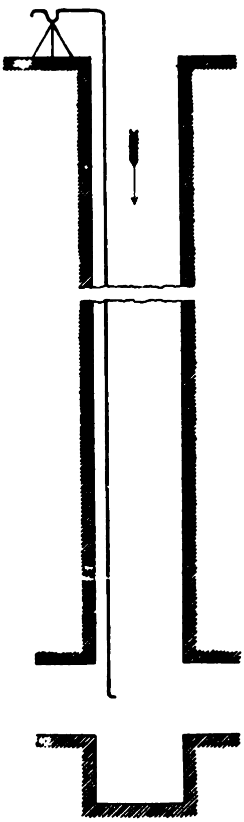


Fig. 422d.—Ascertaining the Difference of Pressures between the Top and Bottom of a Downcast Shaft.

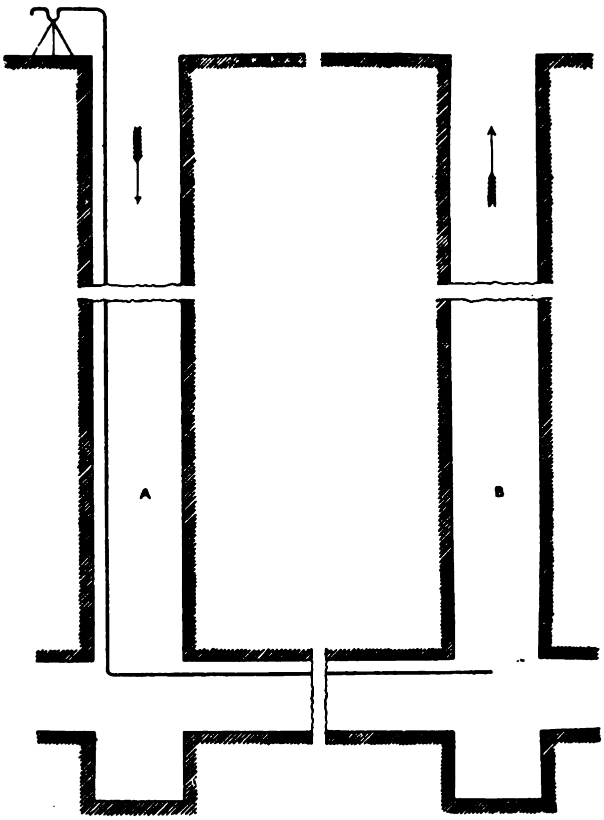


Fig. 422e.—Ascertaining the Difference of Pressures between the Top of a Downcast and the Bottom of an Upcast Shaft.

as shown in the sketch. A pressure of 10 lbs. at the downcast shaft becomes $10 - 3{\cdot}36 = 6{\cdot}64$ lbs. at the point A and midway is $\frac{10 + 6{\cdot}64}{2} = 8{\cdot}32$ lbs., as shown on the sketch. Any other points may be ascertained by proportion. The diminution of pressure in the airway, A B D, and in the airway A C D, being $3{\cdot}53$ lbs., if six equidistant points along their courses be chosen at $54\frac{1}{6}$-yard intervals, there will be a regular loss of $\frac{3{\cdot}53}{6} = {\cdot}59$ lb. nearly at each, and the pressures are as marked in the sketch for those points.

In the case of a mine with six splits of unequal length of the same perimeter and sectional area dividing at the downcast, and re-uniting in the upcast, and therefore subject to a common pressure, it will be found that the shortest split

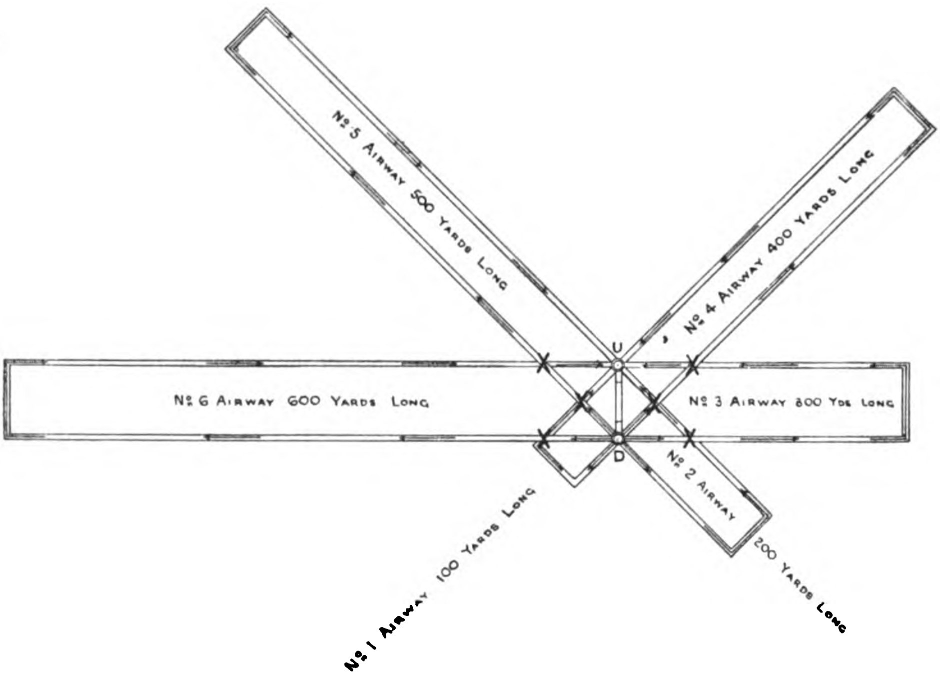


Fig. 422J.—Plan of Six Airways of Different Lengths.

gets the greatest and the longest the smallest proportion of the total quantity of air circulating. Assume these airways to be 6 feet square and of the following lengths:

No. 1 airway	....	100 yards.
2 ,,	....	200 ,,
3 ,,	....	300 ,,
4 ,,	....	400 ,,
5 ,,	....	500 ,,
6 ,,	....	600 ,,

as shown in Fig. 422J. The relative quantities in the airways will be the same as the relative velocities in them, because they are all of the same area. The velocities may be found by the formula $v = \sqrt{\frac{pa}{ks}}$. But since p and a and k

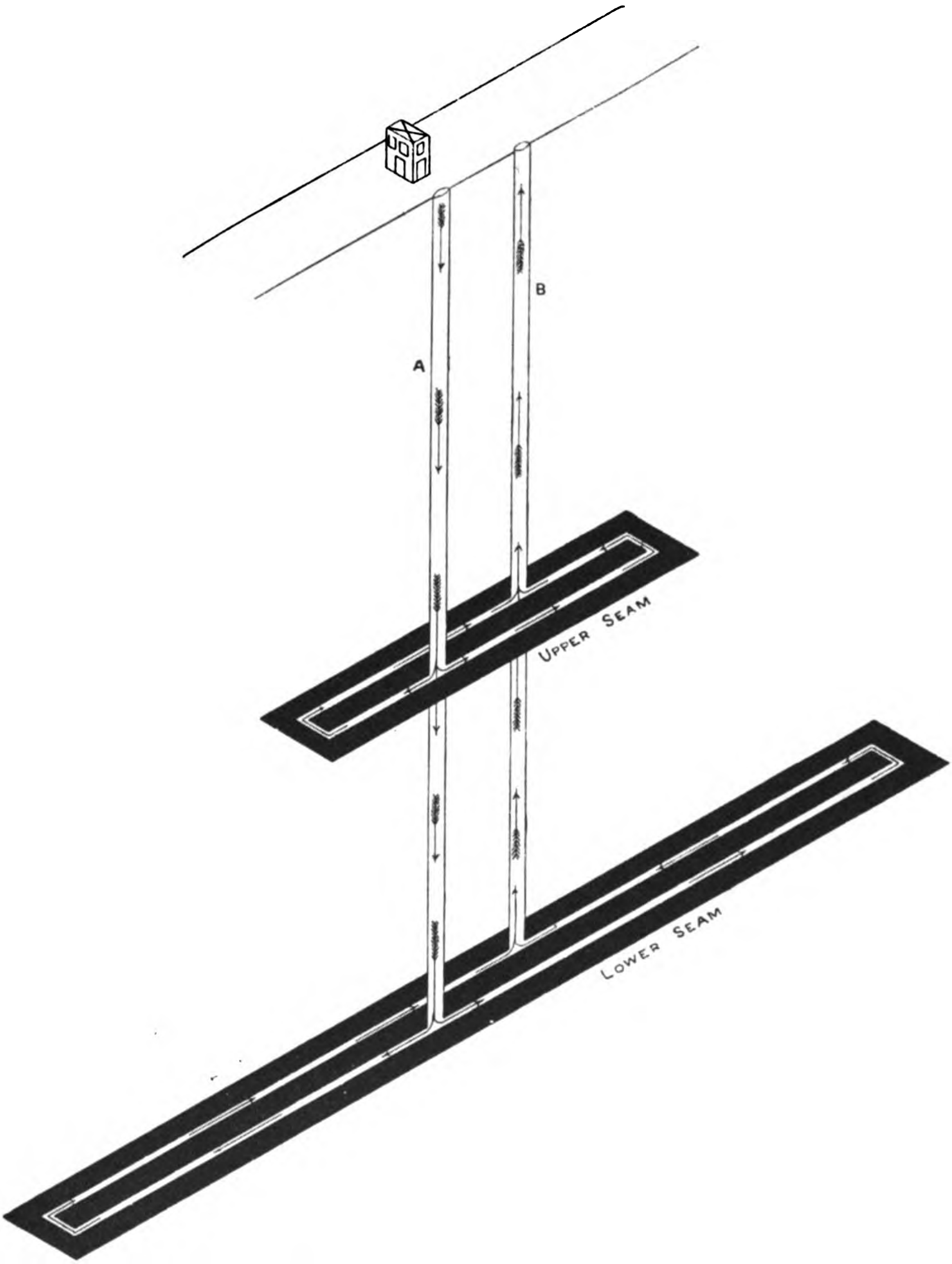


Fig. 422K.—ILLUSTRATING THE DIMINUTION OF PRESSURE IN THE AIR IN THE SHAFTS, AND THE EFFECT OF SPLITTING INTO AN UPPER AND A LOWER SEAM.

per second due to such a column is $8{\cdot}02\sqrt{159{\cdot}5} = 101{\cdot}28$, or per minute it is $101{\cdot}28 \times 60 = 6{,}077$. Now if the area through which air passes at this velocity is $25{\cdot}31$ square feet, then the total quantity passing would be $6{,}077 \times 25{\cdot}31$. Inasmuch, however, as it has been shown that only 65 per cent. of the whole area of the orifice is taken up by the flowing air, we have $\frac{25{\cdot}31 \times 65}{100} = 16{\cdot}452$ square feet as the area of that portion of the orifice through which air issues at the rate of 6,077 feet per minute; $6{,}077 \times 16{\cdot}452 = 99{,}979$ cubic feet per minute as against the 100,000 actually passing and from which the equivalent orifice, 25·31 square feet, is calculated.

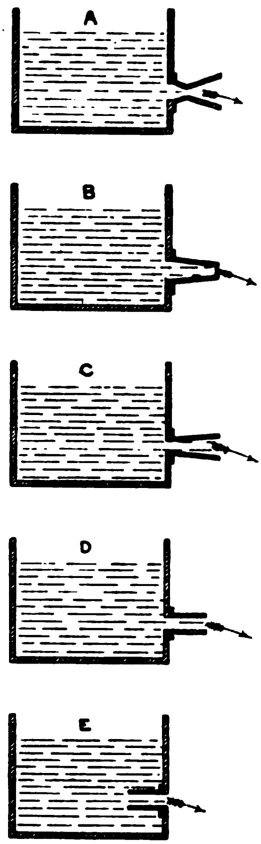


Fig. 422L.—Discharge of Water from Orifices.

It is customary to allow twice the inlet area as calculated from the rule for the equivalent orifice, because the inlet to a fan is not a hole in a thin plate, and considerable allowance is made for additional frictional resistances within the fan. The inlet area may thus be a single one of $2 \times 25{\cdot}31$ square feet area, or there may be a double inlet, one on each side of the fan, and each being 5·68 feet in diameter. Certain types of large fans run more evenly with a double inlet because a more regular supply is thus ensured. Assuming we have the double inlet, then to find the effective area, that is the area of the fan, 13·69 feet in diameter less the area of the inlet 5·68 in diameter or $147{\cdot}14 - 25{\cdot}31 =$ say 122 feet effective area. Each foot in width of this fan then will produce an effective volume of 122 cubic feet. The quantity to be discharged per minute is 100,000 cubic feet and the number of revolutions 100, $\frac{100{,}000}{100} =$ say 1,000 cubic feet per revolution $\frac{1{,}000}{122} =$ fully 8 feet. It is generally considered that a good fan will discharge one-half its effective capacity each revolution. Experiments have, however, proved that many good fans discharge much less, and this subject needs careful investigation. In practice fans are usually made having a certain ratio between their diameters and widths, and the result here obtained may be slightly modified. In the Waddle fan, which receives air only on one side, the diameter of the inlet is about one-third the diameter of the whole fan.

If the speed of the fan be increased from 100 to 150 revolutions per minute the water-gauge will be increased 2·25 times, and will then read $2{\cdot}35 \times 2{\cdot}25 = 5{\cdot}29$ inches, but the volume then circulating will be 150,000 cubic feet per minute.

As another instance to show how we may ascertain approximately the dimensions of a large ventilating fan, let us assume that the volume of air to be circulated is 300,000 cubic feet per minute with a water-gauge of 4·5 inches.

For a water-gauge of 4·5 inches we should have a ventilating column of

$$\frac{815 \times 4{\cdot}5}{12} = 305{\cdot}62 \text{ feet.}$$

The tangential velocity of the fan due to this water-gauge is $v = \sqrt{305{\cdot}62 \times 32{\cdot}2} = 99{\cdot}2$ feet per second, or $99{\cdot}2 \times 60 = 5{,}952$ feet per minute.

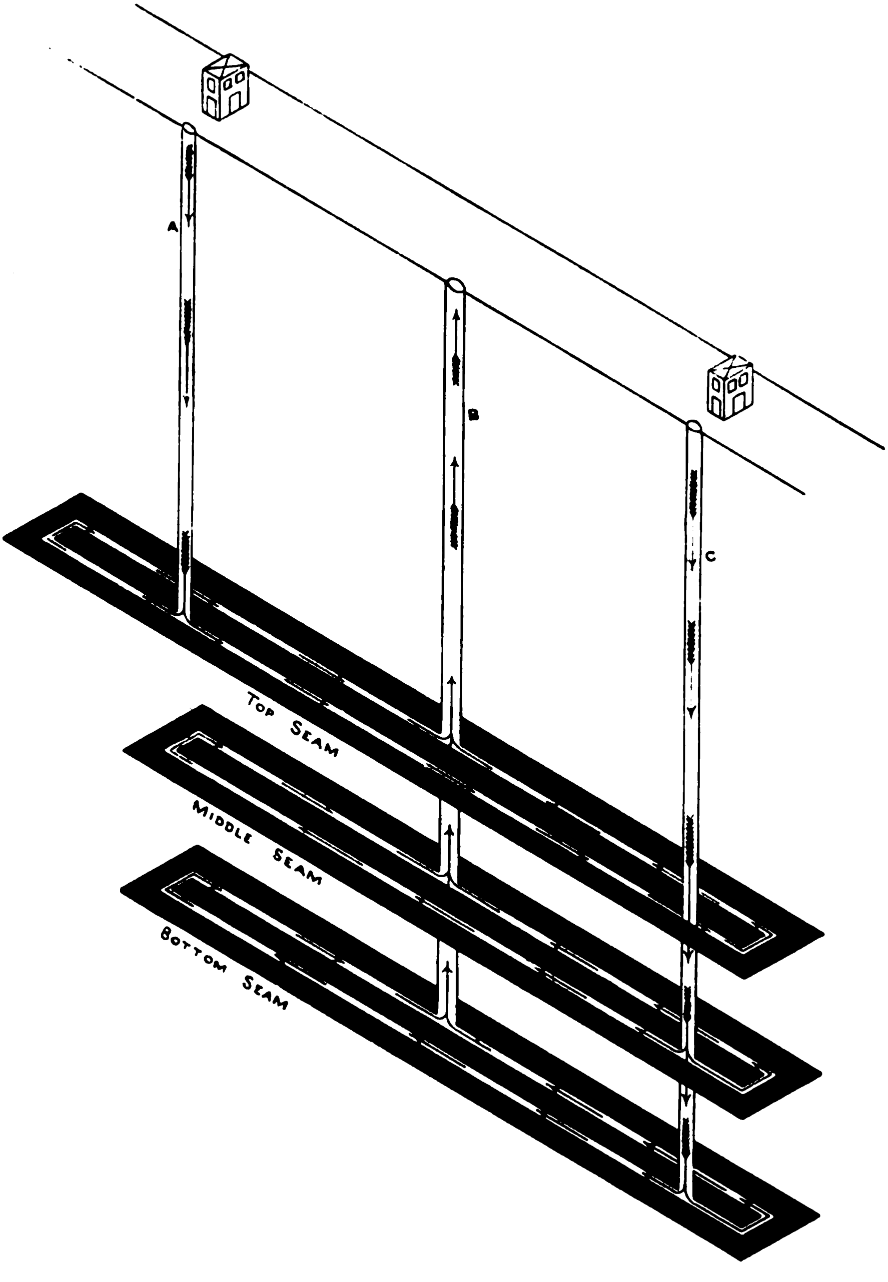


Fig. 422M.—Illustrating the Diminution of Pressure in the Air in the Shafts and the Effect of Splitting into Different Seams.

seams, the shaft being 47·124 feet in circumference and 176·715 square feet area, is

$$p = \frac{(\cdot 00000001) \times (47\cdot 124 \times 100 \times 3) \times (43{,}508)^2}{176\cdot 715^3} = \cdot 048493 \text{ lb.}$$

The area of a 15-foot diameter shaft being 176·715 square feet, and that of a 20-foot diameter shaft being 314·16, the relative velocities of the air in the two shafts between the middle and bottom seams is as 176·715 : 314·16 or as 1 : 1·778, so that if the velocity of the air in the 20-foot shaft be 1, that in the 15-foot shaft will be 1·778. Since $p = \frac{ksv^2}{a}$ and k is common to both shafts, the relative pressures are as $\frac{sv^2}{a}$, or as $\frac{47\cdot 124 \times 1\cdot 778^2}{176\cdot 715} : \frac{62\cdot 832 \times 1^2}{314\cdot 16}$ = as ·8428 : ·2 or as 1 : ·2373. The actual pressure for the 15-foot shaft has been found to be ·048493 lb., and therefore that for the 20-foot shaft is as 1 : ·2373 :: ·048493 : ·011507 lb. The pressure absorbed in overcoming the frictional resistances of shafts B and C between the bottom and middle seams is

$$\begin{array}{l} \cdot 048493 \\ \cdot 011507 \\ \hline \cdot 06 \quad \text{lb. per square foot.} \end{array}$$

The pressure then influencing the flow of air in the middle seam is this ·06 lb. absorbed in the shafts plus that in overcoming the bottom seam or 1·778 lb.

$$\begin{array}{rl} \text{Thus} & \cdot 06 \\ & 1\cdot 778 \\ \hline & 1\cdot 838 \end{array}$$

lb. per square foot as the pressure in the middle seam splits, and we have to ascertain the air currents such pressure would be capable of producing there. We may now apply the formula $v = \sqrt{\frac{pa}{ks}}$ to ascertain the velocities of the air in the two middle seam splits, thus:—

$$\text{For the longer } v = \sqrt{\frac{1\cdot 838 \times 49}{(\cdot 0000000217) \times (7 \times 4 \times 300 \times 3)}} = 405\cdot 82 \text{ feet per min.}$$

$$\text{and for the shorter } v = \sqrt{\frac{1\cdot 838 \times 49}{(\cdot 0000000217) \times (7 \times 4 \times 200 \times 3)}} = 497\cdot 03 \text{ or as } \sqrt{200} :$$

$\sqrt{300}$ :: 405·82 : 497·03 feet per minute, and the respective volumes of air flowing are 405·82 × 49 = 19,885, and 497·03 × 49 = 24,355 cubic feet per minute. The air flowing in the middle seam then is

In the longer split	19,885	cubic feet per minute.
„ „ shorter „	24,355	„ „ „ „
Total	44,240	„ „ „ „

This result may be tested by the relative or proportional quantities for the two splits in the middle seam. The proportion is the same as in the case of the two splits in the bottom seam.

Thus for the 300-yard airway as 44·057 : 19·803 :: 44,240 : 19,885 cubic feet per minute, and for the 200-yard airway as 44·057 : 24·254 :: 44,240 : 24,355 cubic feet per minute.

The total volume of air then entering shaft C, and also passing up shaft B from

the middle to the level of the top seam, is 43,508 cubic feet in the bottom seam,
and 44,240 „ „ „ middle „

Making together 87,748 „ „ per minute.

The pressure necessary to overcome the frictional resistances in shaft C from the surface to the middle seam is

$$p = \frac{(\cdot 00000001) \times (47\cdot 124 \times 400 \times 3) \times (87{,}748)^2}{176\cdot 715^3} = \cdot 789 \text{ lb.},$$

and for the shaft B between the middle and top seams

$$p = \frac{(\cdot 00000001) \times (62\cdot 832 \times 100 \times 3) \times (87{,}748)^2}{314\cdot 16^3} = \cdot 0468 \text{ lb.}$$

The sum of the frictional resistances so far ascertained is as follows:—

	per sq. foot.
Shaft C, from the surface to the middle seam	·789 lb.
„ B, „ the middle to the top seam	·0468 „
For overcoming shaft friction between bottom and middle seams and that of the different splits	1·838 „
	2·6738 lbs.

or a total of 2·6738 lbs. per square foot to the junction of the top seam air coming from shaft A, and which is influenced by this pressure.

Now a difficulty arises in determining the quantity of air which enters shaft A, because we do not know the proportion of the frictional resistances in the top seam splits and those in the shaft itself, the total of which amount to 2·6738 lbs. per square foot.

The relative quantities, however, which would pass into the top seam splits, may be found thus:—

$$R = \sqrt{\frac{a^3}{s}} \text{ or } R = \sqrt{\frac{a^3}{l}} \text{ as the airways are all 7 feet square,}$$

$$\therefore R = \sqrt{\frac{49^3}{400}} = 17\cdot 15 \text{ for the longer,}$$

$$\text{and } R = \sqrt{\frac{49^3}{300}} = 19\cdot 803 \text{ for the shorter split.}$$

$$36\cdot 953$$

Since the areas of the two airways are the same, the relative velocities are as $17\cdot 15$ is to $19\cdot 803$, or as $1 : \frac{19\cdot 803}{17\cdot 15} = 1\cdot 1547$. If the velocity in the longer split is 1, then that in the shorter is 1·1547. The quantity of air passing then is $(1 \times 49) + (1\cdot 1547 \times 49) = 49 + 56\cdot 58 = 105\cdot 58$. If v = the velocity of the air in shaft A, then $176\cdot 715\ v = 105\cdot 58$, and $v = \frac{105\cdot 58}{176\cdot 715} = \cdot 5975$. We have thus as the relative velocities in shaft A and the two top seam splits ·5975, 1, and 1·1547.

The relative pressures necessary for the shaft A and for the two splits in the top seam may now be found

For the shaft, $p = \frac{(\cdot 00000001) \times (47\cdot 124 \times 300 \times 3) \times (\cdot 5975)^2}{176\cdot 715} = \cdot 0000008567$ lb.

$$\text{then } p = \frac{(\cdot 00000001) \times (62\cdot 832 \times 300 \times 3) \times (131{,}269)^2}{314\cdot 16^3} = \cdot 31426 \text{ lb.}$$

The pressure necessary to overcome the whole of the mine and shaft resistances then is—

	2·6738 lbs.
and	·31426
	2·98806 lbs. per square foot, and $\frac{2\cdot 98806}{5\cdot 2} = \cdot 5746$ inch of water-gauge.

It would appear thus that under the conditions given the air would flow into each seam in nearly equal volumes. The shaft resistances between the two bottom seams are slight, and the airways being of similar lengths only a little less air would pass into the bottom seam than into the middle seam, the difference being a measure of the shaft resistances. Then in shaft A, the lessened shaft resistance compared with shaft C is near about balanced by the increased length of the top seam airways, which thus between them get nearly the same quantity of air as is circulating in the bottom seam.

Airways and shafts of large sectional area greatly assist the ventilation of the mine, as may be strikingly shown in the last example, if we take the diameter of shaft A to be only 10 feet in diameter and the top seam airways to be only 5 feet by 5.

The circumference of a 10-foot shaft is 31·416 feet and its area is 78·54 square feet.

The pressure influencing the circulation of the air in the top seam would be 2·6738 lbs. as before, but a great falling off in the quantity follows.

Thus the relative quantities which under the new conditions would flow in the top seam splits are:—

$$R = \sqrt{\frac{25^3}{400}} = 6\cdot 25 \text{ for the longer}$$

$$\text{and } R = \sqrt{\frac{25^3}{300}} = 7\cdot 2168 \text{ for the shorter.}$$

Since the airways have equal areas the relative velocities of air in them are as $6\cdot 25 : 7\cdot 2168$, or as $1 : \frac{7\cdot 2168}{6\cdot 25} = 1\cdot 1547$, being the same as those found for 7-foot square airways. The quantity of air passing then is $(1 \times 25) + (1\cdot 1547 \times 25) = 25 + 28\cdot 867 = 53\cdot 867$. If $v =$ the velocity of the air in shaft A, then $78\cdot 54\ v = 53\cdot 867$ and $v = \frac{53\cdot 867}{78\cdot 54} = \cdot 68585$. The relative velocities in shaft A and the two top seam splits are ·68585, 1, and 1·1547.

The relative pressures then must be—

For the shaft—

$$p = \frac{(\cdot 00000001) \times (31\cdot 416 \times 300 \times 3) \times (\cdot 68585)^2}{78\cdot 54} = \cdot 000016934 \text{ lb.}$$

For the longer airway—

$$p = \frac{(\cdot 0000000217) \times (5 \times 4 \times 400 \times 3) \times (1)^2}{25} = \cdot 000020832 \text{ lb.}$$

Or tested for the shorter—

$$p = \frac{(\cdot 0000000217) \times (5 \times 4 \times 300 \times 3) \times (1\cdot 1547)^2}{25} = \cdot 000020832 \text{ lb.}$$

a single 20-foot downcast compared with two 15-foot downcasts. It may be of interest to see how the total quantity of 137,633 cubic feet of air per minute would be divided in the different seams.

The increased pressure

In the bottom seam would be, As 2·463274 : 2·98806 :: 1·778 : 2·1568 lbs.
" " middle " " " " 2·463274 : 2·98806 :: 1·801014 : 2·1847 lbs.
" " top " " " " 2·463274 : 2·98806 :: 1·893674 : 2·2971 lbs.

and the consequent increased volumes of air in these seams are

In the bottom seam, As $\sqrt{1·778}$: $\sqrt{2·1568}$ :: 43,508 : 47,919
" " middle " " $\sqrt{1·801014}$: $\sqrt{2·1847}$ :: 43,792 : 48,232
" " top " " $\sqrt{1·893674}$: $\sqrt{2·2971}$ :: 37,664 : 41,482

Total 137,633 cub. ft. per min.

or the increased volumes may be more simply obtained by proportion, thus:—

In the bottom seam, As 124,964 : 137,623 :: 43,508 : 47,919
" " middle " " 124,964 : 137,623 :: 43,792 : 48,232
" " top " " 124,964 : 137,623 :: 37,664 : 41,482

Total 137,633 cub. ft. per min.

We may now show the result of the two calculations side by side for comparison.

Name of Seam.	With two 15-foot Downcasts, one 20-foot Upcast, and water-gauge of the whole mine and shafts, ·5746 inch.		With an Upcast and Downcast each 20 feet in diameter, airways and total water-gauge being the same as in the case of the three shafts, viz., ·5746 inch.	
	Pressure in lbs. per square foot.	Quantity af Air in cubic feet per minute.	Pressure in lbs. per square foot.	Quantity of Air in cubic feet per minute.
Top seam	2·6738	43,521	2·2971	41,482
Middle "	1·838	44,240	2·1847	48,232
Bottom "	1·778	43,508	2·1568	47,919
Total.........	2·98806	131,269	2·98806	137,633

It will thus be seen that so far as the total ventilation of the mine is concerned there would be an advantage under the given conditions, of having a single downcast 20 feet in diameter as compared with two such shafts each 15 feet in diameter. With the same total pressure, however, there would be a falling off in the quantity of air circulating in the top seam, if no regulators were placed, and larger volumes in the other seams. This is due to the re-distribution of pressures which must ensue if any such change in the shafts be carried out.

Instead of a colliery consisting of two downcast shafts and one upcast as shown in Fig. 422M, there may be two or three seams being worked with a single downcast and two upcast shafts. Fig. 422N shows such a colliery. In addition to these shafts there is a staple or blind pit used for ventilation between the No. 1 and No. 2 seams and marked with the letter D on the plan.

Supposing the downcast shaft A to be 20 feet in diameter and each upcast 15 feet. Shaft C is only sunk 300 yards to the No. 1 seam, while shafts B and A

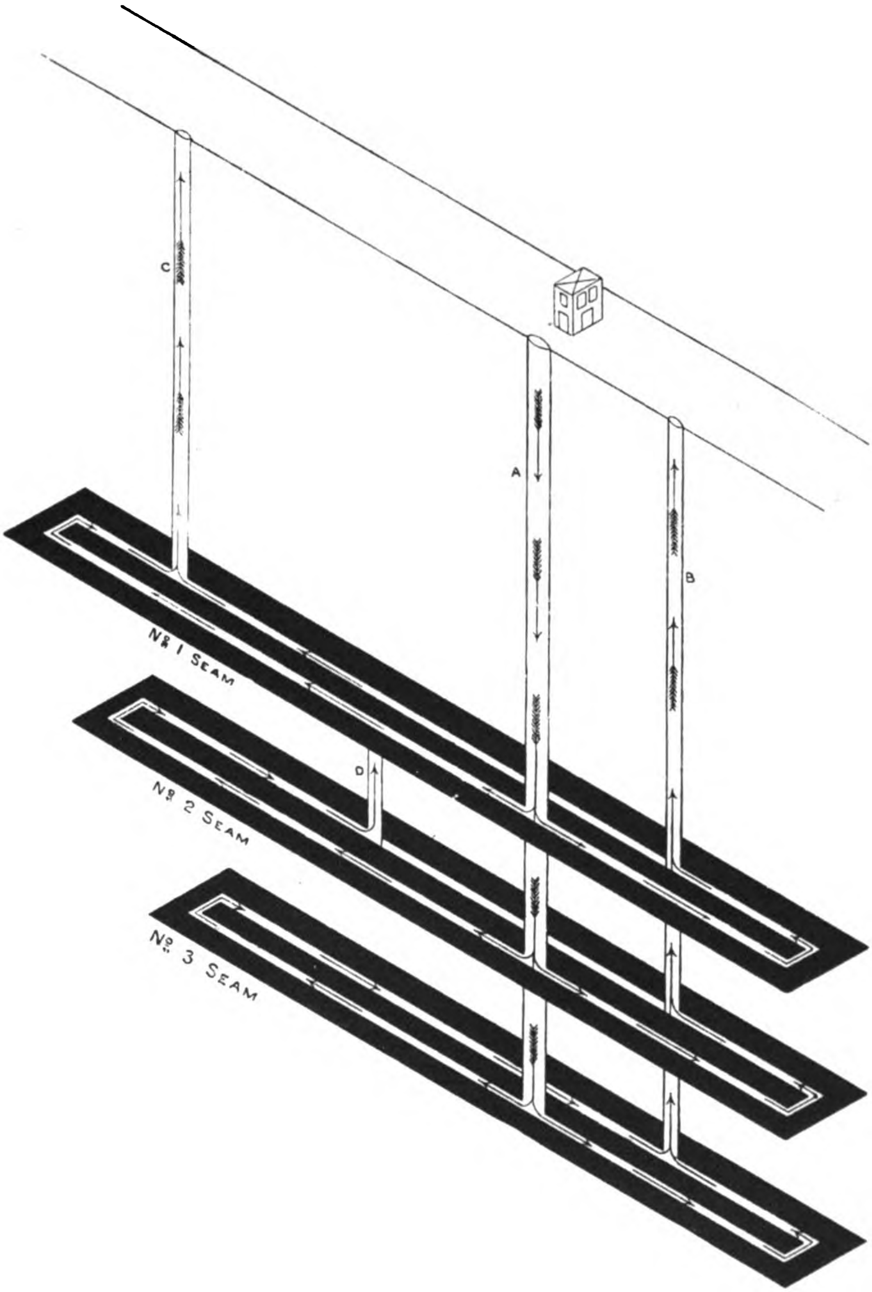


Fig. 422N.—Illustrating the Diminution of Pressure in the Air in the Shafts and the effect of Splitting into different Seams.

The pressure in shaft A, between No. 1 and No. 2 seams, was shown to be ·021777 lb., the proportion for the west split in No. 2 seam being ·0054307 lb., that for the east split is

$$\begin{array}{r} \cdot 021777 \\ \text{less } \cdot 0054307 \\ \hline \cdot 0163463 \end{array} \text{ lb. per square foot.}$$

The pressure affecting the flow of air in the east split of No. 1 seam is

$$\begin{array}{r} \cdot 0163463 \\ \cdot 0517 \\ 5 \cdot 221728 \\ \hline 5 \cdot 2897743 \end{array} \text{ lbs. per square foot,}$$

and the velocity of the air in this airway would be

$$v = \sqrt{\frac{5 \cdot 2898 \times 36}{(\cdot 0000000217) \times (6 \times 4 \times 800 \times 3)}} = 390 \cdot 33 \text{ feet per minute,}$$

and the quantity circulating in it is $390 \cdot 33 \times 36 = 14,052$ cubic feet per minute.

The total volume of air in shaft B is

$$\begin{array}{r} 44,925 \\ 14,052 \\ \hline 58,977 \end{array}$$ cubic feet per minute passing up from the No. 1 seam to the surface.

The value of p for this portion of the shaft is

$$\frac{(\cdot 00000001) \times (47 \cdot 124 \times 300 \times 3) \times (58,977)^2}{176 \cdot 715^3} = \cdot 26732 \text{ lb.}$$

The whole volume of air entering shaft A is the sum of the two volumes discharged from shafts B and C, viz., in shaft B 58,977
„ „ C 29,228

88,205 cubic feet per minute,

and the pressure necessary to overcome the friction in shaft A for such quantity from the surface to No. 1 seam is

$$p = \frac{(\cdot 00000001) \times (62 \cdot 832 \times 300 \times 3) \times (88,205)^2}{314 \cdot 16^3} = \cdot 14189 \text{ lb.}$$

The proportion of this pressure due to the ventilator at shaft C is

As 88,205 : 29,228 :: ·14189 : ·04702 lb.,
and to that at shaft B, „ 88,205 : 58,977 :: ·14189 : ·09487 lb.

The total pressure exerted by the ventilating power at shaft C then is

$$\begin{array}{r} 6 \cdot 85128 \\ \cdot 06565 \\ \cdot 04702 \\ \hline 6 \cdot 96395 \end{array} \text{ lbs. per square foot,}$$

causing a circulation of 14,925 cubic feet per minute through the west split of No. 2 seam, and 14,303 cubic feet through the west split of No. 1 seam, with a total volume of 29,228 cubic feet in the shaft.

The total pressure exerted by the ventilating power at shaft B is

$$\begin{array}{r} 5 \cdot 28977 \\ \cdot 26732 \\ \cdot 09487 \\ \hline 5 \cdot 65196 \end{array} \text{ lbs. per square foot,}$$

producing a circulation of 30,000 cubic feet per minute in No. 3 seam, 14,925 cubic feet through the east split of No. 2 seam, and 14,052 cubic feet through the east split of No. 1 seam, with a total volume of 58,977 cubic feet per minute in the shaft.

It will be noticed that although the pressure for the workings ventilated to shaft B, inclusive of shaft friction, is less than that required for the workings ventilated to shaft C, inclusive of shaft friction, it produces a greater volume of air. This is partly due to the greater number of splits, and partly to their being shorter on the east side of shaft A. Splitting the air into different seams from the downcast shaft has the same beneficial effect on the ventilation of a mine as splitting into airways of one seam direct from the downcast shaft.

We may now increase the ventilating power in shaft B so as to make the ventilating pressure correspond with that in shaft C, that is, raise it from 5·65196 lbs. to 6·96395 lbs. By so doing we should increase the quantity of air in shaft B

As $\sqrt{5 \cdot 65196} : \sqrt{6 \cdot 96395} :: 58{,}977 : 65{,}465$ cubic feet per minute.

The air in No. 3 seam and in the east splits of the other seams would then be increased in proportion to the total increase thus :—

In No. 3 seam, As 58,977 : 65,465 :: 30,000 : 33,300 cubic feet per minute.
" " 2 " " 58,977 : 65,465 :: 14,925 : 16,567 " " "
" " 1 " " 58,977 : 65,465 :: 14,052 : 15,598 " " "

In No. 3 seam the new volume would be proportioned between the two splits thus :—

In the east split, As 16·455 : 8·8182 :: 33,300 : 17,845
" " west " " 16·455 : 7·6368 :: 33,300 : 15,455

33,300 cubic feet per minute.

The increased pressure here would be

For the east split, As $16{,}077^2 : 17{,}846^2 :: 5 \cdot 1932 : 6 \cdot 3993$ lbs,
or tested for the west " " $13{,}923^2 : 15{,}455^2 :: 5 \cdot 1932 : 6 \cdot 3993$ lbs.

In the east split of the No. 2 seam it would be

As $14{,}925^2 : 16{,}567^2 :: 5 \cdot 2217 : 6 \cdot 433$ lbs.,

and in the east split of No. 1 seam

As $14{,}052^2 : 15{,}598^2 :: 5 \cdot 2898 : 6 \cdot 5177$ lbs.

The ventilation of the colliery as a whole has now been increased thus :—

	At first. Cubic feet per minute.		Afterwards. Cubic feet per minute.
Discharged from shaft B	58,977		65,465
" " " C	29,228		29,228
Total	88,205	Total	94,693

and with the higher quantity the pressure is 6·96395 lbs. per square foot as measured at the top of each upcast shaft, and inclusive of shaft friction.

Let us now assume that it is considered desirable to do away with shaft C entirely, to enlarge the shaft B to 20 feet in diameter, make a connection with it for the return airway of the west split in the No. 1 seam, and another for that of the west split in No. 2 seam, thus dispensing with the further use of the staple D. For simplicity of comparison let us assume that all airways remain of their former length, that of the west split in No. 2 seam being 900 yards.

If now of the air entering shaft A, say 30,000 cubic feet per minute, ventilate the No. 3 seam, we shall have in the east split 16,077
" " west " 13,923

30,000 cubic feet,

being the same before and after altering the shafts.

Therefore the quantity passing in the west split of No. 1 seam is

As 7·6367 : 6·8305 :: 13,992 : 12,515 cubic feet per minute,

and the entire ventilation of this seam amounts to—

In the east split	13,992
„ west „	12,515
	26,507 cubic feet per minute.

The whole volume of air in the mine then is

In No.	1 seam	26,507	cubic feet per minute.
„	2 „	28,040	„ „ „ „
„	3 „	30,000	„ „ „ „
	Total	84,547	„ „ „ „

The pressure for each of the shafts A and B between the surface and No. 1 seam is

$$p = \frac{(\cdot 00000001) \times (62\cdot 832 \times 300 \times 3) \times (84{,}547)^2}{314\cdot 16^3} = \cdot 13037 \text{ lb.}$$

or as $88{,}205^2 : 84{,}547^2 :: \cdot 14189 : \cdot 13037$ lb.

The total pressure of the mine then is

5·245102
·13037
·13037
5·505842 lbs. per square foot for the total quantity of 84,547 cubic feet per minute.

But in order to compare the efficiency of the ventilation produced at the two shafts with that existing when the three were in operation we must raise the ventilating pressure to a similar extent. It was shown that with a ventilating pressure of 6·96395 lbs. per square foot measured at the top of upcast shaft C and at top of upcast shaft B with three shafts operative, 94,693 cubic feet per minute were produced. If with only two 20-foot diameter shafts working we raise the pressure from 5·50584 to 6·96395 lbs. we have

As $\sqrt{5\cdot 50584} : \sqrt{6\cdot 96395} :: 84{,}547 : 95{,}085$ cubic feet per minute.

The increased quantity of air in each seam is proportional to the total increase and would be

		Cubic feet per minute.
For No. 1 seam.	As 84,547 : 95,085 :: 26,507 :	29,811
„ „ 2 „	„ 84,547 : 95,085 :: 28,040 :	31,535
„ „ 3 „	„ 84,547 : 95,085 :: 30,000 :	33,739
		95,085

The proportion for each split in the different seams is—

In No. 1 seam.

For the east split.	As 14·4672 : 7·6367 :: 29,811 :	15,736
„ „ west „	„ 14·4672 : 6·8305 :: 29,811 :	14,075
		29,811

In No. 2 seam.

For the east split.	As 15·364 : 8·164 :: 31,535 :	16,757
„ „ west „	„ 15·364 : 7·2 :: 31,535 :	14,778
		31,535

	In No. 3 seam.	Cubic feet per minute.
For the east split.	As 16·455 : 8·8182 :: 33,739 :	18,080
„ „ west „	„ 16·455 : 7·6368 :: 33,739 :	15,659
		33,739

We may compare this result with the quantities circulating when the three shafts were in operation with an equal ventilating pressure of 6·96395 lbs. Thus—

Division of the Mine.		With downcast A 20 feet in diameter and upcasts B and C each 15 feet in diameter. Cubic feet of air per minute.	With downcast A and upcast B each 20 feet in diameter. Cubic feet of air per minute.
No. 1 seam	East split	15,598	15,736
	West „	14,303	14,075
No. 2 seam	East split	16,567	16,757
	West „	14,925	14,778
No. 3 seam	East split	17,845	18,080
	West „	15,455	15,659
Total		94,693	95,085

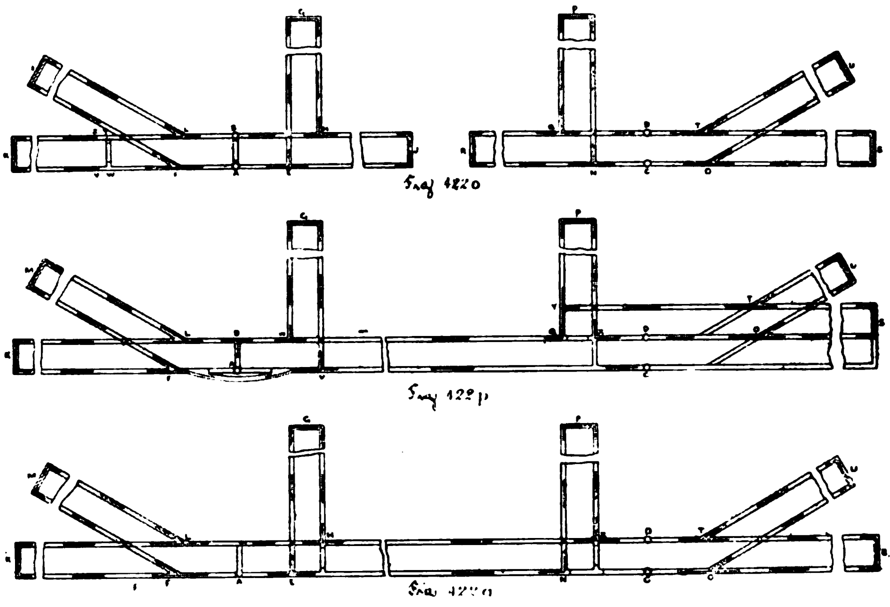
There would thus be a slight increase in the total ventilation of the mine under the given conditions by having a single upcast of 20 feet in diameter as compared with two such shafts each 15 feet in diameter. With the same total ventilating pressure, however, the quantities in the west splits of the two upper seams would be slightly reduced if no regulators were used. In the other splits an increased quantity would circulate, the change in the arrangements of shafts causing a redistribution of the air.

It has already been shown that a single downcast shaft 20 feet in diameter under the stated conditions gives a greater ventilation from the same pressure than two such shafts each of which is 15 feet in diameter, and the benefit in that case was shown to be even greater than in this where one upcast shaft 20 feet in diameter is designed to supplant two such shafts, each being 15 feet in diameter.

Suppose a large mining property to be worked by means of four shafts, as shown in Fig. 422O. Only one seam of coal is being worked, all the shafts being sunk to it. A is a downcast and coal winding shaft 700 yards in depth and 18 feet in diameter in the clear. B, the upcast for A, is 12 feet in diameter inside the walling and 690 yards in depth. The ventilation is produced by means of a furnace at B, which is not fed by a scale of fresh air from shaft A. C is a downcast and drawing shaft 680 yards in depth and 16 feet in diameter, finished. D is a fan upcast for C, 670 yards in depth and 18 feet inside diameter. The fan at the top of D ventilates the portion of workings connected to the downcast C, and the furnace at the bottom of B ventilates a separate portion of workings connected to downcast A. After a time these different portions of workings become connected as shown in Fig. 422P, and it is then decided to abandon shaft B, and either to place a fan at the top of shaft A, making C and D downcast as shown in Fig. 422P, or allow D to remain as the fan shaft and make A and C downcast as shown in Fig. 422Q.

Let us now consider the effect on the water-gauge if these alterations are carried out for the same total volume of air circulating in the mine, there being no regulators or means of checking the flow in any of the airways.

First, the workings connected with shafts A and B, Fig. 422O. It will be noticed that the air divides at the bottom of shaft A, one portion going in the direction A E 50 yards, when it is again split, and the other to A F 60 yards to its point of splitting. The split E G H is 400 yards in length, and at H this current re-unites in the returns with that of the split E J H, which is 300 yards in length to



Figs. 422O, 422P, 422Q.—Illustrating the effect on the Water-Gauge of making certain Changes in the Mine.

the point H. From H to the upcast shaft B the distance is 80 yards, and this airway receives the return air from the two splits E G H and E J H. The split F K L is 200 yards in length, and at L re-joins the current of air from split F M L, which is 300 yards in length. The distance from L to the upcast shaft B is 60 yards. The sectional area of the airways is greater in the intakes and returns A E, A F, H B and L B than in the splits, but we will take the average dimensions of all airways as being 7 feet by 7.

Of the air entering shaft A, supposing 20,000 cubic feet per minute flow through the airway A E, then the relative quantities in the two splits beyond are

$$\text{For } E\,G\,H = \sqrt{\frac{49^3}{400}} = 17{\cdot}15$$

$$\text{,, } E\,J\,H = \sqrt{\frac{49^3}{300}} = 19{\cdot}803$$

$$36{\cdot}953$$

and the actual quantities are

For E G H, As $36{\cdot}953 : 17{\cdot}15 :: 20{,}000 : 9{,}282$

„ E J H, „ $36{\cdot}953 : 19{\cdot}803 :: 20{,}000 : 10{,}718$

$20{,}000$ cubic feet per minute.

The pressure necessary to circulate the quantity in either split may now be found thus

$$\text{For E G H, } p = \frac{({\cdot}0000000217) \times (400 \times 3 \times 7 \times 4) \times (9{,}282)^2}{49^3} = {\cdot}53394 \text{ lb.}$$

$$\text{or for E J H, } p = \frac{({\cdot}0000000217) \times (300 \times 3 \times 7 \times 4) \times (10{,}718)^2}{49^3} = {\cdot}53394 \text{ lb.}$$

The pressure necessary to circulate 20,000 cubic feet of air per minute through the intake AE and the return BH, or a total length of 50 + 80 = 130 yards is

$$p = \frac{({\cdot}0000000217) \times (130 \times 3 \times 7 \times 4) \times (20{,}000)^2}{49^3} = {\cdot}80567 \text{ lb.}$$

Apart from shaft friction the pressure for overcoming the mine resistances then is

$$\begin{array}{r} {\cdot}53394 \\ {\cdot}80567 \\ \hline 1{\cdot}33961 \end{array} \text{ lbs. per square foot.}$$

We have now to ascertain what quantities of air such pressure will produce in the splits F K L and F M L. Since the areas of the two airways are the same, the relative velocities are :—

$$\text{For F K L} = \sqrt{\frac{49^3}{200}} = 24{\cdot}254$$

$$\text{„ F M L} = \sqrt{\frac{49^3}{300}} = 19{\cdot}803$$

$$\overline{44{\cdot}057}$$

If the velocity in the longer split is 1, then that in the shorter is $\frac{24{\cdot}254}{19{\cdot}803}$ or $1{\cdot}2247$, and the velocity in the intake A F and in the return L B must be $1 + 1{\cdot}2247 = 2{\cdot}2247$. The quantity of air passing then must be $2{\cdot}2247 \times 49 = 109$. The relative pressure in overcoming the resistances in each split is the same, viz.:

$$\text{For F K L} = \frac{({\cdot}0000000217) \times (200 \times 3 \times 7 \times 4) \times (1{\cdot}2247)^2}{49} = {\cdot}00001116 \text{ lb.}$$

$$\text{„ F M L} = \frac{({\cdot}0000000217) \times (300 \times 3 \times 7 \times 4) \times (1)^2}{49} = {\cdot}00001116 \text{ lb.}$$

and that for the intake A F and return L B measuring together 60 + 60 = 120 yards.

$$p = \frac{{\cdot}0000000217 \times (120 \times 3 \times 7 \times 4) \times (2{\cdot}2247)^2}{49} = {\cdot}00002209 \text{ lb.}$$

The total relative pressure then is

For the splits	·00001116
For the main intake and return	·00002209
Total ...	·00003325 lb.

But the actual loss of pressure influencing the air from shaft A to shaft B through splits F K L and F M L has been shown to be 1·33961 lb. Since p varies in accordance with q^2 we get actually in the airways A F and L B.

As $\sqrt{\cdot 00003325} : \sqrt{1\cdot 33961} :: 109 : 21,88$: cubic feet per minute.

The proportion of this volume in the two splits is

For F K L, A 7 : 24·254 :: 21,881 : 12,046
,, F M L, ,, : 19·803 :: 21,881 : 9,835

21,881 cubic feet per minute.

The accuracy of these figures may be tested by working out the value of p for either of the splits with the proportion of air now given for it and for the airways A F and L B thus—

For the split F M L, $p = \dfrac{(\cdot 0000000217) \times (300 \times 3 \times 7 \times 4) \times (9,835)^2}{49^3} = \cdot 4496$ lb.

,, ,, airways A F & L B, $p = \dfrac{(\cdot 0000000217) \times (120 \times 3 \times 7 \times 4) \times (21,881)^2}{49^3} = \cdot 8901$ lb.

1·3397 lb.

as compared with the 1·33961 lb. before calculated to be operating, the difference being due to a dropping of decimals in making the calculations.

The quantity of air in the shafts A and B then is

20,000
21,881

41,881 cubic feet per minute,

and we may now calculate the probable pressure for overcoming the friction in each, thus—

For shaft A, $p = \dfrac{(\cdot 00000001) \times (56\cdot 548 \times 700 \times 3) \times (41,881)^2}{254\cdot 469^3} = \cdot 1264$ lb.

,, ,, B, $p = \dfrac{(\cdot 00000001) \times (37\cdot 699 \times 690 \times 3) \times (41,881)^2}{113\cdot 097^3} = \cdot 9462$,,

To this add the pressure necessary to overcome the friction of the mine . . . . 1·3396 ,,

2·4122 ,,

making a total of 2·4122 lbs. pressure per square foot equal to a water-gauge of

$$\frac{2\cdot 4122}{5\cdot 2} = \cdot 46388 \text{ inch.}$$

Turning attention now to the workings connected with shafts C and D, Fig. 422 O. As shown by the arrows, the air divides at the bottom of shaft C, one split going in the direction C N a distance of 50 yards, when it is again divided, and the other to C O, a distance of 60 yards. At O it is again divided. The split N P Q is 400 yards in length and at Q re-unites with split N R Q, which is 300 yards in length to the junction of airways at Q. From Q to the upcast shaft D the distance is 80 yards. The split O S T is 200 yards in length, and at T re-joins the air from split O U T, which is 300 yards in length. The distance from T to the upcast shaft D is 60 yards. The sectional area of the airways is greater in the intakes and the actual quantities are

F K L.	As 44·057 : 24·254 :: 24,207 :	13,326
F M L.	„ 44·057 : 19·803 :: 24,207 :	10,881
		24,207 cubic feet per minute.

The pressure necessary for each split is the same, and may be found for either F K L or F M L thus—

For F K L. As $12{,}046^2 : 13{,}326^2 :: {\cdot}4496 : {\cdot}55026$ lb.
or „ K M L. „ $9{,}835^2 : 10{,}881^2 :: {\cdot}4496 : {\cdot}55026$ lb.

The pressure spent in overcoming friction in airways V F and L B, which are together 200 yards in length

$$= \frac{({\cdot}0000000217) \times (200 \times 3 \times 7 \times 4) \times (24{,}207)^2}{49^3} = 1{\cdot}8157 \text{ lb.}$$

or, as $12{,}046^2 : 24{,}207^2 :: {\cdot}4496 : 1{\cdot}8157$ lb.

The addition of the pressures now ascertained is

For C V	= 9·599
„ V F and L B	= 1·8157
„ F K L and F M L	= ·55026
	11·96496 lbs. per square foot.

That is on the assumption that 42,072 cubic feet of air per minute descend shaft C, the pressure to overcome friction in the airways to the point B is 11·96496 lbs. per square foot.

But on the assumption that a quantity of 42,072 cubic feet per minute enter shaft D it has been calculated that the friction in other airways to the point B is equal to 12·10577 lbs. per square foot, showing that both quantities cannot be right. We may, however, calculate the proper amount of increase in the volume entering shaft C or of decrease in the volume entering shaft D to make the calculations of the loss of pressures at B harmonize. This will be very nearly so if we adopt the former alternative and increase the quantity of air entering at C in proportion to the two calculated pressures at B. Thus—

As $\sqrt{11{\cdot}96496} : \sqrt{12{\cdot}10577} :: 42{,}072 : 42{,}320$ cubic feet per minute.

But this will not give a strictly correct result, because when the quantity of air entering shaft C is increased there is a slight disturbance in the pressure at B owing to a proportionate increase in the quantity of air going round V G W. This disturbance, however, is so very slight that practically we may disregard it.

If 42,072 cubic feet of air per minute enter shaft D then 42,320 cubic feet per minute enter shaft C, and the adjusted quantities must be

In airway V G W	As 42,072 : 42,320 :: 17,865 :	17,970
„ F K L	„ 42,072 : 42,320 :: 13,326 :	13,405
„ F M L	„ 42,072 : 42,320 :: 10,881 :	10,945
	Total	42,320 cubic feet per min.

and the correspondingly adjusted pressures are

For C V	As 11·96496 : 12·10577 :: 9·599 :	9·712
„ F K L and F M L	„ 11·96496 : 12·10577 :: ·55026 :	·5567
„ V F and L B	„ 11·96496 : 12·10577 :: 1·8157 :	1·8371
	Total	12·1058 lbs. per sq. ft.

We have now passing along the return airway B A

$$\begin{array}{r} 42{,}072 \\ 42{,}320 \\ \hline 84{,}392 \end{array}$$

cubic feet per minute, the pressure for overcoming which is

$$\frac{(\cdot 0000000217) \times (30 \times 3 \times 40) \times (84{,}392)^2}{96^3} = \cdot 62885$$

That for shaft A is as	$41{,}881^2 : 84{,}392^2 :: \cdot 1264 :$	$\cdot 51323$
,, ,, ,, D ,,	$41{,}881^2 : 42{,}072^2 :: \cdot 12099 :$	$\cdot 1221$
,, ,, ,, C ,,	$41{,}881^2 : 42{,}320^2 :: \cdot 22128 :$	$\cdot 22594$
		$1\cdot 49012$ lbs.

which, added to $12\cdot 10577$, previously ascertained for the mine, makes a total of $13\cdot 59589$ lbs. per square foot for the whole of the workings and shafts with a total ventilation of 84,392 cubic feet per minute.

Referring now to the calculations made for the ventilation of the colliery before the workings at J and R (Fig. 422O) were connected, it was then shown that there were 92,037 cubic feet per minute circulating with a pressure of $2\cdot 4122$ lbs. per square foot or a water-gauge of $\cdot 46388$ inch. In order to compare the water-gauge after the alterations in the mine are carried out with that before, let us increase the total volume of air being discharged at shaft A from 84,392 cubic feet per minute to 92,037, the amount previously circulated by the two ventilating powers. The pressure then becomes—

$$\text{As } 84{,}392^2 : 92{,}037^2 :: 13\cdot 59589 : 16\cdot 17 \text{ lbs.}$$

or a water-gauge of $\frac{16\cdot 17}{5\cdot 2} = 3\cdot 11$ inches.

The proportion of air entering the downcast shafts now is—

Entering D	As $84{,}392 : 92{,}037 :: 42{,}072 :$	45,884
,, C	,, $84{,}392 : 92{,}037 :: 42{,}320 :$	46,153
	Total	92,037

The quantities of air in the different splits of the mine are—

		Cubic feet per minute.
In O S T	As $42{,}072 : 45{,}884 :: 12{,}046 :$	13,138
,, O U T	,, $42{,}072 : 45{,}884 :: 9{,}835 :$	10,726
,, D O and T Y		23,864
,, D X P Y	,, $42{,}072 : 45{,}884 :: 20{,}191 :$	22,020
	Total volume of air in shaft D	45,884
In F K L	As $42{,}072 : 46{,}153 :: 13{,}326 :$	14,619
,, F M L	,, $42{,}072 : 46{,}153 :: 10{,}881 :$	11,936
,, V F and L B		26,555
,, V G W	,, $42{,}072 : 46{,}153 :: 17{,}865 :$	19,598
	Total volume of air in shaft C	46,153
	In airway B A and shaft A	92,037

upcast shaft, while A and C are downcast, B being abandoned as shown in Fig. 422Q.

In this case, the air entering shaft C passes in its entirety to O, at which point splitting into two currents takes place. One part of the divided volume passes round O S T and the other round O U T, the two divisions becoming re-united at the point T and together following the return airway to shaft D as in the first arrangement.

After descending shaft A, here the whole volume of air becomes divided into two currents, one passing to the point E, where it is again split, while the other follows the main intake A F, to the point F, where it is also subdivided. One split from the point E follows the course E N P X and at the point X, in the main return airway joins the air from other splits and along with it passes to and up shaft D. The other split from the point E ventilates the district E G H, and at the point H in the return airway unites with other air currents, the increased volume following the return airway to the point X where it is further augmented by the current from E N P X, and continues with it to and up the shaft D. A split from the point F passes round F K L and another passes round F M L, becoming re-united at the point L as in the first arrangement, and these afterwards pass in one volume to the point H, where the volume is again increased by the current from E G H, the further course to shaft D having been already indicated.

The lengths of the airways are—

C O	60 yards	A E	50 yards
O S T	200 „	E N P X	700 „
O U T	300 „	E G H	400 „
T D	60 „	H X	300 „
		X D	50 „
		A F	60 „
		F K L	200 „
		F M L	300 „
		L H	140 „

The sectional area of the airways is greater in the main intakes before splitting takes place and in the main returns after re-union of the air-currents, but we will assume the average dimensions of all airways except X D, A E, C O and D T to be 7 feet by 7 and X D, A E, C O and D T to be 8 feet high and 12 feet wide.

Let us assume that 12,046 cubic feet per minute pass round F K L and 9,835 cubic feet round F M L with a pressure of ·4496 lb. used in these splits as in the first instance. Then a volume of 21,881 cubic feet passes along the airways A F and L H, a total distance of 200 yards. The pressure necessary for this quantity for a distance of 120 yards has been shown to be ·8901 lb. per square foot, and, therefore, for 200 yards of similar sized airway it will be

As 120 : 200 :: ·8901 : 1·4835 lb. per square foot.

Apart from shaft friction the loss of pressure at H influencing the current round E G H then is

$$\begin{array}{r} \cdot4496 \\ 1\cdot4835 \\ \hline 1\cdot9331 \end{array} \text{ lb. per square foot.}$$

We do not know the quantity passing along A E at present, and so are unable to calculate the pressure necessary for it. The pressure will not, however, be great, and we shall get a very close approximation to the volume going round E G H by taking the proper pressure of 1·9331 lb. for it, including that necessary to overcome the friction of a larger volume in the airway A E and applying it to ascertain the velocity as though that were uniform from the shaft A to the point H.

		lbs. per square foot.
In A E as above		·1066
„ E N P X	As 12·15099 : 4·50835 :: 8·4204 :	3·12419
	Expended at X	3·23079
„ X D as above		·23814
	Expended at bottom of shaft D	3·46893
„ shaft A	As 12·15099 : 4·50835 :: ·3143 :	·11661
„ „ C	„ 12·15099 : 4·50835 :: ·91236 :	·33851
„ „ D	„ 12·15099 : 4·50835 :: 1·5748 :	·5843
	Total	4·50835
In O S T and O U T	As 12·15099 : 4·50835 :: 6·794 :	2·5208
„ C O and D T	„ 12·15099 : 4·50835 :: 2·5552 :	·9481
	Expended at bottom of shaft D	3·4689

The accuracy of these pressures may be tested by the formula $p = \frac{ksv^2}{a}$ when we have

lbs. per square ft.

For F K L $$p = \frac{(\cdot 0000000217) \times (200 \times 3 \times 7 \times 4) \times (7{,}337)^2}{49^3} = \cdot 16681$$

„ F M L $$p = \frac{(\cdot 0000000217) \times (300 \times 3 \times 7 \times 4) \times (5{,}991)^2}{49^3} = \cdot 16682$$

„ A F & L H $$p = \frac{(\cdot 0000000217) \times (200 \times 3 \times 7 \times 4) \times (13{,}328)^2}{49^3} = \cdot 55044$$

„ A E $$p = \frac{(\cdot 0000000217) \times (50 \times 3 \times 40) \times (26{,}899)^2}{96^3} = \cdot 1065$$

„ E G H $$p = \frac{(\cdot 0000000217) \times (400 \times 3 \times 7 \times 4) \times (9{,}926)^2}{49^3} = \cdot 6106$$

„ H X $$p = \frac{(\cdot 0000000217) \times (300 \times 3 \times 7 \times 4) \times (23{,}254)^2}{49^3} = 2\cdot 5135$$

„ E N P X $$p = \frac{(\cdot 0000000217) \times (700 \times 3 \times 7 \times 4) \times (16{,}973)^2}{49^3} = 3\cdot 1244$$

„ X D $$p = \frac{(\cdot 0000000217) \times (50 \times 3 \times 40) \times (40{,}227)^2}{96^3} = \cdot 23814$$

„ O S T $$p = \frac{(\cdot 0000000217) \times (200 \times 3 \times 7 \times 4) \times (28{,}522)^2}{49^3} = 2\cdot 5208$$

„ O U T $$p = \frac{(\cdot 0000000217) \times (300 \times 3 \times 7 \times 4) \times (23{,}288)^2}{49^3} = 2\cdot 5208$$

„ C O & D T $$p = \frac{(\cdot 0000000217) \times (120 \times 3 \times 40) \times (51{,}810)^2}{96^3} = \cdot 9481$$

„ Shaft A $$p = \frac{(\cdot 00000001) \times (700 \times 3 \times 56\cdot 548) \times (40{,}227)^2}{254\cdot 469^3} = \cdot 11661$$

„ „ C $$p = \frac{(\cdot 00000001) \times (680 \times 3 \times 50\cdot 265) \times (51{,}810)^2}{201\cdot 062^3} = \cdot 3386$$

„ „ D $$p = \frac{(\cdot 00000001) \times (670 \times 3 \times 56\cdot 548) \times (92{,}037)^2}{254\cdot 469^3} = \cdot 5843$$

It will be noted that the same total volume of air is maintained in the mine in the first, second, and third arrangements under given conditions, the water-gauge being respectively ·46388, 3·11, and ·86699 inch.

winding. On the descent of the cage in the downcast shaft, the motion of both cages assists the ventilation; but on the reversal of the winding, the inflowing air is retarded by the ascent of the cage in the downcast, while at the same time the flow is checked in the upcast by the cage in its descent there.

A heavy fall of the roof in an airway may be considered in respect of its influence on the water-gauge of the whole colliery and on that of the split in which the fall occurs. If we imagine a case in which the air is carried round the workings in one continuous current, which plan of ventilation at one time prevailed, the effect of a fall forming a partial obstruction must be to increase the water-gauge. Precisely the same result would follow a fall in a main airway through which all the air of a mine was flowing. If a fan produces the ventilation and the same steam supply be continued for the use of the engine driving it, the speed of the fan is increased in consequence of the falling off in the quantity of air passing through the fan. A higher water-gauge would result from this increased speed, for the pressure depends upon the speed of the fan. A normal water-gauge may be maintained by reducing the steam supply, and, if the fan engine is controlled by a governor, on a reduction of the load of air to be dealt with by the fan a reduced steam supply follows.

Usually the fan engineman at a colliery has orders to maintain a certain speed or to keep a constant water-gauge. Under ordinary circumstances of the mine, regularity of speed ensures corresponding regularity of water-gauge, which may be taken as a standard for the engineman's guidance. Any deviation from this would be easily and quickly noticed by observations made in the fan engine-house. If there is an obstruction in the airway and the engine be kept at the same speed the water-gauge would be increased, but the quantity of air would be lessened. On the other hand, if doors were left open in the mine the water-gauge would be decreased, but the volume of air would become greater. The work of the fan may be assisted by dip workings, or retarded by workings to the rise.

If a furnace be used to produce the ventilation, the temperature in the upcast shaft after such a fall will be increased owing to the distribution of the heat of the fire throughout a smaller quantity of air. For the same coal supplied to the furnace there would be a higher water-gauge and a reduction in the velocity of the air. The ventilating pressure depends upon the temperature of the column of air in the upcast shaft. The precise effect on the water-gauge will be greater or less in accordance with the size of the fall and the amount of opening left for the passage of air.

A modern extensive mine, however, has many main splits of ventilation in one seam of coal or at different levels in two or more seams, and a heavy fall in one split may have no appreciable effect on the total water-gauge, whether the ventilation be produced by fan or by furnace. The greater the number of splits the less noticeable is the effect on the water-gauge if a fall occurs in one of them. If the fall is sufficient to entirely stop the ventilation of one split, a re-distribution of the air follows, by reason of which all other splits get a slightly increased quantity. The total increase in other splits, however, never amounts to the loss of air in the airway where the fall has occurred, and the total ventilation is therefore less. If the speed of the fan is regulated to maintain a constant water-gauge, the shaft friction is reduced, leaving a larger proportion of the total ventilating pressure to force air round the airways still unobstructed, each thus getting a slight increase in quantity. If the ventilation be commanded by a furnace and the weight of coal burnt be regulated to maintain a constant water-gauge the effect is the same.

It must be borne in mind that the water-gauge at any colliery is subject to frequent minor fluctuations above and below a certain standard height set up for constant guidance, and it is highly probable that where there are a number of splits a fall of roof in one does not cause perceptible change in the water-gauge or is not sufficient to cause alarm or likely to be attributed to any cause but that of constant minor fluctuations such as result from variation of the fan speed or irregularity of stoking the furnace.

We may consider the effect of a fall of roof on the water-gauge of a particular split by reference to Fig. 422O. If a heavy fall occurs at the point V or at Z it is immediately followed by a greatly reduced quantity of air in the split F K L, while at the same time there will be an increase in the water-gauge of that split. This would be apparent by testing the water-gauge before and after the occurrence of the fall at the point W, the instrument being applied at a stopping arranged for the purpose. There may be no marked change in the total water-gauge of the mine such as to arouse the suspicions of the officials, but the reduced velocity of the air-current in the split F K L would be so noticeable as to lead to quick discovery by those having the constant supervision of the airways. If water-gauges were kept continuously attached to all main separation doors in the connecting roads between the intakes and returns of a mine and the readings regularly registered, the fall in the split F K L would be indicated by the reading of that water-gauge placed at the point W. No useful purpose would be served by the placing of water-gauges in the workings, and it is customary only to fix them permanently in the manager's office and in the fan engine-house connected with the fan drift at the surface or in an office underground, the water-gauge being then connected by a pipe with the return airway. With a furnace ventilation, the water-gauge is usually applied on the separation doors between pits at the bottom seam or other levels. If the fall takes place at any point in the intake between F and W or in the return airway between the points Y and L there is a reduction in the water-gauge at the point W, and the larger the fall the greater will the reduction be, until, if the airway is entirely blocked, there will be no difference whatever in the two limbs of the water-gauge.

The exact effect of the fall can only be ascertained by a careful measurement of the water-gauge and of the quantity of air circulating before and after the fall. Thus, supposing a fall at the point V or at Z increases the water-gauge for the split F K L from 1 to 1·5 inch and reduces the quantity of air one-half, and that the water-gauge at the point W reads ·7 before the fall, then the friction in F W and Y L is equal to $1 - {\cdot}7 = {\cdot}3$ inch of water-gauge before the fall takes place. Afterwards, when the quantity is reduced one-half, the friction in these airways will be reduced to $\frac{1}{4}$ or ·075 inch, whilst round W K Y the frictional resistances in the airways amount to $\frac{1}{4}$ of ·7 or ·175 inch. Together these resistances make ·25 inch due to the passage of the reduced quantity of air throughout the split. If the water-gauge has been raised to 1·5 inch, it is plain that 1·25 inch of water-gauge is the measure of the resistance due to the obstruction itself.

A slight modification of this calculation would be necessary for a fall so extensive as to reduce the area of the airway over a considerable proportion of its length. Such a fall would seriously reduce the length of the original airway and thereby decrease the frictional resistance of the airway considered apart from the obstructed portion.

*Regulators.**—We have endeavoured to show in preceding examples the quantities of air which would flow into the different splits of a mine with a given water-gauge, under the supposition that the air is free to flow into each without check other than that offered by the length and perimeter of the airway itself. We have also given examples showing the disturbing influence of falls in such airways. In few if any mines, however, is the air allowed to distribute itself unrestrainedly into the different districts. It may be that the district most

* A very useful and practical work on this subject, entitled "The Natural Philosophy of a Ventilating Regulator," has been written by H. W. Halbaum, and is published at 27, Wallgate, Wigan.

airway to ·625 inch, and if we maintain the same total water-gauge there will be expended 2·5 − ·625 = 1·875 inch on the regulators. We may now compare the pressures at the various points. Before the regulators were put in the pressure at the point S was 2·5 − 1· × 5·2 = 7·8 lbs. per square foot, at the point F or D 2·5 − 1·8 × 5·2 = 3·64 lbs. per square foot, and at the point G or E 2·5 − 2 × 5·2 = 2·6 lbs. per square foot. Afterwards the pressure at the point S becomes 2·5 − ·25 × 5·2 = 11·7 lbs. per square foot, at the point F or D 2·5 − ·45 × 5·2 = 10·66 lbs., and at the point G or E 2·5 − ·5 × 5·2 = 10·4 lbs. Immediately on the outbye side of the regulator at the point G or E, the pressure becomes 2·5 − (·5 + 1·875) × 5·2 = ·65 lb. per square foot. The difference of pressure then between that point and S is 11·7 − ·65 = 11·05 lbs., as compared with 7·8 − 2·6 = 5·2 lbs., the difference between the same points before the regulators were put in. The difference of pressure between the points F and S or D and S is only 11·7 − 10·66 = 1·04 lb., as compared with 7·8 − 3·64 = 4·16 lbs., the difference between the same points before the regulators were put in. If the coal is unwrought at the points Q and R, and the pillars H, J, K, L, M, N,

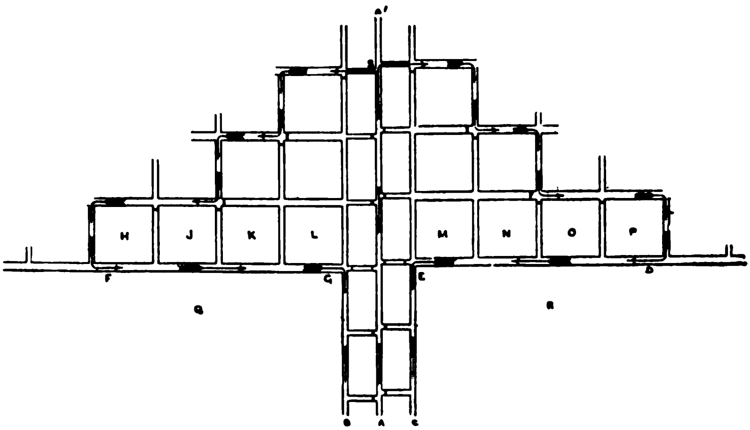


Fig. 422S.—To Illustrate the Effect of Regulators Placed in the Airways of a Mine.

O, P have been worked, the difference of pressure between S and F or S and D before regulators were put in would be more favourable to the escape of gas from the goaf or from a break at the face into the return between F and G or between D and E than afterwards. But if the regulators be placed at F and D the reverse holds good, and the difference of pressure becomes more favourable to the escape of gas into the return airway F G or D E. On the outbye sides of the regulators the pressure would be 2·5 − (·45 + 1·875) × 5·2 = ·91 lb., and the difference of pressure between those points and S would be 11·7 − ·91 = 10·79 lbs. If the mine be fiery and these districts have to be regulated, there is thus an advantage in placing the regulators at F and D in preference to G and E, because fire-damp issues will more easily escape into the return airway so.

Again, take the case of a colliery working the whole and broken mine together, as in Fig. 422T. Here the air passes from the downcast shaft A along the intake airway C to D, continuing through workings in the whole mine E, F, G, and from thence through broken workings to the point H, continuing through the points J, K to and up the upcast shaft B. A regulator, whether placed at E, at J, or at K, would have the same effect in reducing the quantity in the district. But by placing it at H, there is a better distribution of pressures tending to assist the flow of gas from the goaf into the airway behind the regulator, and this then is its proper position in a fiery mine.

It will be readily seen that with the two regulated districts shown in Fig. 422S whether each receives an equal quantity of air or not; either a fall or an obstruction placed in the airway will cause an alteration in the quantity of both. Bratticing the fore-winning place S A will cause slight variations in the two volumes, and this will be more marked in the case of an improperly placed brattice. Fig. 422U shows enlarged views of the fore-winning place. In A the whole of the intake air is made to pass behind the brattice before splitting takes place. When fresh holings to the right and left are made, this brattice is removed and the air is split direct without this restriction, and as the fore-winning place is continued the

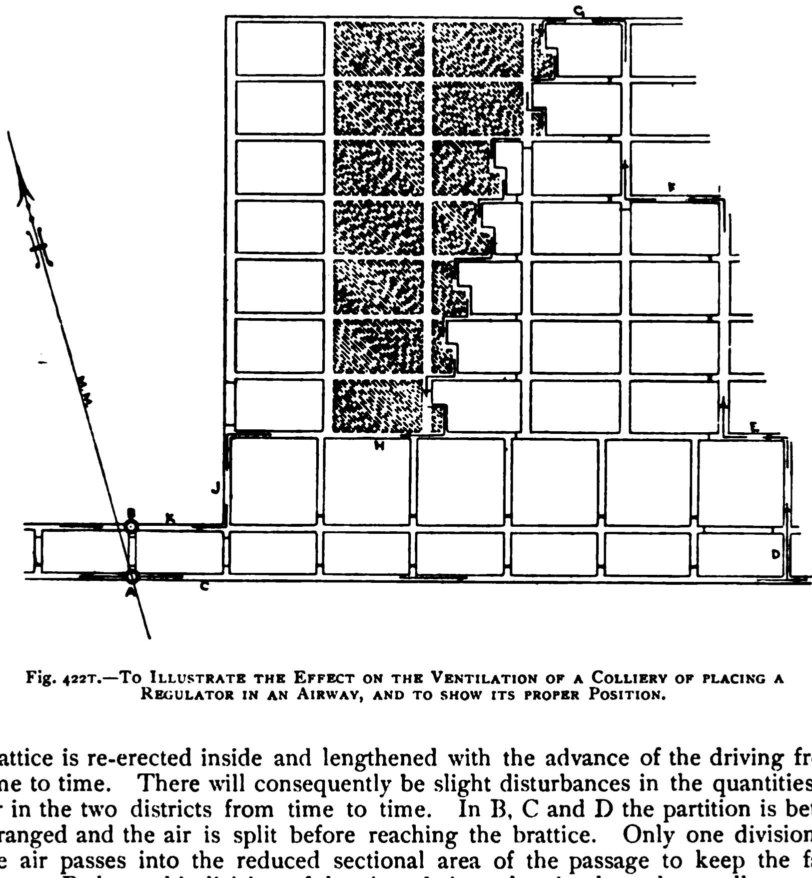


Fig. 422T.—To Illustrate the Effect on the Ventilation of a Colliery of placing a Regulator in an Airway, and to show its proper Position.

brattice is re-erected inside and lengthened with the advance of the driving from time to time. There will consequently be slight disturbances in the quantities of air in the two districts from time to time. In B, C and D the partition is better arranged and the air is split before reaching the brattice. Only one division of the air passes into the reduced sectional area of the passage to keep the face clean. B shows this division of the air as being taken in along the smaller area of the driving, and C shows it taken in along the larger sectional area, while D shows an arrangement if holings are opposite each other. There will be slight variations in volumes even in these arrangements, because of the distance holings are apart and the constantly increasing length of brattice between any two of such holings. Moreover, the back-winning places and the whole " sheth " of bords on each side of the fore-winning places may have to be bratticed to keep the faces clean, and there must be slight variations of the quantities in the two districts due to alterations in the relative amounts of brattice in them which are subject to constant changes. Bratticing the fore-winning place in accordance with A, Fig. 422U, however, is likely to make greater fluctuations in quantities of air than that arising from bratticing the bords, because of oftentimes passing the whole volume of air which feeds the two districts behind the brattice, while only occasionally splitting the air into two full-sized airways in the entire absence of brattice.

Opinions differ as to whether the intake air-current should enter by the smaller or larger side of the partition. Theoretically, as the current expands in its course round the workings, the larger volume should return through the larger section of passage. Practically, it will not matter much, so far as quantity is concerned, which side of the brattice is made the intake; the resistances on both sides must be overcome in turn. Experience seems to show that under certain conditions some benefit to purification of the face is derived by assigning the return air to the smaller section of passage, because the pressures are then distributed with the best possible effect. In workings swept by a brisk current of air we prefer making

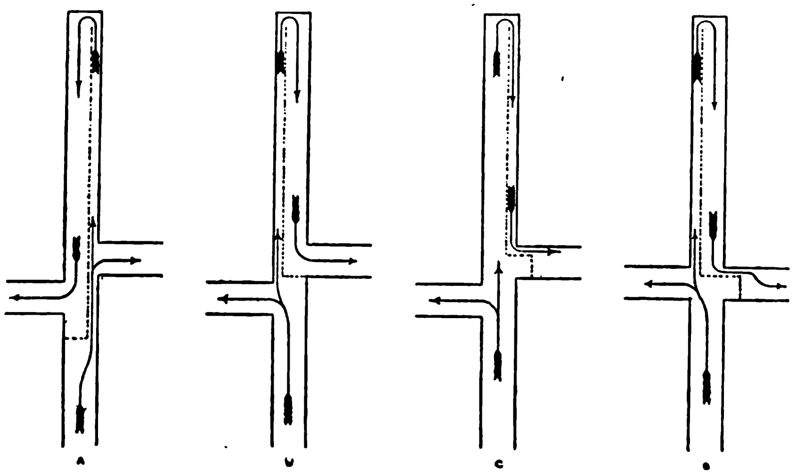


Fig. 422U.—To Illustrate the Effect of Obstructions formed by Brattice in the Airways of a Mine.

the smaller passage the intake. If we take the case of a drift in which a great length of brattice has to be carried, either in the form of a partition, air-tubes, or pipes, as in Fig. 422V, generally a much more feeble current of air is in circulation than in workings which have only short lengths of brattice. As regards quantity, it would appear not to matter whether the air is coursed as at A or as at B in Fig. 422V. If a small fan has to be worked, however, it will be a better arrangement for it to exhaust the air as at A than to force it into the drift as at B, and this arrangement may tend to purification of the air at the face, because the greatest possible pressure is brought on the face of the drift, and the difference between this pressure and that at the entrance to the tubes encourages a flow of vaporous particles into the return.

The effect of placing a regulator in any mine cannot be calculated without taking into consideration the whole circumstances of the mine, the depth and size of shafts, and full particulars of the airways leading from the shafts before and after splitting takes place. Splits subject to a common ventilating pressure must have that pressure increased by placing a regulator in any one of them. The increased resistance of the regulator reduces the quantity of air in the regulated split and also the total quantity of air circulating in the mine. The increased pressure in the unregulated splits increases their ventilation; this increase will be greater or less, in accordance with the length and dimensions of airways between the junction of splits and the shafts, and also by the number and particulars of all other splits in the mine. The total ventilating pressure of the mine must be increased by placing a regulator in any split.

Supposing we have a downcast and an upcast shaft, each of which is 500 yards deep and 10 feet in diameter, connected by a single gallery 1,000 yards long, its

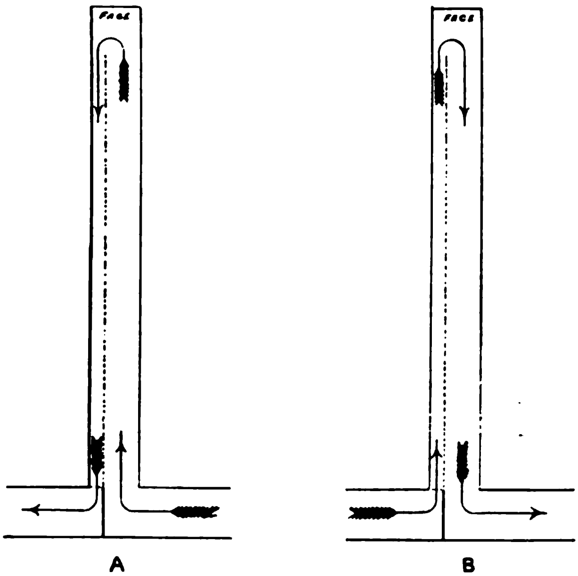


Fig. 422v.—The Ventilation of a Stone Drift or Driving in the Coal by Air-boxes, Pipes, or Brattice.

dimensions being 8 feet by 6. For a 10-lb. pressure in the airway we should have as the velocity

$$v = \sqrt{\frac{10 \times 48}{\cdot 0000000217 \times 84{,}000}} = 513{\cdot}16 \text{ feet per minute,}$$

and $q = 513{\cdot}16 \times 48 = 24{,}631$ cubic feet per minute.

The pressure in each shaft would be

$$p = \frac{(\cdot 00000001) \times (500 \times 3 \times 31{\cdot}416) \times (24{,}631)^2}{78{\cdot}54^3} = \cdot 59 \text{ lb. per square foot.}$$

The total pressure would be:

For the gallery		10·
„ downcast shaft		·59
„ upcast „		·59
		11·18 lbs. per square foot.

Let us now calculate the effect of placing a regulator in the airway with different areas of opening, first to command a ventilating current of 20,000 cubic feet, 20,000 :: 8·345 : 6·775. If the horse-power is to be maintained at 8·345 for the 20,000 cubic feet we have—

$8{\cdot}345 = \dfrac{p \times 20{,}000}{33{,}000}$ whence $p = \dfrac{8{\cdot}345 \times 33{,}000}{20{,}000} = 13{\cdot}77$ lbs. per square foot. With the regulator opening reduced until 10,000 cubic feet sweep the airway and with the same horse-power we have $p = \dfrac{8{\cdot}345 \times 33{,}000}{10{,}000} = 27{\cdot}54$ lbs. With the regulator opening still further reduced to pass 5,000 cubic feet, for the same horse-power we have $p = \dfrac{8{\cdot}345 \times 33{,}000}{5{,}000} = 55{\cdot}07$ lbs., or a water-gauge of 10·59 inches. Such a high water-gauge is rarely found in any mine.

If instead of a single gallery communicating between an upcast and a downcast shaft, let us take an instance of two splits of air commencing at the downcast and ending at the upcast. Each shaft is 500 yards deep and 10 feet in diameter as in the last instance, and each airway is 8 feet by 6, but one is 1,000 yards long and the other 500.

If we take a common pressure of 10 lbs. for the two splits, then in the long airway—

$$v = \sqrt{\frac{10 \times 48}{{\cdot}0000000217 \times 84{,}000}} = 513{\cdot}16 \text{ feet per minute,}$$

$$\text{and } q = 513{\cdot}16 \times 48 = 24{,}631 \text{ cubic feet per minute.}$$

In the 500-yard airway, by the formula $R = \sqrt{\dfrac{a^3}{s}}$ the quantity passing is as $\sqrt{\dfrac{1}{2}} : \sqrt{\dfrac{1}{1}}$ or as $\sqrt{{\cdot}5} : 1 :: 24{,}631 : 34{,}834$ cubic feet. The volume of air in each shaft is 24,631 + 34,834 = 59,465 cubic feet per minute.

The pressure in each shaft for this quantity would be—

$$p = \frac{({\cdot}00000001) \times (500 \times 3 \times 31{\cdot}416) \times (59{,}465)^2}{78{\cdot}54^3} = 3{\cdot}439 \text{ lbs.}$$

$$\text{or as } 24{,}631^2 : 59{,}465^2 :: {\cdot}59 : 3{\cdot}439 \text{ lbs. per square foot.}$$

The total pressure would be—

For the two splits	10·
,, ,, downcast shaft	3·439
,, ,, upcast ,,	3·439
	16·878 lbs. per square foot.

If now a regulator be placed in the short airway it must cause a re-distribution of pressures. Supposing it is so arranged as to pass an equal quantity with the long split, the resistance of the regulator then is equal to the difference of the resistances in the two airways. For a pressure of 10 lbs. in these, the quantities passing must be 24,631 cubic feet per minute in each airway. The frictional resistances in the short airway must be half that for the long, because they pass equal quantities and are similar in area and perimeter, and the frictional resistances must therefore be proportional to their lengths. That is, as 1,000 : 500, or as 2 : 1 :: 10 : 5 lbs. for the 500-yard airway. This leaves 10 − 5 = 5 lbs. as the resistance of the regulator.

But it is impossible that the shaft resistances can remain the same because of the reduced quantity now circulating in them. There would only be a total volume of 24,631 + 24,631 = 49,262 as compared with 59,465 cubic feet per minute in the shafts before the regulator was put in. The pressure for each shaft must be in proportion to the square of the quantity, and would therefore be

$$\text{As } 59{,}465^2 : 49{,}262^2 :: 3{\cdot}439 : 2{\cdot}36 \text{ lbs.}$$

If the pressure for the splits is to remain the same as before, the total pressure must be reduced, because $10 + 2 \cdot 36 + 2 \cdot 36 = 14 \cdot 72$ lbs. as compared with $16 \cdot 878$ before the regulator was put in. The horse-power of ventilation was then

$$\frac{16 \cdot 878 \times 59{,}465}{33{,}000} = 30 \cdot 413.$$

If this same power is maintained to produce 49,262 cubic feet of air, then

$p = \dfrac{30 \cdot 413 \times 33{,}000}{49{,}262} = 20 \cdot 373$ lbs., of which $2 \cdot 36 + 2 \cdot 36 = 4 \cdot 72$ lbs. are absorbed by the shafts. This leaves $20 \cdot 373 - 4 \cdot 72 = 15 \cdot 653$ lbs. as the pressure for the splits. The quantity of air in the long airway then is

$$\text{As } \sqrt{10} : \sqrt{5 \cdot 653} :: 24{,}631 : 30{,}817 \text{ cubic feet per minute.}$$

or we may say the velocity in the air for the long airway

$$\text{is } \sqrt{\frac{15 \cdot 653 \times 48}{\cdot 0000000217 \times 84{,}000}} = 642 \cdot 02 \text{ feet per minute.}$$

and the quantity passing is $642 \cdot 02 \times 48 = 30{,}817$ cubic feet per minute. It is impossible, therefore, if the total quantity is 49,262 cubic feet per minute after the regulator is fixed, and the same power of ventilation maintained, for the two splits to have equal quantities of air in them. The quantity for the shorter must be $49{,}262 - 30{,}817 = 18{,}445$ cubic feet per minute. The pressure necessary to overcome the frictional resistance in the 500-yard airway due to this quantity is

$$p = \frac{(\cdot 0000000217) \times (500 \times 3 \times 28) \times (18{,}445)^2}{48^3} = 2 \cdot 8038 \text{ lbs.}$$

The pressure expended on the regulator then is $15 \cdot 653 - 2 \cdot 8038 = 12 \cdot 8492$ lbs., or $\dfrac{12 \cdot 8492}{5 \cdot 2} = 2 \cdot 471$ inches of water-gauge. The regulator area then must be

$$\frac{\cdot 388 \times 18 \cdot 445}{\sqrt{2 \cdot 471}} = 4 \cdot 5528 \text{ square feet.}$$

If we wish to maintain the same total ventilation of 59,465 cubic feet of air per minute in the shafts, and to split it into two even volumes by placing a regulator in the short split, we shall now show that it can only be done at an increase in the ventilating power. The pressure for the shafts would be $3 \cdot 439$ lbs. for the downcast and $3 \cdot 439$ lbs. for the upcast, together making $6 \cdot 878$ lbs. per square foot. The volume of air for each split is $\dfrac{59{,}465}{2} = 29{,}732$ cubic feet per minute. The pressure necessary to overcome frictional resistances in the 1,000-yard airway is as $24{,}631^2 : 29{,}732^2 :: 10 : 14 \cdot 57$ lbs. That for the shafts being $6 \cdot 878$, we have now a total ventilating pressure of $6 \cdot 878 + 14 \cdot 57 = 21 \cdot 448$ lbs., as compared with $16 \cdot 878$ lbs. pressure for the same quantity of air before the regulator was put in. The pressure spent in overcoming the frictional resistances of the 500-yard airway would be one-half that needed for the 1,000-yard airway, viz., $\dfrac{14 \cdot 57}{2} = 7 \cdot 285$ lbs., the other $7 \cdot 285$ common to the splits being expended on the regulator. This is equivalent to a water-gauge of $\dfrac{7 \cdot 285}{5 \cdot 2} = 1 \cdot 4$ inch. The area of opening in the regulator then must be $\dfrac{\cdot 388 \times 29 \cdot 732}{\sqrt{1 \cdot 4}} = 9 \cdot 746$ square feet. The horse-power of ventilation has been increased from $30 \cdot 413$ to $\dfrac{21 \cdot 448 \times 59{,}465}{33{,}000} = 38 \cdot 65$, or directly in proportion to the increased pressure if the quantity remains the same, viz., as $16 \cdot 878 : 21 \cdot 448 :: 30 \cdot 413 : 38 \cdot 65$.

If a regulator be once set to command a certain proportion of the air in an airway, that proportion will be maintained, even if the total pressure be altered. For

For a total ventilation of 59,465 cubic feet per minute, the proportion going into each airway must be—

For No. 1 split, As	493·59	:	152·735	::	59,465	:	18,401
„ „ 2 „ „	493·59	:	108·	::	59,465	:	13,011
„ „ 3 „ „	493·59	:	88·182	::	59,465	:	10,624
„ „ 4 „ „	493·59	:	76·368	::	59,465	:	9,200
„ „ 5 „ „	493·59	:	68·305	::	59,465	:	8,229
						Total	59,465

This proportion of total volume will be found to hold equally good for airways of similar lengths to these and other area, so long as all five airways are equal in area and perimeter.

We may now find the value of p for the splits, by using the figures of one.

For No. 5 split, $p = \frac{(\cdot 0000000217) \times (1{,}000 \times 3 \times 6 \times 4) \times (8{,}229)^2}{36^3} = 2\cdot 2673$ lbs.

The total pressure for the mine then would be—

For the split airways	2·2673
„ „ outer return airway	2·81
„ „ „ intake „	2·81
„ „ downcast shaft	3·439
„ „ upcast „	3·439
	14·7653 lbs. per square foot.

By increasing the number of splits of the given dimensions from two to five, other things remaining as before, the ventilating pressure would be reduced from 22·498 to 14·7653 lbs. per square foot.

If it is desired to pass equal volumes of air into each split, regulators must be placed in all except the longest, and the area of opening in each will differ, because the resistance offered by each regulator must be such as to equal the difference between the frictional resistances of the longest airway and that airway in which the regulator is placed. Let us assume that we maintain the quantity of 8,229 cubic feet per minute in No. 5 airway, and fix a regulator in No. 4 to reduce its quantity to equal that of No. 5. The frictional resistances for No. 5 are equal to a pressure of 2·2673 lbs. and therefore those for No. 4 will be

As 10 : 8 :: 2·2673 : 1·8138 lb.

The resistance of the regulator in No. 4 airway then must be equal to 2·2673 − 1·8138 = ·4535 lb. or $\frac{\cdot 4535}{5\cdot 2} = \cdot 08721$ inch of water-gauge. The area of opening in the regulator must be—

$$\frac{\cdot 388 \times 8\cdot 229}{\sqrt{\cdot 08721}} = 10\cdot 811 \text{ square feet.}$$

Dealing next with No. 3 airway, its frictional resistances are—

As 10 : 6 :: 2·2673 : 1·3604 lb.

The resistance of the regulator in No. 3 airway then must be equal to 2·2673 − 1·3604 = ·9069 lb., or $\frac{\cdot 9069}{5\cdot 2} = \cdot 1744$ inch of water-gauge. The area of the regulator opening must be—

$$\frac{\cdot 388 \times 8\cdot 229}{\sqrt{\cdot 1744}} = 7\cdot 645 \text{ square feet.}$$

The frictional resistances of No. 2 airway are—

As 10 : 4 :: 2·2673 : ·9069 lb.

Therefore, the pressure produced by the resistance of the regulator in No. 2 airway is $2·2673 - ·9069 = 1·3604$ lbs., or $\frac{1·3604}{5·2} = ·2616$ inch of water-gauge. The area of this regulator opening must be—

$$\frac{·388 \times 8·229}{\sqrt{·2616}} = 6·242 \text{ square feet.}$$

The frictional resistances of No. 1 airway are—

As 10 : 2 :: 2·2673 : ·4535 lb.

The pressure produced by the regulator in No. 1 airway is $2·2673 - ·4535 =$ $1·8138$ lb., or $\frac{1·8138}{5·2} = ·3488$ inch of water-gauge. The area of opening then is—

$$\frac{·388 \times 8·229}{\sqrt{·3488}} = 5·406 \text{ square feet.}$$

But if the pressure for the split airways is the same after the regulators are fixed as before, viz., 2·2673 lbs., it is impossible for the pressure in the shafts and outer airways also to remain unaltered, because the total volume of air is now decreased from 59,465 to five equal splits of 8,229 cubic feet each, or a total of 41,145 cubic feet per minute. The pressure in each shaft for such quantity is

As $59,465^2 : 41,145^2 :: 3·439 : 1·6464$ lb.

and in each outer airway „ $59,465^2 : 41,145^2 :: 2·81 : 1·3453$ lb.

The total pressure of the mine now would be—

For the split airways	2·2673
„ outer return airway	1·3453
„ „ intake „	1·3453
„ downcast shaft	1·6464
„ upcast „	1·6464
	8·2507 lbs. per square foot.

If we raise the total pressure to what it was previous to the placing of regulators, we shall have in circulation—

As $\sqrt{8·2507} : \sqrt{14·7653} :: 41,145 : 55,042$ cubic feet per minute.

If the regulators remain undisturbed, equal quantities will continue to pass into each split, so that if the total quantity be 55,042 cubic feet, each split receives $\frac{55,042}{5} = 11,008$ cubic feet per minute. That the regulator openings which have been fixed to give certain proportions of air in the splits, continue to give the same proportion if the total quantity be varied, may be proved by assuming the total quantity of 41,145 to be decreased to one-fourth of this or 10,286 cubic feet per minute. Each split should then receive $\frac{10,286}{5} = 2,057$ cubic feet per minute or one-fourth of its original volume. The value of p for the longest split then would be—

As $4^2 : 1^2 :: 2·2673 : ·1417$ lb.

The frictional resistances in No. 4 airway will be—

As 10 : 8 :: ·1417 : ·1134 lb.,

and the resistance of the regulator placed there must be equal to $·1417 - ·1134 =$ $·0283$ lb. or $\frac{·0283}{5·2} = ·00544$ inch of water-gauge. The area of opening in the regulator then must be $\frac{·388 \times 2·057}{\sqrt{·00544}} = 10·81$ square feet, or the same as calculated when each split received 8,229 cubic feet per minute. If the openings for the other regulators be calculated they will be found of the same size as already fixed for the higher quantity of air. Having once adjusted regulators in the mine to distribute the air in definite proportions each district will continue to receive the same definite proportion of the total ventilation so long as the circumstances of the mine remain unaltered, even if the total volume be varied from time to time. The introduction of a fresh regulator must of course cause an entirely fresh re-distribution of the air.

Turning now to the example previously worked out and illustrated by Fig. 422P. Here the following volumes of air were shown to be in circulation—

In downcast shaft D	45,884	cubic feet per minute.
„ „ „ C	46,153	„ „ „ „
„ upcast „ A and airway B A	92,037	„ „ „ „

The total ventilation is subdivided in the mine as follows—

In O S T	13,138	cubic feet per minute.
„ O U T	10,726	„ „ „ „
„ D O and T Y	23,864	„ „ „ „
„ D X P Y	22,020	„ „ „ „
Total volume of air in shaft D	45,884	„ „ „ „
In F K L	14,619	cubic feet per minute.
„ F M L	11,936	„ „ „ „
„ V F and L B	26,555	„ „ „ „
„ V G W	19,598	„ „ „ „
Total volume of air in shaft C	46,153	„ „ „ „

The pressures necessary for the production of these volumes are—

For splits O S T and O U T	·5347	lbs. per square foot.
„ „ D O and T Y	2·4703	„ „ „ „
„ „ D X P Y	3·0050	„ „ „ „
From Y to W	10·7644	„ „ „ „
For C V and V G W (V G W being 2·2182)	13·7694	„ „ „ „
„ W B	·6287	„ „ „ „
„ B A	·7479	„ „ „ „
„ shaft A	·6104	„ „ „ „
„ „ D	·1452	„ „ „ „
„ „ C	·2687	„ „ „ „
	16·1703	„ „ „ „
For C V	11·5512	lbs. per square foot.
„ F K L and F M L	·6621	„ „ „ „
„ V F and L B	2·1851	„ „ „ „
From C to B	14·3984	„ „ „ „
For B A	·7479	„ „ „ „
„ shaft A	·6104	„ „ „ „
„ „ D	·1452	„ „ „ „
„ „ C	·2687	„ „ „ „
	16·1706	„ „ „ „

Suppose now it were desirable for all the districts of the mine to receive equal volumes of air.

As the district O U T naturally takes the smallest volume of air, let us take the quantity of 10,726 cubic feet per minute going into it as a standard, and regulate the other districts to have an equal air supply. The shaft D, being the source of supply for three districts, would admit $10,726 \times 3 = 32,178$ cubic feet of air per minute, and the pressure in it would be As $45,884^2 : 32,178^2 :: \cdot 1452 : \cdot 07141$ lb. Dealing first with the airway O S T, which must have a regulator placed in it in order to reduce its volume to 10,726 cubic feet per minute, the frictional resistance of the airway must be As $13,138^2 : 10,726^2 :: \cdot 5347 : \cdot 3564$ lb. The resistance of the regulator in O S T, then, must be equal to a pressure of $\cdot 5347 - \cdot 3564 = \cdot 1783$ lb., or $\frac{\cdot 1783}{5 \cdot 2} = \cdot 03429$ inch of water-gauge. The area of opening in the regulator must be $\frac{\cdot 388 \times 10 \cdot 726}{\sqrt{\cdot 03429}} = 22 \cdot 475$ square feet. The quantity of air in D O and T Y being now reduced from 23,864 cubic feet per minute to 21,452, the pressure would be As $23,864^2 : 21,452^2 :: 2 \cdot 4703 : 1 \cdot 9962$ lb. The loss of pressure from shaft D to point Y, influencing the current of air in D X P Y, is

In shaft D	·07141
In the splits O S T and O U T	·5347
In D O and T Y	1·9962
	2·60231 lbs. per square foot.

We do not require to know the volume of air this pressure would induce in D X P Y. What we have to ascertain is the portion of this pressure of 2·60231 lbs. requisite for the circulation of 10,726 cubic feet per minute in that airway. The frictional resistances of the airway D X P Y due to this volume are As $22,020^2 : 10,726^2 :: 3 \cdot 005 : \cdot 713$ lb. The resistance of the regulator in this airway, then, must be equal to a pressure of $2 \cdot 60231 - \cdot 713 = 1 \cdot 88931$ lb., or $\frac{1 \cdot 88931}{5 \cdot 2} = \cdot 36333$, inch of water-gauge. Its area of opening must be $\frac{\cdot 388 \times 10 \cdot 726}{\sqrt{\cdot 36333}} = 6 \cdot 9043$ square feet. The loss of pressure from Y to W is As $45,884^2 : 32,178^2 :: 10 \cdot 7644 : 5 \cdot 294$ lbs. The total loss of pressure in shaft D and in the airways to W is

2·60231
5·294
7·89631 lbs. per square foot.

This must also be the loss of pressure of the air in shaft C and in the airways C V and V G W.

That for shaft C is	As $46,153^2 : 32,178^2 :: \cdot 2687 : \cdot 13061$	
„ airway C V	„ $46,153^2 : 32,178^2 :: 11 \cdot 5512 : \cdot 615$	
		5·74561 lbs. per sq. ft.

The pressure to be expended in V G W then is $7 \cdot 89631 - 5 \cdot 74561 = 2 \cdot 1507$ lbs. Its frictional resistances for a quantity of 10,726 cubic feet per minute are—

As $19,598^2 : 10,726^2 :: 2 \cdot 2182 :: \cdot 7129$ lb.

B; the other portion passing to the right from J went to K and then, combined with a return current from L, passed through the regulator C R and thence outward to the points X and Y and on to the upcast shaft B.

A few workmen were employed near the point J in working the broken mine near the edge of the large goaf, their coals being brought out by way of B R D, A R D, D, and C to the shaft A.

The principal split of air, starting from the point C, passed to the point M where it was divided into two currents, the principal of which passed to the right to the point N. A small scale of air was taken from the point M, through the

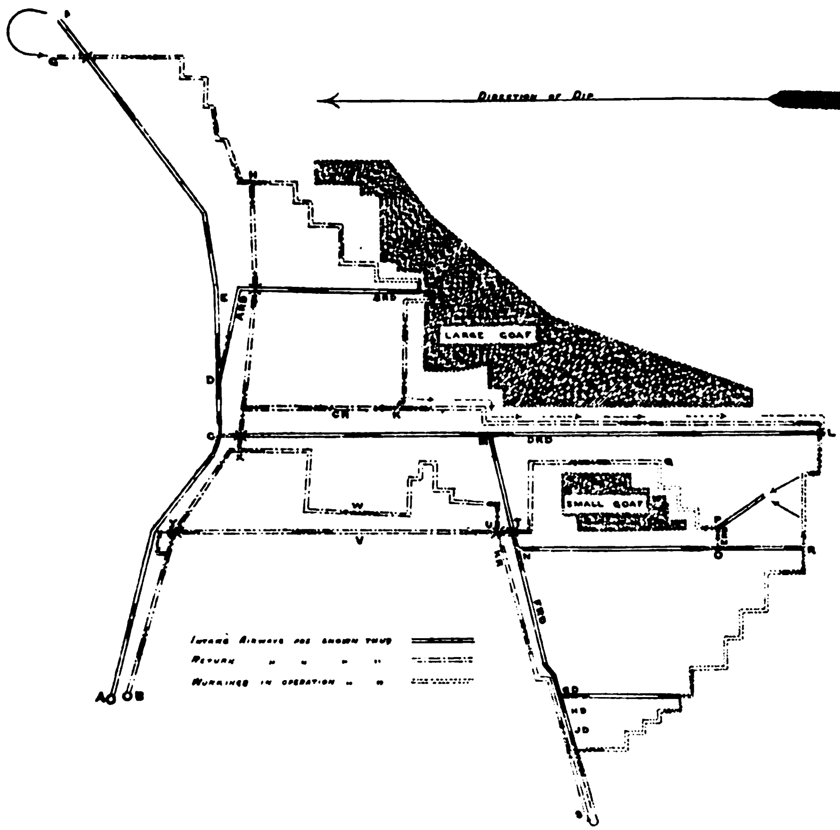


Fig. 422z.—To Illustrate the Effect of Regulators placed in the Airways of a Mine.

regulating door D R D, and another at the point N, through the regulating door F R D, but the chief part of the current passed from N to O, where a small split was taken from it, through the regulator E R D, to ventilate the small goaf, after which this small split passed over the air-crossing T, and so returned through W, X, and Y to the upcast shaft.

The principal current passed forward from O to R, where it was finally split right and left. The split passing to the right from R to the point S was joined by the scales of the doors G D, H D, and J D on its way, and proceeded outward through the regulator K R, and by way of the return V to the point Y, and so on to the upcast shaft. The other split, passing to the left from the point R, flowed to the point L, excepting what leaked through the deal stoppings and went to the small goaf; from L, the last working place, it passed outward in a return airway by the side of the large goaf to K, where it was joined by the split returning from the point J, and along with it passed outward through the regulator C R and thence through X and Y to the upcast shaft B.

These were the ordinary arrangements for the ventilation of the mine, but if the regulating door A R D was pinned open for a short time, while its companion or doubling regulating door B R D was kept closed, it was observed that a portion of the split of air which passed from the point J towards the point K no longer returned outward by way of the regulator C R, nor did any current whatever continue to pass outward from the point L to the point K; but on the contrary, a part of the air from J, on reaching the point K, passed inward from the point K to L, thus converting the return airway between these points into an intake, the air of which returned by one or both of the other general return airways.

The accidental leaving open a regulating door in such a position as that at A R D, even if its fellow regulating door remain properly closed at the time, must be attended with danger if any explosive gas exists at the edge of the large goaf, because then this reversed current after becoming explosive is carried past the workmen's lights at L.

The strata dipped from the point L to the point C, the former being the most elevated part of the workings.

The cause of the reversal of the direction of the air current between the points L and K on opening the regulating door A R D is the re-distribution of pressures throughout the mine brought about by such opening. More or less it would affect the pressure in each airway, and there would in consequence be a readjustment of the various volumes. The pressure necessary to overcome the frictional and regulator resistances of two or more splits is the same. Under the ordinary condition of the mine, then, the loss of pressure from the splitting point C through J to K is exactly the same as that in the other split from the point C through M and L to K. When, however, the door A R D was opened, the additional pressure previously absorbed by the regulator was sufficient to cause a reversal of the air from K to L and the new point of junction of the currents would be the point of equal pressure.

The authors of the paper go on to say, "The understanding of the preceding remarks may perhaps be facilitated by considering—1st. That two splits of air separating from each other have of necessity the same common tension at the point where they separate, or at the splitting point. 2nd. The same splits, on reuniting, have also, at the point of reunion, a common and equal tension. 3rd. Then, from the 1st and 2nd, it follows that the entire resistance, or lost tension, occurring between the splitting point and the point of reunion, is exactly the same in each split; and that the quantity of air that will pass by way of each split will just be such as to produce this result. 4th. The tension of air is increased by the amount of pressure due to the gravitation of the air in such descents as it traverses, in proportion to the vertical fall of such descents, and again, in proportion to the density of the air over each respective portion of such descents, and the tension of air is, in like manner lessened by all the ascents traversed by currents of air."

Again, "This case shows the great danger that may arise from conducting the return current of air from a split having a short run into the return from one having a long run, particularly at a point after the latter has passed a goaf, without having permanent fixed regulators, in positions where they are not liable to be disarranged.

"Had circumstances allowed of ordinary regulators being placed near the points H and K in the three return airways J H, J K, and L K, all the purposes answered by the regulating doors A R D and B R D (both of which were fixed in working

The other split continued a course from L to J, and thence through the district K, after ventilating which it returned by way of N and O to and up the upcast shaft F of the colliery B.

The arrows indicate the direction of the air-currents, air-crossings being shown with a cross. Between the points J and G are a pair of doors separating the air of the two collieries.

The working of the district K proceeded from colliery A, although ventilated by air from the colliery B.

The ordinary conditions of the mine were as stated; but at times, when the ventilation of the colliery A was obstructed or feeble, the current of air ceased to flow from the point L through the district Q, and a current of return air connected with the colliery A passed from the point S through the district Q, and joined the intake air from the colliery B at the point L. The augmented volume then passed through the district K, after which it returned to and ascended the upcast shaft F of the colliery B.

When this reversal of the air occurred, the return air passing from the point S through the district Q to L was most liable to have been charged with firedamp, owing to slackness of ventilation at the colliery A.

Such reversal of the air could only occur when the pressure at the point S, together with the additional pressure operating at the point L, due to the gravitation of the air-column extending from the higher point S, through the district Q to the lower point L, was greater in amount than that of the fresh air coming from the colliery B on reaching the point L, where it was joined by the reversed current.

Any obstruction placed in the shaft D would tend to cause this reversal of air, to guard against the liability of which a new airway, extending from the point P to the point M (Fig. 423A), was driven and an air-stopping was built in the old airway between P and S. After the completion of these works the air passed from the point L, through the district Q, and then on by way of the new airway from P to M, where it joined the return from the district K, and with it passed through N and O to and up the upcast shaft F.

This prevented any intermixture of the air-currents of the two collieries and rendered the ventilation of each separate and distinct from that of the other.

Owing to the air being heated or charged with firedamp in the workings, or perhaps from the effect of both, the return air of most mines is less dense than that of the intake. Usually the leakage to be guarded against is from intake to return, because the pressure in the return airway is less than the pressure in adjacent parts of the intake airway. But it is quite possible for there to be leakage from the return to the intake, although it rarely occurs. Instances of continuous leakage from the return airway of one split into the intake airway of another split have already been given. In a mine having only flat workings continuous leakage from a return to its own intake cannot exist. An intermittent or passing leakage may possibly occur as the result of an obstruction suddenly placed or forming in the return airway at a point outside that of the leakage, or from a sudden pushing forward of the air by such causes as the firing of a shot in the airway inside the point of leakage, or a fall of roof, or the opening or shutting of a door. Continuous leakage from a return airway to its own intake may possibly occur in a mine having dip workings, where the return air is of less density than that of the intake, or in a mine having rise workings, where the return is of greater density than that of the intake. It is caused by natural ventilating pressure. A regulator placed at the outside end of the return airway of such district assists the leakage from return to intake. In shallow mines it is quite possible for the return of a rise district of working to be cooler and of greater density than the intake, and a regulator placed as before stated would assist a leakage from the return to the intake. It will add to the density of the air in the return if CO_2 be given off in the

a = the sectional area of the airway, and s the rubbing surface of the airway. If the airways are of the same length then o the value of their perimeters may be substituted for s. Applying this formula to the splits in question,

$$\text{For No. 1 airway } \sqrt{\frac{30^3}{26,400}} = 1{\cdot}0113 \text{ as the relative volume}$$

$$\text{,, ,, 2 ,, } \sqrt{\frac{36^3}{21,600}} = 1{\cdot}4697 \quad \text{do.} \quad \text{do.}$$

$$\text{,, ,, 3 ,, } \sqrt{\frac{24^3}{16,800}} = {\cdot}907115 \quad \text{do.} \quad \text{do.}$$

$$\text{,, ,, 4 ,, } \sqrt{\frac{20^3}{12,960}} = {\cdot}785674 \quad \text{do.} \quad \text{do.}$$

$$\text{Total } 4{\cdot}173789 \text{ of the} \quad \text{do.}$$

And the actual volumes will be found by proportion thus :—

for No. 1 airway As $4{\cdot}173789 : 1{\cdot}0113 :: 63,480 : 15,381{\cdot}15$ cubic ft.
,, ,, 2 ,, As $4{\cdot}173789 : 1{\cdot}4697 :: 63,480 : 22,352{\cdot}83$,, ,,
,, ,, 3 ,, As $4{\cdot}173789 : {\cdot}907115 :: 63,480 : 13,796{\cdot}53$,, ,,
,, ,, 4 ,, As $4{\cdot}173789 : {\cdot}785674 :: 63,480 : 11,949{\cdot}5$,, ,,

Total $63,480{\cdot}01$,, ,,

The accuracy of this result may be checked by Atkinson's formula $p = \frac{ksv^2}{a}$ when the value of p should work out the same for each of the four airways.

$$\text{For No. 1 airway } p = \frac{{\cdot}0217 \times 26,400 \times {\cdot}5127^2}{30} = 5{\cdot}0196 \text{ lbs. per square foot.}$$

$$\text{,, 2 ,, } p = \frac{{\cdot}0217 \times 21,600 \times {\cdot}6209^2}{36} = 5{\cdot}0196 \quad \text{,,} \quad \text{,,}$$

$$\text{,, 3 ,, } p = \frac{{\cdot}0217 \times 16,800 \times {\cdot}57485^2}{24} = 5{\cdot}0196 \quad \text{,,} \quad \text{,,}$$

$$\text{,, 4 ,, } p = \frac{{\cdot}0217 \times 12,960 \times {\cdot}59747^2}{20} = 5{\cdot}0196 \quad \text{,,} \quad \text{,,}$$

showing that the same pressure satisfies all the splits.

But this pressure of 5·0196 lbs. is only part of the total, there is still to consider that required to pass the whole volume through the large airway. To find the value of p in this airway extending from the pit to the point of splitting $\frac{{\cdot}0217 \times 28,800 \times 1{\cdot}1336^2}{56} = 14{\cdot}34$ lbs. per square foot. Then $5{\cdot}0194 + 14{\cdot}34 = 19{\cdot}3594$ lbs. per square foot as the total pressure, and the water-gauge, according to the formula, would be $\frac{19{\cdot}3594}{5{\cdot}2} = 3{\cdot}72$ inches.

Another method of finding the quantities that would pass along the four splits is, by using Atkinson's formula, $v = \sqrt{\frac{pa}{ks}}$. Assume any value say 1 for p, and having found the relative values of v, the relative quantities in the airways will be these relative velocities multiplied by the areas of the different airways, from which the actual quantities may be obtained.

This method requires more figuring than the formula $R = \sqrt{\frac{a^3}{s}}$, but the student is recommended to try it and compare the result with what is here given.

Example 4.—A mine is ventilated by three splits of air A, B, C;

A being 500 yards long, 5 feet by 6 feet; B is 800 yards long and 5 feet by 4 feet, and C is 700 yards long, 7 feet by 3 feet, all starting and rejoining at the same point. If the quantity in A is 35,000 cubic feet per minute, how much will B and C, which are subject to the same pressure as A, each take?

	Rubbing Surface.	Area.
For A	33,000	30
B	43,200	20
C	42,000	21

By the formula $R = \sqrt{\frac{a^3}{s}}$ the relative quantities are

$$A = \sqrt{\frac{30^3}{33,000}} = \cdot 90453, \text{ the relative volume.}$$

$$B = \sqrt{\frac{20^3}{43,200}} = \cdot 43033 \quad ,,$$

$$C = \sqrt{\frac{21^3}{42,000}} = \cdot 46957 \quad ,,$$

$$\text{Total} \quad 1 \cdot 80443$$

A's actual	quantity is	35,000	cubic ft. per min.
∴ B's is	as ·90453 : ·43033 :: 35,000 :	16,651·2	do.
& C's is	as ·90453 : ·46957 :: 35,000 :	18,169·7	do.
	& the total quantity flowing is	69,820·9	do.

Example 5.—A mine is ventilated by 3 splits of air, A, B, C; A, taking 2,500 cubic feet per minute, B, 1,500 cubic feet per minute, and C, 2,000 cubic feet per minute, out of a total of 6,000 cubic feet, what will each split take if the total ventilation be increased to 75,000 cubic feet per minute?

The quantities would be in proportion thus—

A's quantity is	As	6,000 : 75,000 :: 2,500 :	31,250	cubic ft. per min.
B's ,,	,,	6,000 : 75,000 :: 1,500 :	18,750	do.
C's ,,	,,	6,000 : 75,000 :: 2,000 :	25,000	do.
		Total	75,000	do.

Example 6.—If the quantity passing round a mine in one current before splitting is 10,000 cubic feet per minute where the area of the aircourse is 20 feet (5 feet by 4 feet), and the rubbing surface is 24,000 square feet, what quantity will circulate when the current is split into 2, 3, 4, 5, 6, and 10 equal divisions, the pressure remaining the same?

The formula $R = \sqrt{\frac{a^3}{s}}$ to find relative quantities is equally applicable to this case. First of all take the case of one current of 10,000 cubic feet before splitting, to find the quantity that would pass into 2 equal divisions, the pressure remaining the same.

In the first case before splitting the area is 20 feet, and rubbing surface 24,000 square feet.

In the second case, with 2 splits, there would be an area of 40 feet, and a rubbing surface, the same as before, 24,000. Therefore the relative quantities are

As $\sqrt{\frac{20^3}{24,000}} : \sqrt{\frac{40^3}{24,000}} :: 1$ or as $\sqrt{20^3} : \sqrt{40^3} :: 1 : 2{\cdot}8284$; therefore this simple rule is obtained, that if the rubbing surface and pressure remain unaltered the relative volumes obtained from splitting the air will be in the proportion of $\sqrt{a^3}$. Proceed now to find the relative quantities for 3, 4, 5, 6 and 10 equal divisions thus

For 3 splits as	$\sqrt{20^3} : \sqrt{60^3} :: 1 : 5{\cdot}19614$	as the relative volume.
„ 4 „	$\sqrt{20^3} : \sqrt{80^3} :: 1 : 8{\cdot}$	do.
„ 5 „	$\sqrt{20^3} : \sqrt{100^3} :: 1 : 11{\cdot}18033$	do.
„ 6 „	$\sqrt{20^3} : \sqrt{120^3} :: 1 : 14{\cdot}6969$	do.
„ 10 „	$\sqrt{20^3} : \sqrt{200^3} :: 1 : 31{\cdot}622774$	do.

If 10,000 cubic feet be the volume before splitting then

					cub. ft. along
For 2 splits as	1 : 2·8284	:: 10,000 :	28,284,	or 14,142	each split.
„ 3 „	1 : 5·19614	:: 10,000 :	51,961·4,	or 17,320·5	do.
„ 4 „	1 : 8·	:: 10,000 :	80,000,	or 20,000	do.
„ 5 „	1 : 11·18033	:: 10,000 :	111,803·3,	or 22,360·6	do.
„ 6 „	1 : 14·6969	:: 10,000 :	146,969,	or 24,495	do.
„ 10 „	1 : 31·622774	:: 10,000 :	316,227·74,	or 31,622·774	do.

The same result may be arrived at by working out Atkinson's formula.

Example 7.—Supposing in a mine 50,000 cubic feet of air at the shaft are split into 5 distinct currents of equal volume and subject to the same pressure, that is 10,000 cubic feet pass along each of 5 roads of different lengths, No. 1 being 200 yards, No. 2, 400 yards, No. 3, 600 yards, No. 4, 800 yards, and No. 5, 1,000 yards long, and we wish to know the proportional area of the airways.

Let us assume that No. 1 airway is 3 feet × 3 feet. Then as it is 200 yards long we have $R = \sqrt{\frac{9^3}{200 \times 3 \times 12}} = {\cdot}3182$ as the relative volume for No. 1 airway, and as equal volumes are to go into each airway it is evident that ·3182 must be the relative volume for each of the other airways. Now, taking No. 2 airway we are met with the difficulty that the value of o is unknown; but we will assume that it as well as all the other airways are square, then the value of o will be $4\sqrt{a}$ or $\sqrt{16\,a}$. Therefore $\sqrt{\frac{a^3}{\sqrt{16\,a} \times l}} = R$. $\therefore \frac{a^3}{\sqrt{16a} \times l} = R^2$; $a^3 = R^2 \times \sqrt{16\,a} \times l$ and $a^3 = \sqrt{16\,a}\,R^4\,l^2$; $a^6 = 16\,a\,R^4\,l^2$; $a^5 = 16\,R^4\,l^2$, and therefore $a = \sqrt[5]{16\,R^4\,l^2}$.

Substituting the value of R and of l in the No. 2 airway we have $a = \sqrt[5]{16 \times {\cdot}3182^4 \times 1{,}200^2} = 11{\cdot}87$ and $\sqrt{11{\cdot}87} = 3{\cdot}446$ feet square as the size for No. 2 airway.

For No. 3, $a = \sqrt[5]{16 \times {\cdot}3182^4 \times 1{,}800^2} = 13{\cdot}97$ and $\sqrt{13{\cdot}97} = 3{\cdot}737$ feet square as the size for No. 3 airway.

For No. 4, $a = \sqrt[5]{16 \times {\cdot}3182^4 \times 2{,}400^2} = 15{\cdot}67$ and $\sqrt{15{\cdot}67} = 3{\cdot}958$ feet square as the size for No. 4 airway.

For No. 5, $a = \sqrt[5]{16 \times {\cdot}3182^4 \times 3{,}000^2} = 17{\cdot}13$ and $\sqrt{17{\cdot}13} = 4{\cdot}14$ feet square as the size for No. 5 airway.

The accuracy of the results obtained may be shown by working out the value of p for the 10,000 cubic feet stated to pass along each airway, thus—

For No. 1 airway $p = \frac{{\cdot}0000000217 \times 7{,}200 \times (10{,}000)^2}{9^3} = 21{\cdot}43$ lbs.

„ „ 2 „ $p = \frac{{\cdot}0000000217 \times 16{,}540{\cdot}8 \times (10{,}000)^2}{11{\cdot}87^3} = 21{\cdot}43$ lbs.

$$\text{For No. 3 airway } p = \frac{\cdot0000000217 \times 26{,}906\cdot4 \times (10{,}000)^2}{13\cdot97^3} = 21\cdot43 \text{ lbs.}$$

$$\text{,, ,, 4 ,, } p = \frac{\cdot0000000217 \times 37{,}996\cdot8 \times (10{,}000)^2}{15\cdot67^3} = 21\cdot43 \text{ lbs.}$$

$$\text{,, ,, 5 ,, } p = \frac{\cdot0000000217 \times 49{,}680 \times (10{,}000)^2}{17\cdot13^3} = 21\cdot43 \text{ lbs.}$$

The undesirability of small airways is shown by the great pressure required. If we take the No. 1 airway as being 6 feet × 6 feet, instead of 3 feet × 3 feet, we may ascertain the areas of the others by proportion, thus—

No. 1 airway		36 feet = 6 feet × 6
,, 2 ,,	As 9 : 36 :: 11·87 : 47·48	
	or ,, 1 : 4 :: 11·87 : 47·48	= 6·89 feet × 6·89
,, 3 ,,	,, 1 : 4 :: 13·97 : 55·88	= 7·475 feet × 7·475.
,, 4 ,,	,, 1 : 4 :: 15·67 : 62·68	= 7·917 feet × 7·917.
,, 5 ,,	,, 1 : 4 :: 17·13 : 68·52	= 8·278 feet × 8·278.

Taking No. 1 airway, the proportional value of p for the 9 feet and 36 feet area airways will be

$$\text{As } \frac{\cdot0000000217 \times 7{,}200 \times \left(\frac{10{,}000}{9}\right)^2}{9} : \frac{\cdot0000000217 \times 14{,}400 \times \left(\frac{10{,}000}{36}\right)^2}{36}$$

$$= \frac{1 \times 1 \times 4^2}{1} : \frac{1 \times 2 \times 1^2}{4}$$

$$= \frac{16}{1} : \frac{2}{4} \quad \text{or As } 32 : 1$$

That is, the pressure for the 36 feet area airway is $\frac{21\cdot43}{32} = \cdot67$ lb., and this pressure holds good for the other airways, as may be proved by calculation.

Example 8.—Supposing 9,000 cubic feet per minute circulate through a regulator 30 inches × 20, and it is desired to find how much will circulate if made 30 inches by 30, the total pressure for the whole mine remaining unaltered.

The quantity of air which would pass a regulator after the area of opening is altered must depend upon the ventilation arrangements as a whole, and cannot be calculated apart from their full consideration. Usually, the amount of opening given to a regulator is fixed experimentally. We will take two cases of altering the opening from 30 inches × 20 to 30 inches × 30 in a regulator, to show that the quantity passing must vary under different conditions of ventilation.

First take the case of a downcast and an upcast shaft, each 12 feet in diameter and 600 yards in depth, in which two splits of air ventilate the workings, the one being 1,000 yards long and the other 500, each being distinct and separate from the downcast to the upcast shaft and 6 feet high by 6 feet wide.

The regulator opening of 30 × 20 inches is equal to 4·1666 square feet. By the formula for the equivalent orifice, $a = \frac{\cdot388\,q}{\sqrt{h}}$. Therefore $h = \cdot1505 \left(\frac{q}{a}\right)^2$

Substituting for this, we have $h = \cdot1505 \left(\frac{9}{4\cdot1666}\right)^2 = \cdot7022$ nch of water-gauge or $\cdot7022 \times 5\cdot2 = 3\cdot6513$ lbs. per square foot, absorbed by the regulator. The pressure necessary to overcome friction in an airway 500 yards and 6 feet by 6 feet is $p = \frac{(\cdot0000000217) \times (500 \times 3 \times 24) \times (9{,}000)^2}{36^3} = 1\cdot3562$ lb., and the total loss of pressure in the airway is $3\cdot6513 + 1\cdot3562 = 5\cdot0075$ lbs. This

Example 11.—If 20,000 cubic feet of air pass, the water-gauge being ·9 inch, and it is wanted to know what quantity will pass when the water-gauge is 2·6 inches, remember that the quantity varies as the square root of the pressures. Therefore, as $\sqrt{\cdot 9} : \sqrt{2\cdot 6} :: 20{,}000 : 33{,}993$ cubic feet.

Example 12.—If it is desired to double the quantity of the air, the ventilating pressure must be increased $2^2 = 4$ times.

Example 13.—If with a water-gauge of ·1 inch 20,000 cubic feet of air per minute are obtained, and afterwards the quantity is increased to 60,000 cubic feet per minute, and it is desired to know the height of the water-gauge, remember that the square of the quantity of the air is proportional to the water-gauge, and as there are 3 times the quantity passing, the gauge will be $3^2 = 9$ times higher, and $9 \times \cdot 1 = \cdot 9$ inch as the height of water-gauge.

Example 14.—Supposing two airways (subject to the same pressure) of the same area, passing a total quantity of 100,000 cubic feet of air per minute, the resistances in the airways being in the proportion of 5 to 1, and it is desired to know the quantity going along each.

By the formula $R = \sqrt{\frac{a^3}{s}}$ for the one airway $R = \sqrt{\frac{a^3}{s}} = 1$, supposing a is 1 and s also 1; then for the second airway $a = 1$ and $s = 5$ $\therefore R = \sqrt{\frac{1^3}{5}} = \cdot 447214$. The sum of the two relative quantities is $1 + \cdot 447214 = 1\cdot 447214$, and the actual quantities going along each airway the total of which is 100,000, will be found by proportion, thus:—

As 1·447214 : ·447214	:: 100,000 :	30,901	cubic feet;
and as 1·447214 : 1·	:: 100,000 :	69,098	„ „
	Total	99,999	„ „

per minute, practically 100,000, the difference being due to error in neglecting decimals.

Example 15.—The junction of the two intakes at a colliery having 3 shafts (2 downcast and one upcast) is 300 yards from the downcast in one case and 600 yards in the other, and the distance from the junction to the upcast is 200 yards. The area of all the airways is the same, being 6 feet by 6, and all are subject to the same pressure. What total quantity of air will pass, and what quantity in each airway, the water-gauge at the bottom of the upcast being 1·5 inch?

Assume that v = the velocity of the air per minute through the longer intake,
and $v_{,}$ = the velocity do. do. do. the shorter do.

$$\text{then } p = \frac{\cdot 0000000217 \times 600 \times 3 \times 24 \times v^2}{36} \quad (1)$$

$$\text{and also } p = \frac{\cdot 0000000217 \times 300 \times 3 \times 24 \times v_{,}^2}{36} \quad (2)$$

Therefore the right hand side of (1) = the right hand side of (2), and by cancelling common factors $2v^2 = v_{,}^2$, $\therefore 1\cdot 414213v = v_{,}$. As the areas of the airways are equal, the velocity of the air in that part between the junction and the upcast will be equal to the sum of the velocities in the two intakes, that is, it will be $v + 1\cdot 414213v = 2\cdot 414213v$. As the water-gauge at the bottom of the upcast shaft is 1·5 inch, then $5\cdot 2 \times 1\cdot 5 = 7\cdot 8$ lbs. per square foot as the total pressure producing ventilation. If p represents that part of the pressure which

Substituting the values of p and Q in the formula

$$\sin. a = \frac{93{\cdot}75}{1{,}687{\cdot}5} \times 2 \times \sqrt{\frac{93{\cdot}75}{3}}$$

$$a = 38^\circ 24'$$

$$\text{and } \tfrac{1}{2}\, a = 19^\circ 12'$$

$$\tan. B = \sqrt[3]{\text{Tan. } 19^\circ 12'}$$

$$\text{Log. tan. } 19^\circ 12' = 9{\cdot}541875$$

Less 10

which divided by 3 to cube root $)\bar{1}{\cdot}541875$

$$\bar{1}{\cdot}847292$$

add 10

$$9{\cdot}847292$$

$$= \text{Log. tan. of } 35^\circ 8'$$

$$2\, B = 70^\circ 16'$$

$$x = \frac{2\sqrt{\frac{93{\cdot}75}{3}}}{\sin. 70^\circ 16'}$$

$$x = 11{\cdot}88$$

11·88 feet then is the width of the rectangular airway, and $6 \times 11{\cdot}88 = 71{\cdot}28$ square feet of sectional as compared with 69·659, the area of the square airway and 66·375 the area of the circular airway.

If the water-gauge in the above instance is required, then $p = \frac{{\cdot}0217 \times 160{,}000 \times {\cdot}25^2}{25}$ $= 8{\cdot}68$ lbs. per square foot pressure and $\frac{8{\cdot}68}{5{\cdot}2} = 1{\cdot}67$ inch as the water-gauge.

Observing now somewhat closely the examples just worked out, it is seen that to increase the quantity from 6,250 cubic feet per minute to 22,500 cubic feet per minute the airway must be increased from 5 feet square to one of 8·346 feet square, the pressure and length of airway remaining constant.

It may now be shown that the simplest way of finding the desired side of square for the altered airway is by taking the $\frac{3}{8}$th root of the ratio of the quantities of air and multiplying this result by the side of the known airway, thus in the present example $\left(\frac{22{,}500}{6{,}250}\right)^{\frac{3}{8}} \times 5 = 1{\cdot}66925 \times 5 = 8{\cdot}34625$ feet as the side of the square forming the altered airway, being the same result as already proved.

For circular airways this rule holds good also, applied to the diameters of the airways in the same way as applied to the side of a square airway. That this is true may be seen by another example.

Example 17.—If 8,000 cubic feet of air pass through a circular airway whose diameter is 5 feet, the length being 10,000 feet, and the quantity has to be increased to 20,000 cubic feet per minute by enlarging the airway in circular form, the pressure remaining the same, proceed thus :—

The ratio of the altered quantity to the old quantity would be $\frac{20{,}000}{8{,}000} = 2{\cdot}5$

airways is 10,000 cubic feet per minute and it is desired to know the quantity passing in each, and also what the area of the short airway should be so that each may pass an equal quantity, proceed first as in *Example* 1. The relative quantities passing into the two airways is as $\sqrt{2} : \sqrt{1}$ or as $1{\cdot}4142 : 1$. The sum of these relative quantities is $2{\cdot}4142$, and therefore the actual quantities passing are

For the 800-yard airway as $2{\cdot}4142 : 1 :: 10{,}000 : 4{,}142$ cub. ft.

" 400 do. $2{\cdot}4142 : 1{\cdot}4142 :: 10{,}000 : 5{,}858$ cub. ft.

Next, if each airway is to have an equal quantity passing through it with the same pressure, proceed by the formula $R = \sqrt{\frac{a^3}{s}}$, but as the quantities in the two airways are to be equal, the result is not affected if the square root sign be omitted. For the 14 feet area airway, which is 400 yards long $R = \frac{14^3}{17{,}960} = {\cdot}1527845$. If $a =$ the area of the square airway 800 yards or 2,400 feet long, which, being subject to the same pressure as the 14 feet area one, is to pass a similar quantity of 5,858 cubic feet per minute, then its perimeter is represented by $4\sqrt{a}$ or $\sqrt{16a}$. The value of R therefore for the proposed airway is $\frac{a^3}{2{,}400\sqrt{16a}} = {\cdot}1527845$ and $a^3 = 366{\cdot}68 \times \sqrt{16a}$; $a^3 = \sqrt{2{,}151{,}300\,a}$; $a^6 = 2{,}151{,}300a$; $a^5 = 2{,}151{,}300$ and $a = 18{\cdot}473$ which is the area the longer airway must be made if the shorter one is to remain 14 feet area and subject to the same pressure, or, if the long one is to remain 14 feet, then the shorter one would be

As $18{\cdot}473 : 14 :: 14 : 10{\cdot}61$ feet area.

A square airway 10·61 feet area 400 yards long, and a square airway 14 feet area 800 yards long, both subjected to the same pressure, would always pass equal quantities of air, whether the pressure was increased or decreased, as would also a square airway 14 feet area 400 yards long and a square airway 18·473 feet area 800 yards long under the same circumstances.

If the pressure per square foot was maintained, the quantity passing in each airway would be 4,142 cubic feet per minute or a total of 8,284 cubic feet per minute, as a result of reducing the short airway from 14 to 10·61 feet area so that the quantity of air circulating would thus be reduced $10{,}000-8{,}284=1{,}716$ cubic feet per minute. On the other hand as a result of enlarging the longer airway to 18·473 feet area, there would be 5,858 cubic feet passing in each airway if the pressure per square foot is maintained, or a total of 11,716 cubic feet for the two airways, showing in that case an increase of $11{,}716 - 10{,}000 = 1{,}716$ cubic feet.

If it were desirable to have the same total quantity passing as before with the same pressure per square foot, then in that case observe that the areas of both airways must be altered. They would then pass 5,000 cubic feet each and the question to solve becomes; if a 14 feet area square airway 400 yards long passes 5,828 cubic feet per minute, what must its area be to pass 5,000 cubic feet if the pressure remain constant? The side of a 14 feet area square airway is $\sqrt{14} = 3{\cdot}74166$, and therefore the value of the side of the altered airway would be $\left(\frac{5{,}000}{5{,}858}\right)^{\frac{2}{5}} \times 3{\cdot}74166 = 3{\cdot}512$ and its area $3{\cdot}512^2 = 12{\cdot}334$ which is the area of the 400-yard airway; and similarly for the longer airway the question would become; if a 14 feet area square airway 800 yards long passes 4,142 cubic feet per minute, what must its area be to pass 5,000 cubic feet per minute with the same pressure?

Here $\left(\frac{5{,}000}{4{,}142}\right)^{\frac{2}{5}} \times 3{\cdot}74166 = 4{\cdot}0343$ and $4{\cdot}0343^2 = 16{\cdot}2755$ as the area of the 800-yard airway.

$\cdot 27125$ and $\frac{8\cdot 68}{\cdot 27125} = 32$ as the ratio between the pressures of the two airways. Now take the sides of the two square airways; divide one into the other and take the 5th power of the remainder; in other words take the 5th power of the ratio between the two lengths of sides forming the airways and the result is 32, thus $\frac{10^5}{5^5}$ or $\left(\frac{10}{5}\right)^5 = 32$.

Therefore the rule to find the relative powers for square airways of equal length to pass equal quantities of air is to remember that they are in inverse proportion to the power of their sides. Thus to pass an equal quantity through a 5-foot square and a 10-foot square airway it requires $\left(\frac{10}{5}\right)^5 = 32$ times the power.

Example 31.—Again, it may be shown that the same rules apply equally to rectangular airways, for take a road 4 feet high by 6 feet wide, 1,000 feet long, and passing 5,000 cubic feet of air per minute. Here $p = \frac{\cdot 0000000217 \times 20{,}000 \times 208\cdot 3^2}{24} = \cdot 7846$. Now find the value of p for a rectangular road 8 feet high by 12 feet wide to pass the same quantity along the same length of road. $p = \frac{\cdot 0000000217 \times 40{,}000 \times 52\cdot 083^2}{96} = \cdot 02452$ and the ratio between the pressures in the two cases is $\frac{\cdot 7846}{\cdot 02452} = 32$. But take either the heights or widths of the two roads and find the ratio between them by dividing one into the other and then take the 5th power of the remainder, and the result is 32, thus $\left(\frac{8}{4}\right)^5$ or $\left(\frac{12}{6}\right)^5 = 32$. It is true that in using the rule for rectangular roads the form of the altered roadway must be similar to the original one, or in other words the rule is not applicable unless the proportion of the width to the height is the same in the altered airway as that prevailing in the original one. Bearing this in mind, the relative powers for rectangular airways of similar form and of equal length may be found to pass equal quantities of air by taking the proportion of the fifth power of either the two widths or the two heights of the roads.

Example 32.—If a mine be ventilated by one air current, and with a given power there are obtained 10,000 cubic feet through it, the quantities that would pass if the air be divided into 2, 3, 4, 5 or 10 equal splits, the power remaining the same would be in direct proportion to the area. That is with two splits there would be 20,000, with 3 splits 30,000, with 4 splits 40,000, with 5 splits 50,000, and with 10 splits 100,000 cubic feet. This is apparent when the rule to find the quantity that will pass along airways when the power u is given is observed closely. It is, $q = \sqrt[3]{\frac{u}{ks}} \times a$. In the case of the 2 splits, 3 splits, &c. the values of u, k, and s, are equal in each case, and only a would vary. It will be seen that this is a very different result from that obtained by splitting the air into a number of currents when the *pressure* remains the same as worked out in *Example* 6.

Example 33.—From the formula $q = \sqrt[3]{\frac{u}{ks}} \times a$ is deduced $u = ks \times \frac{q^3}{a^3}$ or $ks\left(\frac{q}{a}\right)^3$ thus:

$$q = \sqrt[3]{\frac{u}{ks}} \times a$$

$$\therefore q^3 = \frac{u}{ks} \times a^3$$

$$\frac{q^3}{a^3} = \frac{u}{ks}$$

$$u = ks \times \frac{q^3}{a^3} \text{ or } ks \left(\frac{q}{a}\right)^3 \text{ or } ksv^3.$$

Example 34.—The rule to find the horse-power producing the ventilation is to divide the units of work by 33,000. If the quantity of air passing in a mine is 50,000 cubic feet per minute, with a 1·5-inch water-gauge, the horse-power producing the ventilation will be found thus :—$\frac{50{,}000 \times 1{\cdot}5 \times 5{\cdot}2}{33{,}000} = 11{\cdot}82$ horse-power.

Example 35.—If 20,000 cubic feet of air pass in a circular airway of 12 feet diameter, what quantity will pass in one of 6 feet diameter ?

As $\sqrt[3]{\left(\frac{12}{6}\right)^5} : \sqrt[3]{1} :: 20{,}000$ or $\frac{20{,}000}{\sqrt[3]{32}} = 6{,}300$ cubic feet per minute.

Example 36.—If a furnace and steam jet combined produce 50,000 cubic feet of air per minute, and the furnace alone 44,000 cubic feet, and it is required to know what the steam jet alone would produce, set down the power of the furnace which alone produces 44,000 cubic feet per minute at 1, then the power of the furnace and steam jet combined will be in the ratio to the power of the furnace as $44{,}000^3 : 50{,}000^3 :: 1 : 1{\cdot}46741$ or $\left(\frac{50{,}000}{44{,}000}\right)^3 = 1{\cdot}46741 : 1$, therefore the power of the steam jet is ·46741. To find the quantity the steam jet alone will produce if a power of 1 produces 44,000 cubic feet proceed thus—As $\sqrt[3]{1} : \sqrt[3]{\cdot 46741} :: 44{,}000 : 34{,}147$, or say if a power of 1·46741 produces 50,000, what will the power of ·46741 produce, thus—As $\sqrt[3]{1{\cdot}46741} : \sqrt[3]{\cdot 46741} :: 50{,}000 : 34{,}147$, which is the quantity in cubic feet per minute the steam jet alone will produce.

Example 37.—If by applying 50,000 units of work 40,000 cubic feet of air are produced, and it is desired to know how many units will be required to circulate 75,000 cubic feet, then as $40{,}000^3 : 75{,}000^3 :: 50{,}000 : 329{,}590$.

Examples have already been worked out showing how the area of an air-course alters to pass different quantities of air with the same pressure, and in these examples the power must vary directly with the increased or decreased quantity.

Example 38.—Take for instance that of an aircourse 8,000 feet long, 5 feet by 5, passing 6,250 cubic feet per minute, and it is proposed to enlarge the area so as to pass 22,500 cubic feet per minute, the pressure remaining the same, which is once more using the figures of *Example* 16. The power u would be increased $\frac{22{,}500}{6{,}250} = 3{\cdot}6$ times. That this is so must be evident because $u = p \times q$. In the first case the power would be $8{\cdot}68 \times 6{,}250$ and in the second $8{\cdot}68 \times 22{,}500$. Also take the formula $q = \sqrt[3]{\frac{u}{ks}} \times a$ to find the quantities in each case, and

In the first, $q = \sqrt[3]{\dfrac{8{\cdot}68 \times 6{,}250}{{\cdot}0000000217 \times 160{,}000}} \times 25 = 6{,}250$ cubic feet.

In the second, $q = \sqrt[3]{\dfrac{8{\cdot}68 \times 22{,}500}{{\cdot}0000000217 \times 286{,}080}} \times 71{\cdot}28 = 22{,}500$ cubic feet.

If required to increase the quantity from 6,250 cubic feet per minute without altering the 5 feet by 5 feet airway the power must be increased thus :—

As $6{,}250^3 : 22{,}500^3 :: 1$ or $\left(\dfrac{22{,}500}{6{,}250}\right)^3 = 46{\cdot}656$ times.

In other words, if the quantity were increased by enlarging the airway as indicated from a 25-foot area one to a 6-foot by 11·88 or 71·28-foot area one, then by the consumption of 3·6 times the amount of coals previously consumed there would be an increase in the quantity of air from 6,250 cubic feet per minute to 22,500 cubic feet per minute ; but had it been attempted to increase the quantity of air to the same extent while the airway remained the same the increased consumption of coal would be 46·656 times.

If the airway had been altered from 5 feet by 5 to 6 feet by 11·88, and it became desirable to have the original quantity of 6,250 cubic feet per minute through the altered airway, the consumption of coals necessary to pass this quantity will be found as follows :

The value of p in the original aircourse has been shown to be 8·68 lbs. For the altered aircourse it would be $p = \dfrac{{\cdot}0000000217 \times 286{,}080 \times 87{\cdot}68^2}{71{\cdot}28} = {\cdot}66954$ lb. and since $u = p \times q$, in each case it would be $8{\cdot}68 \times 6{,}250$ and ${\cdot}66954 \times 6{,}250$.

The quantity 6,250 being common to both, the value of u is in direct proportion to p $\therefore \dfrac{8{\cdot}68}{{\cdot}66954} = 12{\cdot}964$ times less coal required with the enlarged airway to pass the same quantity of air.

Again, by allowing the power to remain the same after the alteration in the airway from 5 feet square to 6 feet by 11·88, then the quantity passing along the enlarged airway will be found thus: for the original airway $u = ks\left(\dfrac{q}{a}\right)^3$, and therefore $\cdot0000000217 \times 160{,}000 \times \left(\dfrac{6{,}250}{25}\right)^3 = 54{,}250$, and applying $q = \sqrt[3]{\dfrac{u}{ks}} \times a$ to the altered airway $\sqrt[3]{\dfrac{54{,}250}{{\cdot}0000000217 \times 286{,}080}} \times 71{\cdot}28 = 14{,}682$ cubic feet per minute, and the velocity would be $\dfrac{14{,}682}{71{\cdot}28} = 205{\cdot}977$ feet per minute, and in that case the value of p for the altered airway is $= \dfrac{{\cdot}0000000217 \times 286{,}080 \times 205{\cdot}977^2}{71{\cdot}28} = 3{\cdot}695$ lbs. as compared with 8·68 lbs. pressure for the original airway to pass 6,250 cubic feet per minute.

It must be borne in mind that the examples which have been worked out refer entirely to the overcoming of frictional resistances, because for simplicity the pressure necessary for velocity need not be considered—it is so little. The velocity of air without resistance is the same that a body would attain in falling the height of the motive column, so that if there is a difference of pressure equal to 36 feet of air column, the theoretical velocity of the air would be $8\sqrt{M}$, where M = motive column in feet or $8\sqrt{36} = 48$ feet per second, because a falling body under the force of gravity would attain a velocity of 8 times the square root of 36 or 48 feet. The co-efficient of friction ·0000217 lb. per square foot of area of section is applicable only to the underground roadways. That for the shafts of different forms has not been definitely fixed by experiment. It is usual, however, to take ·00001 lb. per square foot for circular shafts, where the smooth surface gives little impediment to the passage of air.

Question 86.—If one ventilating fan produces 25,000 cubic feet of air per minute, what total quantity of air will be given if another fan of the same dimensions be added to it and worked with an equal power?

As $\sqrt[3]{1} : \sqrt[3]{2} :: 25{,}000$, or as $1 : 1{\cdot}25992 :: 25{,}000 : 31{,}498$ cubic feet per minute.

Question 87.—What is the general efficiency of fans? If the horse-power of an engine is 40, and the water-gauge is 1·5 inch, what quantity of air would you expect to get?

The general result obtained from a fan is about 50 per cent. of the indicated horse-power of the engine, realised in useful work done on the air. With an engine of 40 indicated horse-power the horse-power in the air therefore would be as $100 : 50 :: 40 : 20$, and as the rule is; horse-power in the air $= \dfrac{\text{Quantity in cubic feet per minute} \times \text{water-gauge} \times 5{\cdot}2}{33{,}000}$ $\therefore$ the quantity $= \dfrac{\text{H. P} \times 33{,}000}{\text{water-gauge} \times 5{\cdot}2}$ $\therefore$ $\dfrac{20 \times 33{,}000}{1{\cdot}5 \times 5{\cdot}2} = 84{,}615$, which is the quantity in cubic feet per minute we may expect to get.

Question 88.—The temperature in the downcast is 50° F., that in the upcast is 100° F. What would be the increase in the velocity of the air by doubling the depth of the pits, the temperature remaining the same?

The power of a furnace consists in the difference it can produce between the pressures per square foot at the bottom of the two shafts, multiplied by the velocity at which this difference is maintained, and as the said difference of pressure depends on the difference between the weights of a cubic foot of air in the two shafts multiplied by the depth of the upcast shaft, therefore if the temperature in the shafts remains constant, so also will the weight of given volumes of air, and hence the pressure produced by the furnace will then vary as the depth of the upcast shaft.

By doubling the depth of the upcast shaft the pressure produced will be doubled, and as the velocity varies as the square root of the pressure giving rise to it, the velocity will be increased $\sqrt{2} = 1{\cdot}4142$ times. The power of the furnace would be $1{\cdot}4142 \times 2 = 2{\cdot}8284$ times as great in the latter case. The quantity of coals consumed will be a little more than 1·4142 times, for the quantity of air to be heated is increased to that extent, and the initial temperature will need to be higher in the last case to get the same average temperature in the deeper shaft. This is disregarding the extra friction added by lengthening the upcast shaft, which would reduce the quantity, and this may or may not be to a serious extent according as the shaft friction is or is not a large proportion of the friction of the whole mine.

Question 89.—A barometer registers 30 inches at the surface, what will it register at a point 1,890 feet below the surface?

By the formula given under the remarks on barometer in Chapter XI., $I = \dfrac{D \times B}{26{,}216}$;

$I = \dfrac{1{,}890 \times 30}{26{,}216} = 2{\cdot}16$, therefore the barometer would register $30 + 2{\cdot}16 = 32{\cdot}16$ inches at a point 1,890 feet below the surface. Or say that a barometer rises $\frac{1}{10}$th of an inch for every 88 feet of descent. Therefore $\dfrac{1890}{88} = 21{\cdot}5$, which divided by 10 gives 2·15 as the rise of the barometer for the depth given, almost as before.

Question 90.—Give the size of two airways whose perimeters are equal; the area of one being one and a half times larger than that of the other.

Here 9 feet × 6 feet = 30 feet perimeter and area 54 feet.
12 ,, × 3 ,, = 30 ,, ,, ,, 36 feet and
$36 \times 1\frac{1}{2} = 54$ feet.

Question 91.—How much more resistance will a current of 600 feet per minute meet with than one 500 feet per minute, the aircourse being the same? The latter has a water-gauge of ·76 inch, what will that of the former be?

The resistance is as the square of the velocity while the aircourse remains the same, therefore if the velocity be increased from 500 feet to 600 feet per minute the resistance will be as $5^2 : 6^2 = \frac{36}{25} = 1{\cdot}44$ times greater, and the water-gauge would be $\cdot 76 \times 1{\cdot}44 = 1{\cdot}0944$ inch.

Question 92.—The quantity of air which must be made to flow in a new mine is 116,640 cubic feet per minute. What size ought the shaft to be for the air to have a mean velocity in the shaft of $7\frac{1}{2}$ feet per second.

Here $7\frac{1}{2}$ feet per second = 450 feet per minute and $\frac{116{,}640}{450} = 259{\cdot}2$ square feet area of shaft. Its diameter therefore is $\sqrt{\frac{259{\cdot}2}{\cdot 7854}} = 18{\cdot}166$ or 18 feet 2 inches.

Question 93.—There are two airways; the first is 8 feet square, the second 10 feet by 4. The same quantity of air is wanted to be passed in each airway; the water-gauge stands at ·2 inch in the first, what will it register in the second case?

By Atkinson's formula $p = \frac{ksv^2}{a}$, and as k is common to both airways omit it and for s substitute the value of o. The relative velocities will be in proportion to the areas of the roads, viz., as 40 : 64, or as 5 : 8. The value of p in the two roadways will then be in the following proportion:—

As $\frac{32 \times 5^2}{64} : \frac{28 \times 8^2}{40} = 12{\cdot}5 : 44{\cdot}8$, and therefore the pressure in the smaller airway is $\frac{44{\cdot}8}{12{\cdot}5} = 3{\cdot}584$ times that of the larger, and the water-gauge in the larger being ·2 inch, that of the smaller is $\cdot 2 \times 3{\cdot}584 = \cdot 7168$ inch.

Question 94.—What horse-power would an engine exert when yielding 60 per cent. of duty to move 100,000 cubic feet of air a minute; the water-gauge stands at 1 inch?

$\frac{100{,}000 \times 5{\cdot}2}{33{,}000} = 15{\cdot}75$ horse-power effective ∴
As 60 : 100 :: 15·75 : 26·25 horse-power exerted by engine.

Question 95.—In an airway 10 feet by 6 passing 30,000 cubic feet of air per minute, with a velocity of 500 lineal feet per minute, the water-gauge stands at ·5 inch. What will the water-gauge show if the same quantity be passed through an airway 8 feet by 5?

Here $\frac{100}{20}$ = 5 stentons 10 yards long = 50 yards, and the winning places 100 × 2 = 200 yards. 200 + 50 = 250 × 3 × 40 the area = 30,000, which is the total number of cubic feet of undiluted gas.

If $\frac{1}{30}$th be taken as the proportion of gas that shows a slight cap on the safety lamp flame, the quantity of gas that can be taken away per minute will be $\frac{6{,}000}{30}$ = 200 cubic feet, $\therefore$ the time occupied in removing the whole will be $\frac{30{,}000}{200}$ = 150 minutes = $\frac{150}{60} = 2\frac{1}{2}$ hours.

> *Question* 100.—An explosive mixture of air and gas at the highest explosive point passes along an airway 4 feet by 5 at a velocity of 500 feet per minute. What quantity of fresh air must be added so that you cannot detect it on the flame of a safety lamp?

Here the area of the airway is 4 × 5 = 20 feet × 500 = 10,000 feet of air and gas. When at the highest explosive point the mixture is 1 of gas to 8 or 9 of air, but taking it at one of gas to 9·5 of air, as proportioned by some, then

As 10·5 : 9·5 :: 10,000 : 9,047·6 cubic feet of air,
and as 10·5 : 1 :: 10,000 : 952·4 ,, ,, firedamp.

When the mixture is 31 of air to 1 of the gas its presence cannot be detected, so that 952·4 × 31 = 29,524 cubic feet as the quantity of air required, but as there are already 9,047 cubic feet of air it would take an additional quantity of 29,524 − 9,047 = 20,477 cubic feet per minute of fresh air to dilute the gas so that it could not be detected by means of the lamp.

> *Question* 101.—How long would it take an air-current of 20,000 cubic feet per minute to make the circuit of an airway 4,000 yards long, 10 feet by 10 feet?

10 feet × 10 feet = 100 feet sectional area, and $\frac{20{,}000}{100}$ = 200 feet per minute as the velocity of the current. $\therefore$ $\frac{4{,}000 \times 3}{200}$ = 60 minutes or 1 hour to make the circuit.

> *Question* 102.—A current of 2,000 cubic feet a minute is at explosive point. How much fresh air must be mixed with it to prevent its showing a "cap"?

When the firedamp forms as much as one part out of thirteen of an air-and-gas mixture it becomes explosive.

The gas evolved, therefore, would be $\frac{1}{13}$ of 2,000 = 153·846 cubic feet per minute, and the quantity of air is 2,000 − 153·846 = 1,846·154. In a mixture of one part of firedamp to 30 of air the flame will not show a cap. 153·846 × 31 = 4,769 cubic feet per minute as the total ventilation, when the requisite change has taken place. Then the fresh air required to accomplish this will be 4,769 cubic feet, but as already 1,846 cubic feet of fresh air are passing, an additional quantity of 4,769 − 1,846 = 2,923 cubic feet per minute will be required.

> *Question* 103.—The depth of downcast is 100 fathoms, with a temperature of 50° F., the depth of upcast is 150 fathoms, with a temperature of 100° F. What water-gauge should this show?

feet per minute circulating after putting an additional airway of the same dimensions as the original one.

Or $q = \sqrt[3]{\frac{u}{ks}} \times a$.

The value of u, s, and a, may be set down at 1 each when 20,000 cubic feet pass, and the altered conditions will then necessitate u being made 1, and s and a 2 each. The value of k may be omitted, as it is common to both airways, and will not affect the result. The quantity that would pass then is as $\sqrt[3]{\frac{1}{1}} \times 1 : \sqrt[3]{\frac{1}{2}} \times 2 :: 20,000$. $\therefore 1 : \cdot 7937 \times 2 :: 20,000$, and $20,000 \times \cdot 7937 \times 2 = 31,748$.

Question 108.—A steam jet and fan, both acting together in an upcast shaft, produce 50,000 cubic feet of air per minute; when the fan is stopped the jet gives 10,000 alone; what would be the result were the jet removed?

The power required to ventilate a given mine varies as the cube of the quantity; if therefore the power of the steam jet which gives 10,000 cubic feet is set down at 1, then the power of the two combined which produces 5 times the quantity will be in the ratio to the power of the steam jet as $5^3 : 1^3$ or as $125 : 1$. Therefore the power of the fan is 124, that of the steam jet being 1. To find the quantity the fan alone will produce—if a power of 1 will produce a quantity of 10,000 what quantity will a power of 124 produce? or, if a power of 125 produce a quantity of 50,000 what will be produced by a power of 124? It is equally true that the quantity varies as the cube root of the power; therefore, As $\sqrt[3]{1} : \sqrt[3]{124} :: 10,000 : 49,866$;
or, as $\sqrt[3]{125} : \sqrt[3]{124} :: 50,000 : 49,866$;
that is, the fan alone would give 49,866 cubic feet per minute.

Question 109.—There are two airways in a mine whose lengths are 2,000 feet and 3,000 feet respectively, subject to the same ventilating pressure. The shorter is 5 feet × 5, what size must the longer be made so that each may pass an equal quantity?

Here it is possible to proceed by the formula $R = \sqrt{\left(\frac{a^3}{s}\right)}$, but a simpler way of getting the same result is to remember that the side of the square airway will vary in the proportion of the 5th root of the lengths, when the roads are subject to the same pressure and are to pass the same quantities; thus, As $\sqrt[5]{2,000} : \sqrt[5]{3,000} :: 5$; or,

As $\sqrt[5]{2} : \sqrt[5]{3} :: 5 : 5\cdot 42235$ as the side of the longer airway and its area would therefore be $5\cdot 42235^2 = 29\cdot 4$.

Similarly for a circular airway the diameter varies in the same ratio. Thus if the airways were circular in form, the diameter of the shorter would be $\sqrt{\frac{25}{\cdot 7854}} = 5\cdot 6419 \therefore \sqrt[5]{2} : \sqrt[5]{3} :: 5\cdot 6419 : 6\cdot 1185$ as the diameter of the 3,000-foot airway and its area therefore would be $6\cdot 1185^2 \times \cdot 7854 = 29\cdot 402$, the same as that already worked out for the square airway.

So also for a rectangular airway the height and the width of the airway vary in the same ratio. Thus if the airway had been rectangular and 4 feet high, its width would be $\frac{25}{4} = 6\cdot 25$ feet. For the 3,000-foot airway then As $\sqrt[5]{2} : \sqrt[5]{3} :: 4 : 4\cdot 3379$ which is the height of the longer airway : $\sqrt[5]{2} : \sqrt[5]{3} :: 6\cdot 25$: or put it as $4 : 4\cdot 3379 :: 6\cdot 25 : 6\cdot 7779$ as the width of the longer airway, and therefore the area would be $4\cdot 3379 \times 6\cdot 7779 = 29\cdot 4$ as before.

Question 111.—Compare the friction in the following roads:—No. 1 is 8 feet by 5; No. 2 is 6 feet by 12; (*a*) for the same quantity and (*b*) for three times the quantity of air.

First, by the formula $R = \sqrt{\left(\frac{a^3}{s}\right)}$, or as R must be the same in each case, omit the sign of the square root, and the result is not affected. Therefore, there are, as the relative resistances in the two roads for the same quantity of air, As $\frac{40^3}{26} : \frac{72^3}{36}$, or as 2,461·5 : 10,368 :: 1 : 4·212; that is, the resistance offered by No. 1 airway will be 4·212 times as great as that offered by No. 2 airway for the same quantity of air.

If 3 times the quantity of air is to circulate through No. 2 as compared with No. 1, then proceed by the formula $u = ks\left(\frac{q}{a}\right)^3$ or ksv^3 to find the relative resistances.

The value of k will be the same in each case, and being a common factor, may be omitted, and the relative resistances then become sv^3. There being no definite quantity of air assigned as passing through the airways, for convenience make the quantity passing through No. 1 a multiple of its area, say 40, then the quantity passing through No. 2 will be $40 \times 3 = 120$. The relative velocities must then be $\frac{40}{40} = 1$ and $\frac{120}{72} = 1 \cdot \dot{6}$. The relative pressures will be

For No. 1 airway $26 \times 1^3 = 26$ (substituting o for s).

" " 2 " $36 \times 1 \cdot \dot{6}^3 = 166 \cdot \dot{6}$ do.

Therefore $\frac{166 \cdot 6}{26} = 6 \cdot 41$ times as much friction in No. 2 as in No. 1 for 3 times the quantity of air.

Question 112.—Given 10,000 cubic feet of air per minute through an airway 10 feet by 8, with a pressure of 10 lbs. per square foot, what is the length of the airway?

By the formula $s = \frac{pa}{kv^2}$, $s = \frac{10 \times 80}{\cdot 0000000217 \times 125^2} = 2,359,446$ square feet of rubbing surface, and since the perimeter is 36, the length of the airway is $\frac{2,359,446}{36} = 65,540$ feet.

Question 113.—What precautions should be adopted where candles and safety lamps are used in different parts of the same mine?

The practice of having "mixed lights" in a mine is one to be condemned, but if some extraordinary event made it necessary for a temporary purpose, the points to be observed are to see that no return air after having passed through the workings lighted by safety lamps should be allowed to pass through the workings lighted by candles or open lights. Each district should be ventilated by means of a separate intake and return airway. Proper precautions should be taken to prevent persons using candles from entering that part of the mine lighted by safety lamps.

By General Rule 8 of the Coal Mines Act, 1887—"When it is necessary to work coal in any part of a ventilating district with safety lamps, it shall not be allowable to work the coal with naked lights in another part of the same ventilating district situated between the place where such lamps are being used and the return airway."

The theoretical water-gauge (close to the fan) that a Guibal fan $21\frac{1}{4}$ feet in diameter, going at 40 revolutions per minute, would produce, is ·902 inch (the height of barometer and thermometer slightly affects the water-gauge).

Thus, by the formula $h = \frac{v^2}{64·4}$, where v = speed of the tips of vanes in feet per second, and h = the height of column necessary to create such velocity in a falling body,

$$\therefore h = \frac{\left(21·25 \times 3·1416 \times \frac{40}{60}\right)^2}{64·4}$$

$h = 30·7$, and therefore the water-gauge would be $\frac{30·7 \times 2}{68} = ·902$ inch.

The depth and size of the shafts are not stated, and even if they were it could only be told approximately what proportion of the ventilating pressure will be spent in forcing the air through them. Assume it to be $\frac{1}{4}$th of the total pressure; for overcoming the friction due to the shafts, then $·902 - ·226 = ·676$ as the water-gauge due to the underground workings between the two shaft bottoms; and as the airways must all leave the main intake at 60 fathoms from the downcast, and all come into the return at the bottom of the upcast, the water-gauge due to each of the 4 split airways will be the same.

The relative pressures for overcoming the friction are as follow:—

	Area in feet.	Rubbing surface in square feet.
Main Intake	72	12,960
No. 1 airway	30	132,000
„ 2 „	30	132,000
„ 3 „	40	187,200
„ 4 „	40	187,200

Assuming now that any quantity, say 50,000 cubic feet, is passed along the main airway, the relative quantities going into the four airways, which must get the total among them will be found by the formula $R = \sqrt{\left(\frac{a^3}{s}\right)}$, thus,—

$$\text{For No. 1 airway } R = \sqrt{\frac{30^3}{132,000}} = ·45226$$

$$\text{„ „ 2 „ „ } = \sqrt{\frac{30^3}{132,000}} = ·45226$$

$$\text{„ „ 3 „ „ } = \sqrt{\frac{40^3}{187,200}} = ·5847$$

$$\text{„ „ 4 „ „ } = \sqrt{\frac{40^3}{187,200}} = ·5847$$

Total 2·07392

For a total of 50,000 cubic feet circulating, the proportion of volumes going into each airway must be,

For No. 1 As 2·07392 : ·45226 : : 50,000 : 10,904
„ 2 „ 2·07392 : ·45226 : : 50,000 : 10,904
„ 3 „ 2·07392 : ·5847 : : 50,000 : 14,096
„ 4 „ 2·07392 : ·5847 : : 50,000 : 14,096

By the formula $p = \frac{ksv^2}{a}$ now proceed to find the relative pressures for the produced if the air was split into three divisions, the first airway being as the above, the second 600 fathoms in length, 9 feet in breadth, and 5 feet in height, the third 800 fathoms in length, 10 feet in breadth, and 6 feet in height, the power being the same?

Use the formula $u = ks\left(\frac{q}{a}\right)^3$ or ksv^3 and for the airway passing 20,000 cubic feet at a velocity of $\frac{20,000}{32} = 625$ feet a minute $u = \cdot0000000217 \times 72,000 \times 625^3 = 381,444\cdot7$ units, or $\frac{381,444\cdot7}{33,000} = 11\cdot5589$ horse-power.

Then, for the power to remain the same for the 3 airways,

	Area in feet.	Rubbing surface in square feet.
For the 1st 8 feet × 4 feet × 3,000 feet long	32	72,000
,, 2nd 9 feet × 5 feet × 3,600 ,,	45	100,800
,, 3rd 10 feet × 6 feet × 4,800 ,,	60	153,600
Total	137	326,400

Then by the formula $q = \sqrt[3]{\frac{u}{ks}} \times a$; $\sqrt[3]{\frac{381,444\cdot7}{\cdot0000000217 \times 326,400}} \times 137 = 51,736$ cubic feet as the quantity which would circulate with the same power.

Question 121.—If in a heading 7 feet 6 inches by 6 feet 8 inches the air travels 40 yards in 12 seconds, what would be the quantity of air passing per minute? If the water-gauge was 2·5 inches, what would be the horse-power?

Here $7\frac{1}{2} \times 6\frac{2}{3} = \frac{15}{2} \times \frac{20}{3} = \frac{300}{6} = 50$ feet area of airway. 40 yards = 120 feet, and if the air travels that distance in 12 seconds, its velocity must be as 12:60::120:600 feet per minute, and the quantity passing is 600 × 50 = 30,000 cubic feet per minute. With a 2·5-inch water-gauge the horse-power of ventilation is $\frac{30,000 \times 5\cdot2 \times 2\cdot5}{33,000} = 11\cdot82$.

Question 122.—If with a ventilating fan running at 65 revolutions per minute 1·02 inch of water-gauge is produced, what will be the water-gauge if the fan speed be increased to 96 revolutions per minute?

The water-gauge varies as the square of the quantity and also, since the quantity is proportional to the speed of the fan, as the square of the revolutions. $\therefore \left(\frac{96}{65}\right)^2 \times 1\cdot02 = 2\cdot22$ inches.

Question 123.—If with a ventilating fan running at $93\frac{1}{2}$ revolutions per minute 1·3 inch of water-gauge is produced, what will be the water-gauge if the fan speed be altered to 82 revolutions per minute?

As the water-gauge varies as the square of the revolutions, then as $93\cdot5^2 : 82^2 :: 1\cdot3$ or $\left(\frac{82}{93\cdot5}\right)^2 \times 1\cdot3 = 1$ inch of water-gauge.

Question 124.—If a ventilating fan is running at 40 revolutions per minute with 1·5 inch of water-gauge, and it be altered so that the water-gauge reads 2·6 inches, what will be the fan speed?

The quantity of air passing varies as the square root of the water-gauge, and as the quantity is in direct proportion to the fan speed, the latter also varies as the square root of the water-gauge $\therefore \sqrt{\frac{2 \cdot 6}{1 \cdot 5}} \times 40 = 52 \cdot 67$ revolutions.

Question 125.—If a ventilating fan is running at 80 revolutions per minute with 3·75 inches of water-gauge, and the speed be altered so that the water-gauge reads 1·82 inch, what will be the fan speed?

Here, As $\sqrt{3 \cdot 75} : \sqrt{1 \cdot 82} :: 80$ or $\sqrt{\frac{1 \cdot 82}{3 \cdot 75}} \times 80 = 55 \cdot 73$ revolutions.

Question 126.—In an explosion of gas at 70° F., what would be the difference of expansion in volume, the combustion taking place at 9,564°?

A gas expands $\frac{1}{460}$th of its volume at 0° F. for each degree it is raised above that point under a constant pressure.

Therefore any 460 volumes at 0° become 460 + 70 = 530 volumes at 70°, and at 9,564° the volume would be 460 + 9,564 = 10,024. In other words, the relative volumes occupied by a gas at the respective temperatures of 70° F. and 9,564° F. will be represented by the figures 530 and 10,024, so that the difference of expansion in volume is as 530 : 10,024 or as 1 : 18·9, that is, every cubic foot of the explosive mixture at 70° becomes 18·9 cubic feet at 9,564°.

Question 127.—How would you ventilate a mine giving off CH_4 and CO_2 freely, and what kind of airways would you adopt, and what proportion should they be to one another?

I should make no distinction in ventilating a mine giving off CH_4 or light carburetted hydrogen gas at one portion and CO_2 or carbonic acid gas at another, except not allowing the air from the one portion to return so as to mix with the other. By the first general Rule of the Mines Act, 1887, we are bound to provide an adequate amount of ventilation in every mine to dilute the noxious gases so as to render them harmless. Both CH_4 and CO_2 are noxious gases, and the means of diluting them the same, viz., by providing and coursing round the districts of the mine such quantities of pure air as to ensure the rendering harmless of these gases. Air containing 3 or 4 per cent. of CO_2 is unfit to be breathed, and therefore we must be sure that a district giving off that gas has air in the proportion of 100 to 3 of the quantity of CO_2 given off, or about 33 to 1, and as also CH_4, when mixed with the air in the proportion of 1 in 30, will show a "cap" on the flame of a lamp, we shall require 30 parts of air in any district giving off CH_4 to every 1 part of CH_4 so given off. I do not mean to say that it is sufficient merely to dilute these gases so that the one may be just in a breathable state and the other be just beyond the point of its showing a cap; the figures are given merely to show that the relative quantity for this purpose is nearly the same. Therefore the proportion of air required in the different districts of a mine, taken in consideration simply of these gases, will be practically the same, but other considerations, such as the number of workmen employed in each, &c., may affect the relative quantities we should send into

CHAPTER XIII.

THE PRIESTMAN OIL ENGINE: PETROLEUM AND NATURAL GAS.

Application of the Oil Engine to Mining—Description of its Action—Cost of Working—Its Advantages in certain Positions—Rules for the Prevention of Accident from its Use—Particulars of its Application as a Hauling Engine—Different Nature of Work Performed by Oil Engines—Quality of the Oil Used—Character of Petroleum—Geological Formations in which Found—Possibility of a Boring first Tapping Petroleum, Water, or Gas—A Theoretical Mode of Production—Professor Mendeleeff's Theory of Petroleum Formation—Comparison of Manufactured and Natural Petroleum—Possibility of the Exhaustion of Coalfields and Continuance of Oil-fields—Chemical Composition of Petroleum—Natural Gas in Commercially Profitable Quantities—Where Found—Particulars of the Findlay Gas Well—Increase of Capital Employed in the Use of Natural Gas—Shrinkage in its Supply—Burning the Gas on the Surface of River Water and on the Ground—Analysis of Pittsburg Natural Gas—Its Occurrence in the United Kingdom—Petroleum in Europe with the Number of Wells bored and their Depth in the Baku Oil-field—Known Oil Regions of the United States, Canada, and Mexico—Number of Wells Bored and their Average Depth in America—Oil-fields of South America, Australia, New Zealand, North Africa, South Africa, Persia, Burmah and India—Petroleum in China, Sumatra, Java, Borneo, and Japan.

THE PRIESTMAN OIL ENGINE.

During the past few years the use of mineral oil as a source of power has increased considerably, and oil engines are rapidly gaining ground among colliery proprietors for pumping, haulage, winding and other purposes.

Messrs. Priestman Bros., Ltd., of Hull, may be said to have been the first to introduce an engine which works satisfactorily.

Fig. 423D is a drawing of this engine. The oil is placed in the tank, Y; it is then put under a pressure of air by working the hand pump lever, D, after which the stopcock is turned so that the air and oil pass along the copper pipes to the heating lamps, E, E. One, two or more heating lamps are used according to the size of the engine. A three-way cock, the lever for which is at A, is fixed so that the oil and air can be turned either to the heaters or through small copper pipes to the spray-maker. As the oil will not ignite in the working cylinder, Z, unless the temperature is raised, the vapouriser, O, is heated for a few minutes by a flame from the heaters, E, E, which are kept alight by working the hand pump lever, D, or by other means which can be employed in order to save the manual labour of pumping. When the vapouriser is sufficiently heated, the stopcock is turned by means of the cock lever, A, so that the oil and air are shut off from the heaters, E, E, and admitted through pipes to the spray-maker, and so into the central passage of the vapouriser. The spray-maker and regulator are of special construction. The air and oil come into contact in a nozzle, both being forced through by the air-pressure in the oil tank and pass into the vapouriser in the form of spray. A further supply of air is drawn through small holes in the cover of the vapouriser. Each charge of oil mixed with air is correctly regulated by means of the governor in proportion to the amount of work to be done, and the valves are so carefully adjusted as to ensure constant proportions of air and oil and great regularity in the running. During the working of the engine the air pressure is maintained in the oil-tank by means of an air-pump worked by an eccentric from the

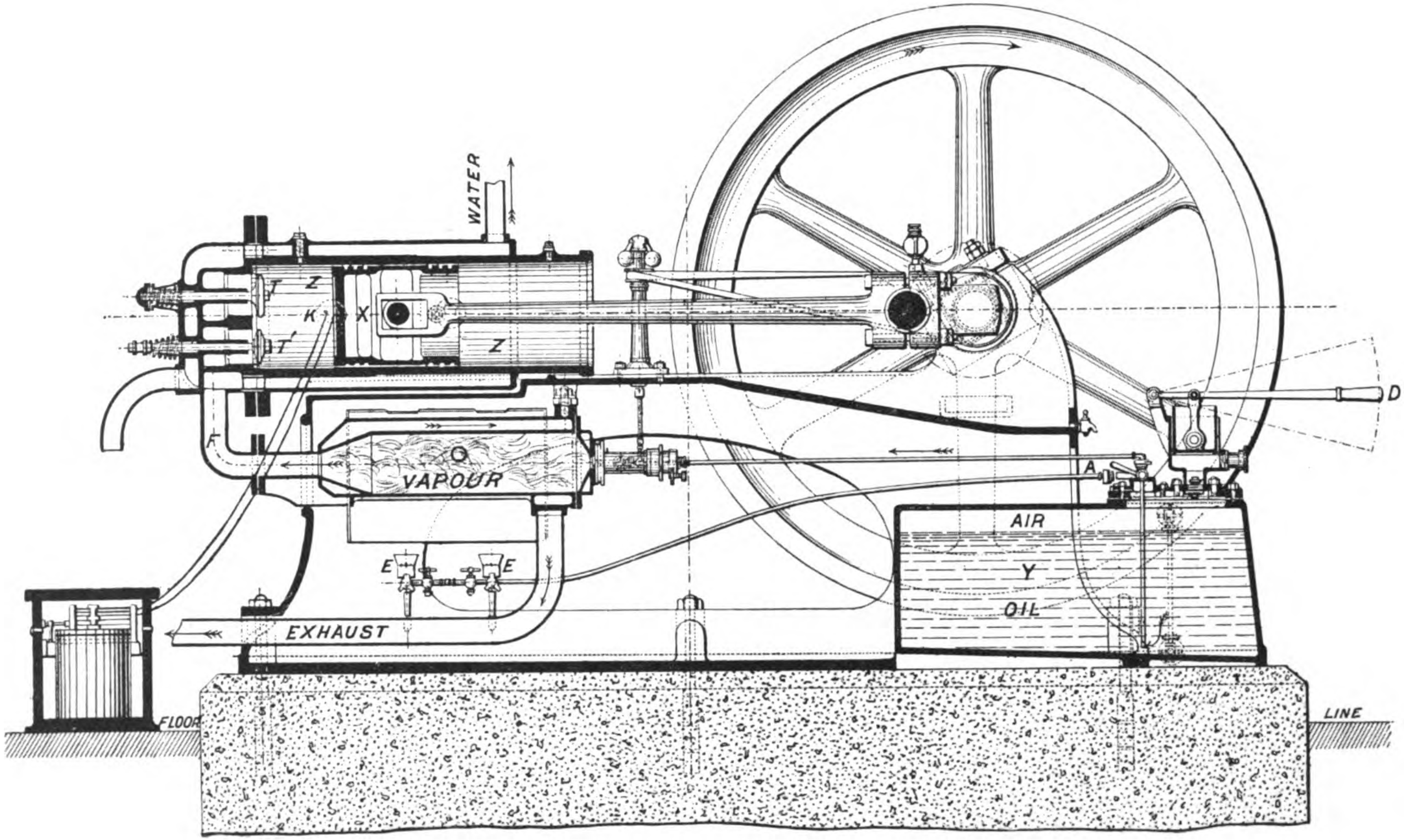


Fig. 423D.—Sectional Elevation of the Priestman Horizontal Oil Engine.

The temperature at which the oil shall flash has been settled at 100° Fahr. for the open test, but as the Scotch shale oils of high flashing points are admirably suited for these engines, no difficulty need be experienced in complying with the rules laid down by the Board of Trade. The oil used in the Priestman engine is heavy and has a high flashing temperature. It does not require the attention of specially skilled men, no expense is necessary for boiler or in the erection of chimney, the fuel is cleanly to handle, and the engine can be started in a few minutes.

It is desirable in underground workings that a good ventilation should be maintained, and that where it is possible the exhaust gases from the engine should be conveyed direct to the main return airway. The gases will then quickly mix with the return air and be reduced in temperature; any vapour passing into the airway from incomplete combustion in the engine will thus be rendered inoffensive and the smell of petroleum be carried away by a good current of air before it becomes objectionable. The products of combustion vary slightly when the engine is working under varying loads but are inexplosive. Under ordinary working conditions there is no risk of petroleum passing unconsumed into the exhaust.

A large number of these engines have already been put to work in underground workings in England and Scotland, New South Wales, and Victoria. In a mine in South Wales, the engine was coupled up direct to a set of three-throw pumps 6 inches × 12 inches, and the water pumped a distance of about 300 yards.

A set of pumps in the county of Durham, which were originally worked on the tail-rope haulage system, were replaced by a Priestman oil engine and a double-acting pump. These were placed 2,400 yards from the shaft, and at a point 165 vertical feet in the dip. The engine is of 5 nominal H.P. and drives the pump by a belt. The pump has a 6-inch barrel and a stroke of 18 inches. The water is forced a distance of 1,320 yards a total height of 72 feet.

Mr. W. H. Wain, speaking before the North Staffordshire Institute of Mining Engineers on the 21st March, 1892, said:—

"At the Midland Coal Co.'s colliery some time ago, having no available power within a thousand yards of a point where it was desired to drive down a pair of exploring places, it was decided to have one of the Priestman oil engines. The work done was the haulage of the coal from the bottom of the dip, and water from the different points at which it was caught. The distance hauled was 491 yards, the weight being about 37 cwts., and the average time 5 minutes 55 seconds. The total cost per hour for stores and wages was 2*s.* 1·58*d.*"

The oil engine, slightly altered, has been largely employed for rock drilling in ironstone mines in the Cleveland district, this having been found to effect a great saving, both in first cost and in maintenance, when compared with hand labour, compressed air, or electricity.

From the fact that the British and Norwegian Governments have placed repetition orders for this engine for printing and fog-signalling, that it is in use in several of the Lighthouses of Great Britain, and is adopted by the Chinese and United States Governments for the same purpose, and further that the Governments of Victoria, New South Wales, New Zealand, and Spain, are using it for electric lighting, pumping, sawing, &c., it may fairly be assumed that it is perfectly satisfactory.

PETROLEUM AND NATURAL GAS.

Petroleum, or rock-oil, has been found in many parts of the world and probably exists in many others. In character it resembles both naphtha, which is more fluid, and asphaltum, which is solid. None of them are soluble in water or in with such force as at first to reach a height of 40 or 50 feet above the surface, this being diminished as the force grows less. In a discharge of this kind the yield of the well is large for a time, during which its commercial prosperity is at its greatest. After a short time the ejection ceases, but the oil may afterwards be pumped from the well and yield a steady supply for many years. If the gas is first tapped by the boring it is allowed time to escape, after which the oil can only be got by pumping. If, however, the gas is present in a considerable quantity, it is not allowed to escape, but is piped away for use.

In some instances the gas has been securely retained and conveyed a distance of a quarter of a mile in pipes, and has then been utilized to drive a steam engine just as though it were compressed air or high pressure steam produced on the spot.

The American oilfields often occur where the strata are very flat over extensive areas, and where the absence of faults favours the circulation of imprisoned fluids. Where there are great flat anticlines and synclines it is easy to understand that the wells drilled along the crests of the former yield gas, those in the troughs only water, while a zone between them may be expected to yield oil. The effect of boring into the different zones formed by flexures in the strata is similar to that described for borings in a fissure, but occurs on a larger scale. It is thought that although the oil is confined to particular strata, the beds are not uniformly porous, and a well may strike a close-grained rock through which no fluid can find its way.

Bore-holes drilled on an oil-belt whose direction is known vary in production according as the strata are more or less disturbed. A productive oil-well may exist close alongside one yielding comparatively little.

The crude petroleum is useful for some purposes, but it must be distilled in order to convert it into other commercial products; these again often requiring purification.

From the first distillation crude naphtha, and burning and paraffin oils are obtained, but sometimes the crude naphtha is got rid of before distillation begins. Further distillation causes the crude naphtha to yield gasolene, benzine, and refined naphtha. The burning oil usually requires treatment and purification before it is fit for use. The presence of naphtha, even in small proportions, lowers the flashing temperature of the burning oil, and, in larger proportions, renders it still more dangerous in use.

It has been assumed that petroleum is of the nature of coal, and has been formed out of the turpentine oils and resins of the coniferous trees of primeval forests which entered into the composition of coal seams, or out of the remains of marine animals, and that in course of time the deposits will be exhausted. No satisfactory theory has been suggested to account for the decomposition, at great depths beneath the surface, of vegetable and animal remains, and no explanation as to whether or not the process is still going on.

Professor Mendeleeff, a distinguished Russian chemist, has propounded the theory that petroleum is constantly being formed by the action of water on metallic deposits in the heated interior of the earth. Professor Mendeleeff states, that the oil-bearing regions generally lie parallel to mountain ranges, such as the Caucasus, in Russia, whence the greatest European supply comes, the Alleghanies, in the United States and Canada, and the Andes in Peru, and that petroleum does not appear to belong to any particular geological formation. In Europe it usually occurs in rocks of the Tertiary formation, while in the United States it is found in the Devonian and Silurian periods. He also shows that, on account of the volatile nature of rock-oil, it could not have been transferred from a distance, like many other deposits, but must have been formed very near the spot where it is found. The formation of mountain chains and terrestrial disturbances have fissured the earth's crust, making openings through which surface water penetrates to the interior; there, coming in contact with the glowing metals and their carbides, chemical reactions have been set up, resulting in hydrogen taking up the carbon to

The use of natural gas as fuel at steelworks and in rolling mills in the United States was for a time increasingly prevalent; in 1892, owing to a smaller supply, its use declined. In that year there were 74 mills wholly or partially depending upon this product, this being a reduction of 30 in two years, notwithstanding the development during that period of the gas-bearing region in the central part of Indiana, where several iron and steel industries have arisen. Bituminous coal, gas coal, and petroleum are used as substitutes at those works which have been obliged to abandon the use of natural gas. Fresh bores are continually being made in order, if possible, to maintain a gas supply.

From the bed of a diminutive river called the Cat Fish Run, in the petroleum regions of Pennsylvania, the natural gas rises in bubbles, which explode on the surface. Not many years ago, some visitors to the neighbourhood, by way of experiment, floated some burning rags down the stream towards the gas, which then ignited with an explosion. Other instances have occurred in which the gas has been lighted on the surface of the water at night, giving the magnificent appearance of a sea on fire.

Such a fire was observed in 1845 on a small scale on the river Wear, near the city of Durham, and from its singular appearance attracted much notice. The *Durham Advertiser* in August of that year relates that at certain points in the river near Framwellgate Bridge on perfectly calm days large numbers of air or gas bubbles issued from below as though the water were boiling. A gentleman in the neighbourhood, thinking that it might be due to the escape of light carburetted hydrogen through a fissure from a seam of coal under the bed of the river, determined to make an experiment. For this purpose, choosing a day when there was no wind, he moored a boat alongside the disturbed surface of the water and set fire to the gas. Subsequently a pipe with a bell-mouthed end was fixed over the supposed fissure in the river bed, and the gas collected and conveyed through it to a reservoir floating on the surface. The receptacle was provided with a burner and glass chimney. A brilliant jet of flame could then be commanded at pleasure. There were several other places beside the one at which the experiment was conducted, the total escape from which amounted to many cubic feet of gas per minute. In favourable weather these could be ignited by holding a light close to the water until the ascending gas came in contact with it. In the absence of wind, rain, &c, separate clusters of gas jets could thus be lighted on the water, the extreme clusters being distant about 100 yards from one another. This series of burning jets scattered about in irregular patches on the surface of the water when seen from the bridge at night looked like a burning river. No coal workings were known to exist within two miles which could cause a subsidence and the occurrence was probably due to a large natural accumulation of firedamp at a fault, which extended to the surface with interstices at the lines of broken strata.

In the neighbourhood of Baku, on the shores of the Caspian, gas issues from the ground in great abundance, and over large tracts, as though the soil were completely impregnated. The land glare of the fires by night is an amazing sight, as may well be imagined. In some instances, the flame is utilized for domestic purposes.

The analysis of the gas at Pittsburg is as follows:—

Marsh gas	67 %
Hydrogen	22 ,,
Ethylic hydride	5 ,,
Nitrogen	3 ,,
Olefiant gas	1 ,,
Carbonic acid, carbonic oxide, oxygen	2 ,,
	100

Natural gas in the United Kingdom occurs only in limited quantities. That which is found in coal mines has been fully dealt with elsewhere in this work. Gas given off at the Wallsend Colliery was burned on the surface for several years, and in many other collieries, "blowers" of firedamp have been conveyed to the surface and used there. In two of the three series of experiments made with safety lamps in explosive mixtures by the Royal Commissioners on Accidents in Mines, 1886, two natural gases were made use of, one given off at a powerful blower at the Garswood Hall Colliery, near Wigan, and the other at the Llwynypia Colliery, South Wales. In the former case, the gas issued from the Wigan 9-foot seam which had just been reached in the newly sunk shaft, and was quite unworked at the time of the experiments; in the latter, the gas was obtained from the sandstone about 60 yards above the 6-foot seam of the Rhondda valley, and piped to the surface. Gas occurs in the jet rock of the Upper Lias in East Yorkshire, together with some heavy liquid bitumen; and it also finds its way down into the ironstone mines worked in the Middle Lias. A blower was burned for over 20 years in the Grag Hall ironstone mine, a few miles south-east of Saltburn. In some of the first borings for salt at Middlesborough, gas was found, and oil at Seaton Carew, both being as it is believed, from the upper beds of magnesian limestone. A deep boring at Port Clarence was continued 150 feet below the salt, in order to prove the magnesian limestone. The limestone yielded traces of bitumen, and there was also a constant escape of gas, which contained 83·2 per cent. of hydro-carbons, and 16·8 per cent. of nitrogen. Gas was found afterward at the same place in greater quantities under considerable pressure, together with some petroleum.

Petroleum has also been found at Worsley (near Wigan), and West Leigh in Lancashire, in Shropshire, in Derbyshire, in North Staffordshire, and in Lanarkshire, but never in sufficient quantities to justify its working. The latest discovery is that in Somersetshire.

In Scotland, the working of the bituminous shales of the coal-measures is a very important industry. The oil shales are found in calciferous sandstone measures situated between the carboniferous limestone measures and the old red sandstone. They occur in small basins in the centre of Scotland. The seams of shale vary in thickness and quality, and are usually won by mines commencing at the outcrops on the surface and following the seam to the full dip. The thick seams are worked by pillar and stall method, the thin ones by longwall. The small shale which passes through one-inch riddles is stowed underground. Fire-damp is given off in the shale mines, which, however, are worked with open lights.

On distillation the shales yield naphtha, sulphate of ammonia, and crude oil. The crude oil when treated in the refinery produces burning, lubricating, and paraffin oils.

It has been suggested that although the strata of the United Kingdom are as fossiliferous as other regions in which petroleum and gas are largely produced, the palæozoic strata have been subjected at various ages to repeated foldings and denudations, leaving exposed edges which have facilitated the escape of any gas or petroleum which might else have been retained.

In Europe, petroleum has been found in Germany, France, Italy, West Sweden, in Spain in small quantities, Austro-Hungary, Hungary, Roumania, and Moldavia. There are also several deposits in Russia, beside that of Baku, which is one of the largest and oldest known. The oil is here found nearly colourless in soft sandstones interstratified with impervious clays of different colours, and requires no purification before being burned in lamps. Out of 1,600 square miles of oil-bearing territory, only about 5 square miles have been developed. Up to 1890 some 500 wells had been bored on that area, the deepest being about 850 feet, while the average depth is about 500.

The known oil-fields of the United States are situated chiefly in New York, Pennsylvania, Ohio, West Virginia, Kentucky, Tennessee, and California. Between 1860 and 1890, about 50,000 wells had been bored, some being 2,500 feet in depth, while the average was 1,500 feet. The bore-holes vary in size, but are usually from 3 to 4 inches in diameter. At the 1893 World's Fair at Chicago, the Exhibition authorities used petroleum exclusively for the boilers there. It was brought from oil-wells in Ohio and Indiana. The oil-fields of Canada are near Lake Erie, and also on the coast and near the Rocky Mountains. The most important of these are in the valley of the Mackenzie and Athabasca rivers, and, according to the report of the Senate of Canada about 1886, they constituted the largest petroleum field in the world. About 10,000 square miles of this petroleum territory has been marked out as a reserve to be constituted as a Crown domain. The whole area is quite undeveloped. In Mexico, there are large deposits which have not yet been extensively worked.

Petroleum occurs in many of the West Indian Islands.

Turning to South America, Venezuela and Peru have very extensive oil-fields; although no great efforts have yet been made to develop them. Those of Peru are likely to be of great importance. Petroleum occurs also in the Argentine Republic, and is said to have been found in Bolivia and New Granada.

Oil-fields have been discovered in South Australia on the banks of the Coorang, a little to the north of Salt Creek, and in New South Wales.

In New Zealand, there are similar fields undeveloped near Poverty Bay, Auckland, and in the Taranaki district. The latter are interesting from the fact that the oil bubbles through the pulverised iron ore that forms the beach along the west coast.

In North Africa, petroleum has been found both in Egypt, and, according to report, in Algiers.

In South Africa, there are oil-fields in the Transvaal and in the Orange Free State. Little is known of these as yet, and there is room for speculation as to the future of Africa as an oil-producing country.

The Persian oil-fields are situated in the valley of the river Karun, but little has been done to develop them.

In Burmah and India petroleum occurs, and may some day lead to the development of important industries there.

It is also stated to have been discovered in China, and is known in Sumatra, Java, Borneo, and other islands in the eastern seas.

Petroleum deposits have also been worked in Japan.

The oil-bearing regions of the world, therefore, are vast and practically inexhaustible, even apart from the theory of constant production. The use of paraffin, both in a crude state and greatly purified, has made enormous strides during the last few years for public and private illumination, and for fuel and manufactures. The application of mineral oil as a fuel for ocean-going vessels, locomotives, and stationary engines is likely to undergo extensive development.

With the chain, on the surface, 10 or 11 arrows also are used, the latter number being preferable, but they are not often used in underground measuring as they will not stick in the floor, and the custom is to mark the end of a chain with chalk on the floor or the rails. Care is required to measure straight between the two marks set up, which on the surface are usually rods, and underground must be lights.

The best land chains are of steel, and made light and strong. Fig. 425 shows a land chain and arrows. The chain, whether made of steel or iron, has long links formed by turning up the ends of a length of wire. Three small oval links are placed between each pair of long links. Three interval links are found to cause the chain to *kink* less than when only two are used. Each oval link is sawn through at the meeting line, which is brought up on one flat side of the oval in bending it from the wire. The small links are used for the adjustment of the chain, as they may be closed to shorten, or forced open to lengthen it. There are generally four *swivels* in the length of the chain, two of which are at the handles: these prevent the chain from becoming twisted in turning the handles over in use.

The length of an ordinary measuring-chain is always tested, and, if necessary, adjusted by a standard chain before it is used in a survey of any importance.

Standard chains are of the same form as the ordinary steel chain, but all the links are hard-soldered after being adjusted link by link.

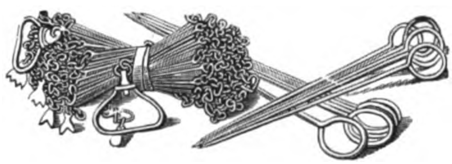


Fig. 425.—Land Chain and Arrows.

Standard chains are used at collieries only for testing and adjusting ordinary chains.

Horizontal distances only are shown on the plan, and these will not be the actual distances measured where the ground rises or falls. On the surface, approximate horizontal measurement may be obtained by holding one end of the chain up, so as to keep it in a horizontal position, and plumbing from the handle to the ground; but where the ground is very steep, it is impossible to hold out a whole chain-length in this way, and it has to be done by using a part of the chain, say 40, 30, or 20 links at a time, according to the fall of the ground. It is, however, difficult to tell when the chain is being held horizontally, and it is much better instead of adopting this method, to measure the distances along the surface of the ground; and by taking the angles of elevation or depression over the several inclined parts of the line with the instrument used for measuring the angles, the correct horizontal distances may be computed. The underground measuring admits of no other method. Tables prepared from calculations, may be obtained, showing this correction for every 100 links for any angle whatever. For measuring the depths of shafts, or in making plans whose areas are required in square feet, or it may be for plans of surface buildings or other purpose, a chain 100 feet in length, divided into links a foot long, is to be preferred. Chains made of steel are lighter to use than those made of iron.

The tape may be made of any length, but those mostly in use are 33, 50, 66, or 100 feet long, and there are different ways of dividing them. The most generally useful one is divided into feet and inches on one side and links on the other, so that it can be used to measure either. In surveying it is used to measure long offsets from the chain, the lengths of buildings, &c., on the surface, where its use is a necessity in addition to the chain, but it is seldom required in underground surveying. Tapes shrink in use, and should be frequently tested by standard measurement.

The Spirit-Level is an instrument used in measuring the vertical distances between different places. It consists of a glass tube which is not quite cylindrical in form, but having its diameter largest in the middle and decreasing slightly and with great regularity from the middle to the ends. The tube is nearly but not quite filled with spirits of wine, thus leaving in it a bubble of air, which rises to the highest part of the tube and which, when the instrument is adjusted, has the two ends of the bubble equally distant from the middle. A scale is generally scratched on the glass to guide the operator in adjusting the instrument. This spirit-level is fixed to the telescope by a joint at one end and a capstan-headed screw at the other so as to raise or depress it for adjustment.

The oldest form of surveyor's level is that termed the *Y-Level*, a modern form of which is shown in Fig. 426. It is but little used in Great Britain now, although it has much to recommend it, because of the facility its construction affords of verifying its adjustments before commencing work. It receives its name from the Y-formed bearings YY″ which support the telescope. These in turn rest upon the limb L. The telescope has two circular collars soldered upon it at positions exactly corresponding with the Y's. The collars are turned perfectly cylindrical,

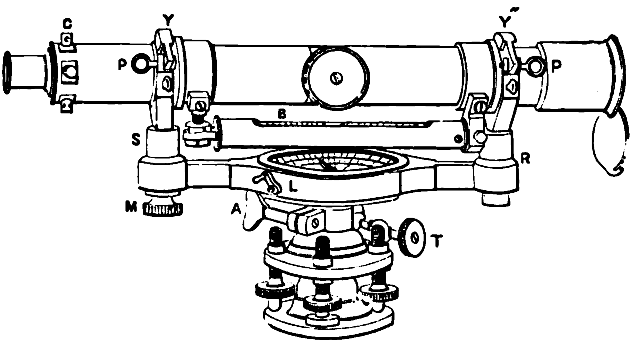


Fig. 426.—Surveyor's Y-Level.

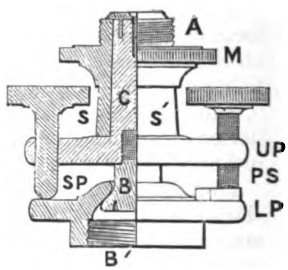


Fig. 427.—Section of Parallel Plate and Vertical Axis—Arrangement of Y- and other Levels.

and parallel on the surface with the axis of the telescope, and ground in a gauge-plate to exact size, so that the telescope may be turned end for end in the Y's without altering the lineal direction of its axis in reversing it. The telescope is prevented from shifting longitudinally in its Y's by a pair of flanges placed on the inside of the collar-pieces.

The Y's are erected upon the limb, to which they are each fixed firmly by a clamping-nut at one end R, and a milled-head clamp at M. The telescope is held down by strap-pieces, each of which has a joint at one end and a loose pin at the other PP. The pin is attached to the instrument by a piece of cord and to a loop formed in its head, so that it dangles, but cannot be lost, when out of use. At the top of the inner side of the strap-piece under YY″ a piece of cork is inserted in a cave. The cork, by its elasticity, keeps an equal but light pressure upon the collar of the telescope. It will be seen that by the above plan of holding the telescope, it is so far free that it may be revolved on its axis, by which perfect adjustment of the diaphragm may be made in any direction.

The general construction of the vertical axis and parallel plate of the Y-level is shown in Fig. 427, the left-hand side being a half-section. A is a screw by which the parallel plate-foot is attached to the limb of the instrument; M, a large milled head, by means of which the screw can be brought up firmly to its collar; SS′, the socket which is ground to fit the cone C; C forms a part of the upper parallel plate UP; B, a ball pin which screws firmly into C; LP, lower parallel plate, part of which forms the ball socket, so that the whole instrument rocks about the ball B as a centre by the action of the parallel plate-screws PS; B′, female screw for fixing this part, which is called altogether the parallel plates, to the tripod head. In the old Y-level there was usually a clamping-screw upon the axis for slow motion, which generally caused a strain upon it. Modern instruments are made without clamping-screws. The parallel plate-screws are tapped, that is, have female threads cut into the upper plate, UP, and their points press the lower parallel plate, LP, at certain points, there being a stop-piece placed round the point of one screw to prevent rotation.

The diaphragm of the telescope, Figs. 428 and 429, is formed of a stout disc of brass, having a central hole of about ·3 inch diameter. Upon the side which is placed next the eye-piece the hole is brought to a thin edge by an internal bevel or countersink, which leaves the hole much larger at its off-surface, Fig. 429. The disc is held in its place and adjusted by four capstan-headed screws, termed collimating-screws, two of which are shown in section as C C′, their points being tapped into the rim of the diaphragm frame P. The screws are placed through a stout collar. The diaphragm has generally three spiders' webs crossed in the

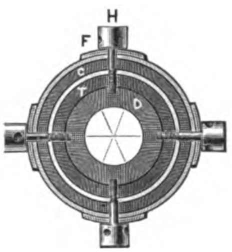


Fig. 428.—Elevation of Diaphragm.

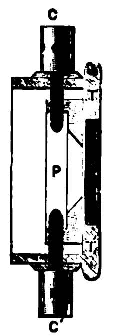


Fig. 429.—Section of Diaphragm.

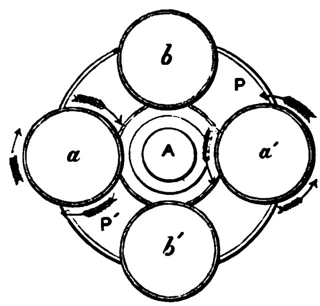


Fig. 430.—Diagram Plan of Parallel Plate Screw Milled Heads.

manner shown in the centre of Fig. 428. The eye-piece is screwed into the thick plate, Fig. 429, and adjusts to the focus of the webs. The webs are fixed to lines drawn on the diaphragm.

To set up the Y or other level with parallel plates, the tripod stand is opened out to form a firm support for the instrument; the toes of the legs are then each separately pressed into the ground sufficiently to make the instrument stand quite firmly. The instrument is then screwed down tightly upon the tripod head.

The eye-piece is then adjusted by sliding it gently in and out until the webs can be seen most distinctly. On a bright day a white pocket-handkerchief or other light covering may with advantage be thrown singly over the object-glass, to prevent any confusion from objects in the field of view during the focussing of the eye-piece.

For the setting-up adjustment of the telescope, it is brought to be directly over one pair of parallel plate-screws, PS, Fig. 427. The milled heads only of these screws can be seen in the diagram plan, Fig. 430, *a a′* being the opposite pair over which the telescope will be assumed to be at first placed. The level tube is now brought to adjustment by bringing the bubble to the centre of its run by means of the parallel plate-screws, *aa′*, by taking the milled heads of these screws, one between the balls of the thumb and forefinger of each hand, and rolling them simultaneously, the one in one direction and the other in the reverse. This action gives the axis of the telescope motion in one direction or the other. Thus, by the adjustment nearly always disturbs the other, the whole of the process may have to be repeated until the instrument reverses in any direction; but this for final adjustment is better deferred until the adjustment of the level tube, to be next described, has been made.

To adjust the level tube, the telescope is placed as before over an opposite pair of parallel plate-screws, and these are adjusted until the bubble is in the centre of its run. The telescope is then turned half a revolution, so that it is placed over the same pair of screws in the reverse direction, and the displacement of the bubble from the centre is now noted. The capstan-headed bubble-screws at the end of level B, Fig. 426, are then adjusted to one-fourth of the difference observed, and the parallel plate-screws are adjusted for the other fourth, so that by these two adjustments the difference of the run in two positions, obtained by sighting a staff, is bisected. The same process is repeated over the second

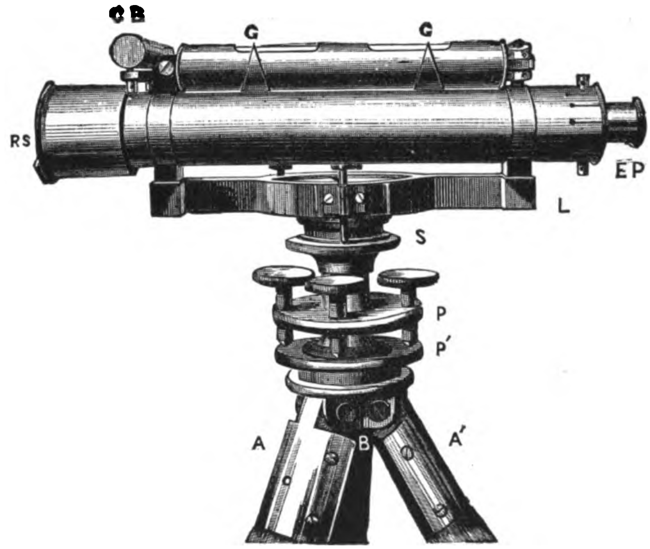


Fig. 431.—Dumpy Level.

opposite pair of parallel plate-screws. If this be very carefully done with a correctly-divided bubble, the Y's of the telescope may be opened out and the telescope be reversed end for end in its Y's, and the bubble will remain true. As a check, however, the adjustments should be all gone over a second time.

A modern form of *Dumpy Level* is shown in Fig. 431. The telescope carries a ray-shade, RS, at the object-glass end, to work in the field to eastward or westward, facing a low sun. The eye-piece, EP, is adjustable to the webs in the telescope by pressure in and out. Two straps or bands are accurately fitted and soldered round the tube of the telescope, one of which carries a hinge joint, and the other a pair of locking-nuts to support the level-tube, GG, which at the same time permit its adjustment. The level casing-tube has two three-quarter bands, which slide upon it, pointed at one end, GG; these adjust to the length of the bubble for changes by temperature. The lower part of each strap-piece is left a solid block of metal, to give very firm support to the telescope as it rests upon the limb, L, beneath. The limb may be either a casting with a socket-screw only in its centre, or a compass-box may be formed in the centre and the socket-screw be placed under this, as it is shown in the figure. The attachment of the telescope support to the limb is made by three screws, two of which draw the limb down and one in the centre presses it upwards, as shown in the section, Fig. 432 —CC′, telescope; TT′, drawing-screws; P, pressing-screw. By this means firm adjustment may be made either by raising or lowering one end of the telescope, as also by a lateral rocking motion should the web or bubble not be quite to position. The general construction of the vertical axis, parallel plates, tripod head, and tripod is the same as that of the Y-level already described. The cross-level, CB, Fig. 431, at right-angles to the principal level, allows the setting-up of the instrument to be completed approximately, without turning the level a quarter revolution backwards and forwards several times during the operation, as was necessary in the setting-up of the old form of Y-level. Modern Y-levels are, however, made with cross-levels.

The diaphragm of the dumpy-level is generally webbed with two vertical webs and one horizontal. In use, the staff is viewed through the vertical webs, and these

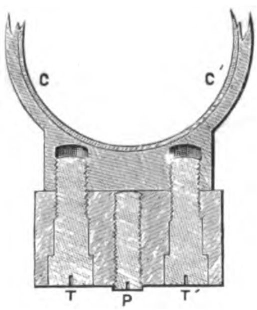


Fig. 432.—Attachment of Telescope Block to Limbs.

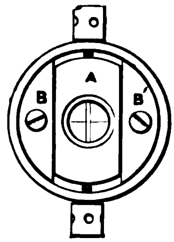


Fig. 433.—Diaphragm of Dumpy Level with Webbed Stop.

indicate whether it is held upright sideways, although they cannot be used to detect a backward or forward leaning of the staff. The upper margin of the portion of the horizontal web between the two vertical ones is the index of level to which all readings are made, either for adjustment or for reading the levelling-staff in the field. The four-screw adjustment for a level used in rough work with capstan-head screws, shown in Fig. 428, which is necessary for the adjustment of the telescope in Y's, has been abandoned for many years in the construction of the dumpy-level, and given place to that shown in Fig. 433. In this plan there is no lateral adjustment; the diaphragm is carried as a frame in a dovetail slide, and is adjustable by vertical screws only. The figure shows the face of diaphragm BB′ slide-pieces. A is the slide moved by capstan-head screws.

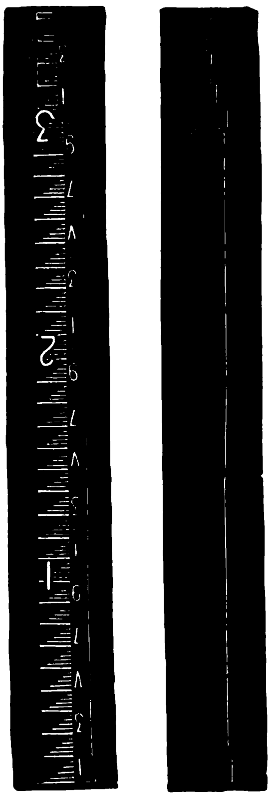
A 12-inch dumpy level should read the ·01 foot on a Sopwith staff at 5 chains with a webbed diaphragm; a 14-inch dumpy should read the ·01 foot at 10 chains.

The Dumpy level sometimes has a circular level placed immediately under the telescope in the place of a cross level.

Fig. 434 shows Messrs. Davis & Son's Dumpy level, with Hoffman patent joint. It has achromatic lenses, and all parts are of gun-metal. Two eye-pieces are provided with the instrument, and a tripod of polished mahogany, the head of which carries a screw about $1\frac{1}{2}$ inches in diameter with coarse thread, which fits into a socket on the lower parallel plate of the level. The three legs shut together in one round staff or may be opened out to stand firmly on the ground, however uneven it is. The telescope has a focussing arrangement, and near the eye-drawn out, and the same band further uncoiled till it reads continuously to 9 feet.

Another form of levelling staff for use in mines has been introduced. In it the figures are painted in dark colours on a background of ground glass or other transparent material. When being used a light is placed behind the staff, and this gives the surveyor a better opportunity of making his observations.

To prevent injury to the staff during carriage, a convenient plan is to have two pads formed of stout ox-hide butt, each pierced with two slots near their ends at the exact distance apart of the width of the staff—Fig. 442. The strap, of calf leather, is passed from one slot round the staff into the other slot, and then passed round the tripod and pulled up tightly and buckled. The pad of course protects the front of the staff from grazing by the friction of the tripod against it. One pad is placed at the top, the other at the bottom of the closed staff and tripod.



A. Front view. B. Side view.

Fig. 441.—Jee's Underground Levelling Staff.

Stanley's spring pads, Fig. 443, form a better protection against the cylindrical tripod pressing against the front of the staff than ordinary pads. In these a hard-rolled German-silver spring covered with leather distributes the pressure, which is thus greatest near the edges of the front, where the staff is strongest to resist. The section of this spring is shown at E, with the general arrangement of the staff S, and the tripod D, fixed upon it by the strap F.

For the entire protection of the staff a leather-bound sailcloth case is very generally used. This may be divided in two compartments for the staff and the tripod, with pads between. The whole case has a neat appearance, and forms a protection from slight bruises and dirt, either in travelling or when set up in an office corner for future use.

A triangular plate of iron, as represented in Fig. 444, may be used with advantage to rest the staff on during an observation. It is trodden down firmly by the staff-holder before he places the staff upon it. In use it gives a certain base to turn the staff upon from fore to back sight, and removes the necessity for the staff to be held continuously in one spot while the level is changed from one position to another.

The staff-level consists of a small circular level, shown in section, Fig. 445, the upper surface of which is formed of a glass worked slightly concave and fixed over a short cylindrical box. The glass is generally burnished into the box and cemented round with elastic cement. The box is nearly filled with spirit from a hole covered by a large-headed screw, which is fitted with an india-rubber washer. The circular level is mounted on a plate with studs. The studs fit in two holes with bayonet slots in the holding-plate, which is attached to the back of the staff. In use the staff-holder has to observe when the bubble under the concave glass is in its centre. A very little practice is required to hold the staff vertically by means of this little contrivance, which only weighs, with its pocket case, about 2 ozs.

fore-sight, the difference of level is called a fall. If in the above case the back-sight read 6·42 feet and the fore-sight 10·17 feet, there would be a fall of $10{\cdot}17 - 6{\cdot}42 = 3{\cdot}75$ feet, which, in other words, would mean that B was 3·75 feet below the level of A. When from the nature of the ground it is not possible to see from one point or position of the instrument the two extremes, a series of simple levels like the one indicated above must be made, the fore-station at each plant of the instrument being made the back-station at the next, and from the combination of all the results is obtained the required difference of level between the desired points.

When it is required not only to find the difference of level between two

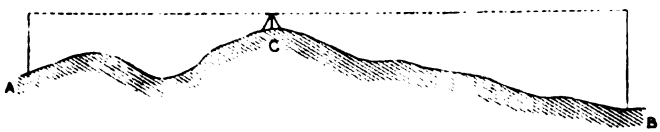


Fig. 447.—Illustrating the Operation of Levelling.

distant points, but to make observations that will enable a section to be drawn, showing the undulations of the ground along some specified route, Fig. 448, the stations must be chosen so that they are at the commencement of each change in the inclination of the ground, the distances between each station must be carefully measured, and these measurements must be reduced to horizontal measurements when plotting the section.

A number of sights may be taken with the instrument from the same position if the ground be very undulating and it is desired to note every change of inclination; in that case, it is usual to call all but the fore and back stations intermediate stations, and book their staff-readings as intermediate sights. It is usual also to assume that the starting point is so many feet (frequently 100)

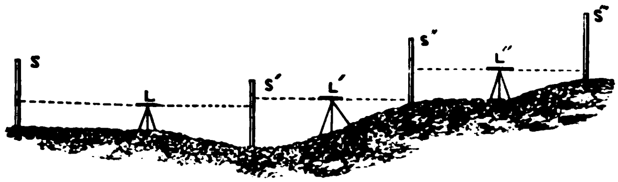


Fig. 448.—Practice of Levelling.

above datum line, unless some well-defined datum line, such as would be obtained from an Ordnance bench mark, is used; the reason for assuming the starting point to be a certain number of feet above datum line being the convenience both of keeping the book and of plotting the levelling.

Ordnance bench marks are the broad arrows cut on stone walls, window-sills, coping-stones, or other prominent and convenient places by the ordnance surveyors. The ordnance plan-sheets of any locality have similar arrow-marks on them at the places shown on the plans where the marks have been cut, and, in addition, a number printed distinctly at the arrow, thus B.M. 562·25, which indicates that that particular arrow is 562·25 feet above mean sea-level at Liverpool. These marks serve as a record by means of which subsidence or elevation of land may be measured. The counties of Lancashire and Yorkshire, *e.g.*, were re-surveyed by the Ordnance surveyors in 1890, and many changes in

LEVELS OF PROPOSED BRANCH RAILWAY AT ——— COLLIERY,
FEBRUARY 10TH, 1886.

Distance in Links.	Sights. Back.	Sights. Inter.	Sights. Fore.	Rise.	Fall.	Reduced Rise.	Remarks.
0	8·59					100·00	On — Railway 150 feet below the Red Lion Inn.
100		7·27		1·32		101·32	
156		10·15			2·88	98·44	
193		8·76		1·39		99·83	
235		9·58			·82	99·01	
280	12·43		6·04	3·54		102·55	
345		9·78		2·65		105·20	
460		7·56		2·22		107·42	
524		6·82		·74		108·16	
596		8·45			1·63	106·53	
620							Crosses hedge.
694		9·52			1·07	105·46	
753		10·89			1·37	104·09	
806	7·56		11·53		·64	103·45	
920		8·21			·65	102·80	
985		9·40			1·19	101·61	
1,050		7·79		1·61		103·22	
1,124		9·65			1·86	101·36	
1,243		11·33			1·68	99·68	
1,297		12·94			1·61	98·07	
1,365		13·84			·90	97·17	
1,500	6·45		14·00		·16	97·01	
1,582		8·42			1·97	95·04	
1,644		9·81			1·39	93·65	
1,700		11·32			1·51	92·14	
1,800			12·47		1·15	90·99	
1,900	7·84		9·42		1·58	89·41	
	42·87		53·46 42·87	13·47	24·06 13·47		
	Difference ...		10·59		10·59		

and 100 − 10·59 = 89·41, the reduced rise.

The following is the same levelling in another form :—

Distance in Links.	Sights. Back.	Sights. Fore.	Height of Instrument.	Reduced Rise.	Remarks.
				100·00	On —— Railway, 150 feet below the Red Lion Inn.
0	8·59		108·59		
100		7·27		101·32	
156		10·15		98·44	
193		8·76		99·83	
235		9·58		99·01	
280	12·43	6·04	114·98	102·55	
65		9·78		105·20	
180		7·56		107·42	
244		6·82		108·16	
316		8·45		106·53	
414		9·52		105·46	At 340 crosses hedge.
473		10·89		104·09	
526	7·56	11·53	111·01	103·45	
114		8·21		102·80	
179		9·40		101·61	
244		7·79		103·22	
318		9·65		101·36	
437		11·33		99·68	
491		12·94		98·07	
559		13·84		97·17	
694	6·45	14·00	103·46	97·01	
82		8·42		95·04	
144		9·81		93·65	
200		11·32		92·14	
300	7·84	12·47	98·83	90·99	
100		9·42		89·41	
1,900	42·87	53·46 42·87			
Difference being a fall of ...		10·59			

Starting point above datum ... 100 feet.
Deduct fall between start and finish 10·59

Finishing point above datum ... 89·41

Another form, requiring only four columns, is as follows:—

Distance Links.	Level Readings.	Reduced Levels.	Remarks.
		108·59	
0	8·59	100·00	On —— Railway, 150 feet below the Red Lion Inn.
100	7·27	101·32	
156	10·15	98·44	
193	8·76	99·83	
235	9·58	99·01	
280	6·04	102·55	
	12·43	114·98	
65	9·78	105·20	
180	7·56	107·42	
244	6·82	108·16	
316	8·45	106·53	
414	9·52	105·46	At 340 crosses hedge.
473	10·89	104·09	
526	11·53	103·45	
	7·56	111·01	
114	8·21	102·80	
179	9·40	101·61	
244	7·79	103·22	
318	9·65	101·36	
437	11·33	99·68	
491	12·94	98·07	
559	13·84	97·17	
694	14·00	97·01	
	6·45	103·46	
82	8·42	95·04	
144	9·81	93·65	Starting point above
200	11·32	92·14	datum ... ... 100 feet.
300	12·47	90·99	Deduct fall between
			start and finish ... 10·59
	7·84	98·83	Finishing point above
100	9·42	89·41	datum ... ... 89·41

1,900	53·46	Total Fore-sights.
	42·87	Total Back-sights.
Difference being a fall of ...	10·59	

either addition. The Height of Instrument column is dispensed with, and the figures really forming the height of the instrument are entered in the Reduced Level column. These are not to be plotted, but are merely used as a basis each time for ascertaining the levels of other stations which have to be plotted. There is little fear of confusing these figures with those in the Reduced columns, which have to be plotted, as the line drawn immediately over them, together with the blank space in the Distance column alongside, sufficiently distinguishes them.

The plotting of the levelling just given will be shown later on, after plotting instruments have been explained.

A very interesting pamphlet on "Hints on Levelling Operations, as applied to the reading of distances by the law of perspective, and the saving thereby of chainmen in a level survey," by W. H. Wells, C.E., was published in 1879 by E. & F. N. Spon, from which the following remarks are drawn.

The image of the staff, or any portion of it, seen in the telescope of a level will diminish as the distance the staff is moved from the telescope increases, according to the laws of perspective.

If when the staff is held, say, 100 feet from the telescope, the portion of the image of the staff contained within the space A B, Fig. 450 (being the horizontal

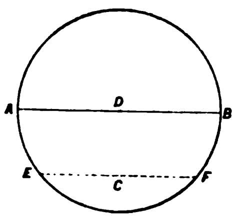


Fig. 450.—Shifting Arrangement in the Diaphragm of a Telescope.

cross web), and C (being the lower edge of the diaphragm), be 1·50 foot, a datum is obtained from which can be ascertained any distance that the staff is afterwards moved, by reading the portion of its image enclosed within the vertical space D C. For instance, if the staff be now moved, and, on directing the telescope to it, it is found that the portion of the staff image contained within the space D C is 2·10; the distance to the staff will be found by a simple proportion sum, thus :—As 1·50 : 2·10 :: 100 : 140; that is, the staff in its new position is 140 feet distant from the telescope.

Instead of using the lower edge of the circular diaphragm, a more accurate outline of a portion of the staff image is obtained by fixing a second horizontal web either near the bottom or top of the diaphragm, as shown dotted at E F.

If this second web is a fixture in the diaphragm, the surveyor must ascertain his own datum at a distance of 100 feet, by carefully measuring out that distance on the ground, the staff being placed at one extremity and the telescope at the other, and then reading the portion of the staff image enclosed between the lines A B and E F. A note of this reading is made in the survey book, and all further levelling operations may be proceeded with without using the chain. All the surveyor has to do after reading the staff in the ordinary way is to read and enter in a proper column in his survey book the portion of the staff image contained between the lines A B and E F. The distances can be worked out after the day's operations are finished, and this is done by a series of proportion sums.

The form of book must necessarily be altered to suit the particular method. The following would appear to be quite clear, although it is open to the objection of having many columns.

Datum distance, reading, say, 1·50 of the staff image = 100 feet.

	Distance Readings.				Level Readings.						
	Reading of 2nd cross web.	Back.	Fore.	Distance in feet worked out.	Back.	Inter.	Fore.	Rise.	Fall.	Reduced Rise.	Remarks.
A	10·32	1·73	......	115·33	8·59	......	......	......	......	100·00	....
B	...8·01	·74	......	49·33	......	7·27	......	1·32	......	101·32	....
C	10·34	·19	......	12·66	......	10·15	......	......	2·88	98·44	....
	....	......	......		......	......	......	......	......	....	....
D	...8·93	......	·17	11·33	......	8·76	......	1·39	......	99·83	....
E	10·17	......	·59	39·33	......	9·58	......	......	·82	99·01	....
F	...7·08	......	1·04	69·33	......	......	6·04	3·54	......	102·55	....
	....	......	......		......	......		......	......	....	....
F	15·20	2·77	......	184·67	12·43	......	......	......	......	....	....
G	11·91	2·13	......	142·00	......	9·78	......	2·65	......	105·20	....
H	...8·55	·99	......	66·00	......	7·56	......	2·22	......	107·42	....
J	...7·18	·36	......	24·00	......	6·82	......	·74	......	108·16	....
	....	......	......		......	......	......	......	......	....	....
K	...8·80	......	·35	23·33	......	8·45	......	......	1·63	106·53	....
L	10·84	......	1·32	88·00	......	9·52	......	......	1·07	105·46	....
M	12·80	......	1·91	127·33	......	10·89	......	......	1·37	104·09	....
N	13·96	......	2·43	162·00	......	......	11·53	......	·64	103·45	....

In this form, a line is drawn between the distances at C and J, to show the position of the instrument, and these are ruled in the survey book, whilst the operations are going on; the distances may afterwards be filled in. A line is also drawn under the Level Readings Fore column at F, as well as in the Distance column, and this distinguishes between the position of the instrument and the last sight taken by it each time of fixing.

The example above shown is the first two "plants" of the level in the Levelling given previously in other forms of keeping the level book, and taking distances in the ordinary way.

It will be noticed that the reading of the second cross web at F, after moving the instrument, is 15·20, and with a 14-foot staff the instrument could not have

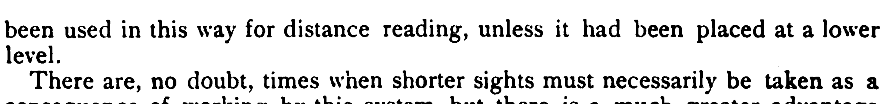


Fig. 451.—Staff Readings obtained from a Levelling.

been used in this way for distance reading, unless it had been placed at a lower level.

There are, no doubt, times when shorter sights must necessarily be taken as a consequence of working by this system, but there is a much greater advantage obtained from having no chainage to do, and all the distances read are horizontal ones. Where great accuracy is desired, the distances may be chained and checked by the staff distance readings.

Fig. 451 shows the position of the instrument when the preceding readings were taken.

It is plain that the distances, as worked out in the level book, will be those between the position of the instrument and the different stations on which the telescope. The cup is formed of a tube which fits the outer surface of the object-end of the telescope. This is prolonged sufficiently to lock the telescope against revolution by a dowel when the points that are used for index in the diaphragm of the telescope are vertical. The tube is cut in two and hinged to turn up, as shown

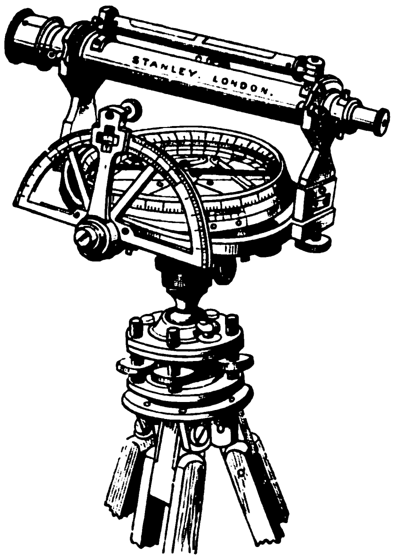


Fig. 468.—Improved Miner's Dial.

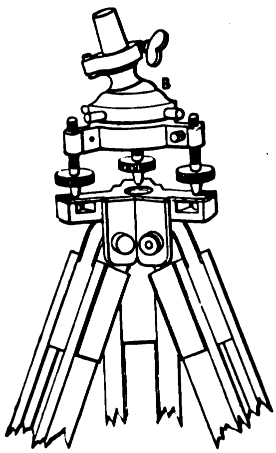


Fig. 469.

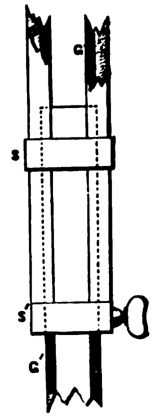


Fig. 470.

Fig. 469.—Stanley's Adjustable Ball Joint and Socket Tribrach Stand. Fig. 470.—Adjustment to Leg of Tripod.

in two positions, H and H′. When turned up it leaves the tube open for direct vision. A reflector, R, is placed in the cup, and there is an opening below it equal to the full aperture of the telescope. By this means a pair of lights or a line may be sighted up or down a shaft, and the azimuth of its direction be reflected to follow a line by slightly rocking the telescope upon its pivots. This may be done, however, with more refinement if there is a clamp and tangent motion to the vertical arc, which is placed only on first-class instruments.

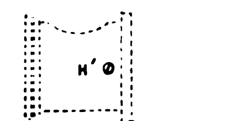


Fig. 471.—Reflecting Cup to Miner's Dial.

The method of using the needle for measuring angles underground consists in placing the instrument in the roadway leading from the shaft or other starting point, having first removed to a distance of 5 yards on either side all iron, which is known to deflect the needle, and having levelled the instrument, the sights of the compass are turned to a light at the starting point, the needle having first been liberated by a spring placed for the purpose. When settled, the needle will point to the magnetic north and the observer has to read and note the number of degrees the sights are lying from magnetic north, and in which quadrant those degrees are. The number of the sight beginning with 1 is entered in the survey book, the magnetic bearing and also the distance in links as measured from the shaft or other starting point to the dial, as well as the angle of inclination (if any) are also noted. The sights are then directed to a forward light, and the reading of horizontal and vertical angles together with the chain distance between the dial and the forward station booked. The dial is then lifted, carried forward, plumbed under the forward station, and a light being held at the station from which the instrument has just been removed, it is sighted through the dial, when the angle reading observed should correspond with the reading last booked; if not, some error has crept in which must be eliminated before proceeding further. If it correspond, the sights of the dial are directed to the forward light and the reading of the needle and any angle of inclination booked, the measurement from the dial to the last sighted station is then made, and noted, the instrument being again moved to the forward station. The process is here repeated and so throughout the length of road it is desired to survey. If a branch road leading from that which has now been surveyed, has also to be included in the survey, the surveyor, after completing the main road survey, must come back to where it branches off, and having made a note in the survey book at the time of advancing along the main road, such as "mark left opposite branch road," he now makes another note to identify the new starting point, thus say, "from 350 in 11," which means that the next sight following this remark was taken from 350 links in the eleventh set or sight going in. The survey may now be continued along this branch road, and afterwards in the same way along any number of other roads leading out of this, or other branches from the main road.

Now supposing it is required to survey from the shaft into the workings with a Hedley dial of ordinary construction, by angles not measured with the needle, but by the vernier, so as to avoid much labour in rail lifting.

First take out the screw which tightly holds the vernier plate to the bottom plate of the dial and prevents the possibility of the upper and lower plates having a separate motion during an ordinary needle survey. This screw will not be replaced till the completion of the survey. Set up the dial at a distance from the shaft, the farther from it the better, so long as a light at the pit bottom can be seen. Great care should be taken that at this first point of setting up the dial, which call A, the rails are taken up, and everything likely to attract the needle of the compass removed to a safe distance.

The instrument having been levelled, the needle is freed, and allowed to steady; after which the sights are turned so that the north end of the needle exactly coincides with the *zero* point of the lower plate of the dial, while at the same time the *zero* point of the vernier plate corresponds with that of the lower plate. The bottom plate is then clamped, by means of the collar tightening screw, the position of which is just above the ball-and-socket joint. The clamping screw of the vernier plate (the position of which is immediately underneath the vernier) is then slackened, and by means of the slow-motion screw (the position of which is underneath the dial plates, and opposite the clamping screw of the vernier plate) the sights and vernier plate are together moved until a light held by an assistant at the centre of the shaft is bisected *through the bottom slit* of the sights. The vernier plate clamping screw is then tightened, the angle read by means of the vernier to 3 minutes, or to 1 minute according to the accuracy of the instrument used, and the reading booked. The vernier plate clamping screw is then slackened, and the slow-motion screw used for bisecting *through the upper slit* of the sights a light which has been taken forward by an assistant along the road to the next station B. The vernier plate clamping screw having been tightened, the angle is read by means of the vernier and booked. Whilst these angles have been taken in the way indicated, the needle has remained on the zero point of the lower plate. The dial is now plumbed, a mark left to look back to, and the needle "locked," after which the instrument is removed to station B, plumbed underneath the mark previously sighted there and set up level. The lower clamping screw must now be slackened, but that of the vernier plate remains tightened, and a light held at A, is bisected *through the bottom slit* of the sights; the lower clamping screw is then tightened. Having proceeded thus far, as a matter of prudence, the angle indicated by the vernier should be again read, and if it does not agree of the survey are marked on the annulus as before. The traverser is then removed from the stand and fixed on the forward stand, and adjusted so as to sight back to the stand it formerly occupied, and clamped. The alidade is next unclamped, sighted to another forward stand, clamped again, and the direction and number or letter of the third line of the survey duly marked on the annulus, and so on for the remainder of the survey. In an enclosed traverse, if marks be left at the ends of the first line and in returning it be made the last survey line, then the first and last lines as marked on the annulus of the disc should exactly correspond, and a sure check as to the accuracy of the work is immediately afforded. The magnetic meridian is taken at any convenient spot in the survey by means of a trough compass, which is placed against the back edge of the alidade. The alidade and compass are then revolved together until the needle points to the north, when the line of the magnetic meridian is marked on the disc. The lengths of the several lines, offsets, &c., are taken and entered in the surveyor's book in the usual manner. The sights of the alidade are graduated to give angles of depression or elevation up to 25°, and thus the traverser with plain sights can be used for all ordinary surveys. For gradients of more than 25°, or for more accurate reading of dips, a quadrant with sights, or with reversible telescope, is supplied. These accessories are in addition to the graduated sights, and can be applied at pleasure.

In plotting, or laying down the work, the disc of celluloid is removed from its circular table, weighted down on the drawing-paper, and the direction of the survey-lines transferred in due order by means of the ordinary rolling parallel ruler. For future reference the disc itself may be kept, the name and date of the survey being recorded thereon, or the magnetic bearings of the lines may be readily read off, and the same duly entered in the survey book. A convenient method of reading the bearings is to place the disc in the centre of an ordinary cardboard protractor (which has been cut out for the purpose); the protractor is fastened down on a board, and the disc pinned down in its proper position. A central metal spill projects from the board, and by means of an alidade, constructed so as to work on this spill, the bearings are read off with ease and rapidity.

The instrument, which is supplied by Messrs. John Davis & Son, Derby, is not suitable for use in wet mines where it is impossible to fix it clear of water-droppings, which obliterate or render indistinct all the pencil-markings and numbers on the disc.

It is well-known that the magnetic north differs from true astronomical or geographical north. The angle formed between them is called the declination of the needle, and is continually changing. The declination may form an angle either east or west of true north; at the beginning of 1893 the magnetic needle had a mean westerly declination of 17°12′ at Greenwich, but is diminishing about 8′ annually. The rate of diminution varies locally and from year to year. In 1663 there was no declination, and the time will probably come again when there will be none. The declination is not the same in all parts of the world. In travelling west of Greenwich it increases. At present the declination is west in Europe and Africa; east in Asia and the greater part of North and South America. The following table gives an idea of the variation in declination for London approximately for a few dates:—

Year 1580	Dec.,	11° 36′ E.	Year 1860	Dec.,	20° 40′ W.
,, 1663	,,	0°	,, 1870	,,	20° 19′ W.
,, 1700	,,	8° 20′ W.	,, 1880	,,	18° 58′ W.
,, 1818	,,	25° 41′ W.	,, 1888	,,	17° 33′ W.
,, 1850	,,	19° 31′ W.	,, 1890	,,	17° 9′ W.

It will be seen by the above table that the needle pointed due north in 1663, and that it attained its greatest western declination in 1818, and that it is now losing westerly declination.

Two magnetic needles have seldom exactly the same variation, so that if a needle survey were made of the underground workings, and the surface were surveyed with another needle, an error would arise in connecting the two if proper allowance were not made for the difference of variation in the needles. The needle is subject to periodical variations, the most important being a diurnal fluctuation from east to west. The north end of the needle reaches its most westerly position at about 2 p.m., and its most easterly position at night or early morning, the difference between its extreme positions amounting to 10 or 12 minutes or more in summer, but to much less in winter, and is irregular at times between the extreme seasons. The average position of the needle in making its diurnal movement is attained about 10 a.m., and again about 6 p.m. The fluctuation is at its minimum near the equator and increases in advancing northwards from it. The needle is acted on by magnetic storms, which may seriously deflect it from its ordinary position.

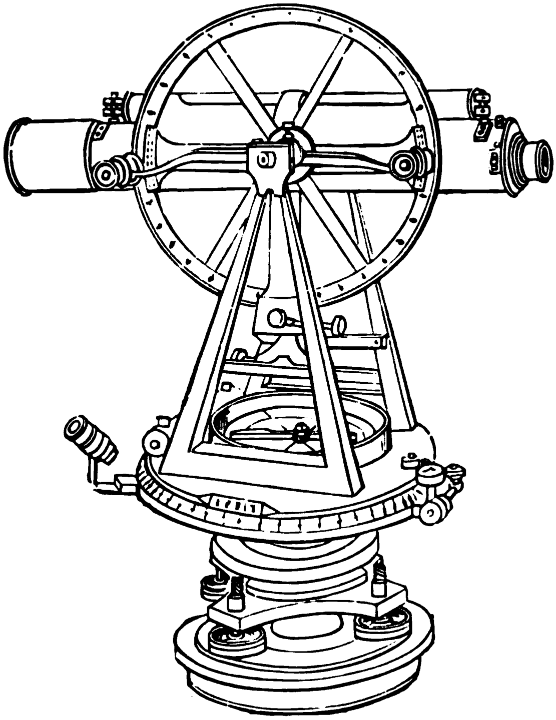


Fig. 476.—Davis's Transit Theodolite.

What is called the dip of the needle is its direction as compared with a vertical line. In order that a needle may assume a horizontal position after being magnetised, it is first carefully balanced before magnetization and is then slightly weighted at its southern extremity. The mean magnetic dip at Greenwich in the year 1886 was $67^{\circ} 27'$ and was diminishing annually at the rate of $1' 24''$. It is uniformly nearly *nil* at the equator, and increases until over one of the magnetic poles, where it becomes vertical. There are two magnetic poles in the northern hemisphere active in directing the needle, one in Siberia, but the most active is about Melville Island; also two in the southern hemisphere, which are supposed to be nearly together, of which the exact position is not ascertained. The plan adopted to balance the needle in opposition to the direction of the dip is to place a rider over it. This clips the needle sufficiently to hold firmly to its place, and yet is sufficiently loose to be moved by the fingers to balance. The rider has to be shifted when the instrument is taken into a country where the dip is different. Where a needle is taken abroad without any rider, it may be balanced by means of a little sealing-wax placed upon its uptending end.

The modern form of theodolite has a telescope which can be moved round the entire circle in the vertical plane, and is called a transit theodolite.

Fig. 476 shows Messrs. Davis & Son's Transit Theodolite, and Fig. 477 that of Hoskold's Miner's Transit Theodolite, as supplied by Messrs. John Davis & Son, All Saints Works, Derby.

In Fig. 476, the telescope, with a spirit level fixed on it, rests on upright supports, which are of such a height above the horizontal circles as to admit of the telescope turning right round on its axis. It is provided with a vertical circle, and by means of verniers and microscopes the angles of elevation or depression are read. The horizontal limb is composed of two circular plates, which fit accurately one upon the other. The lower one is chamfered and graduated at every half or every third of a degree. The upper is called the vernier plate, and has portions of its edge chamfered off, so as to form, with the chamfered edge of the lower plate, continued portions of the same surface, and these chamfered portions of the upper plate are graduated to form the verniers by which the limb is subdivided to single minutes. Usually there are two such verniers placed 180° apart. By means of clamping screws the upper plate may have a motion independent of the lower plate or it may move with it.

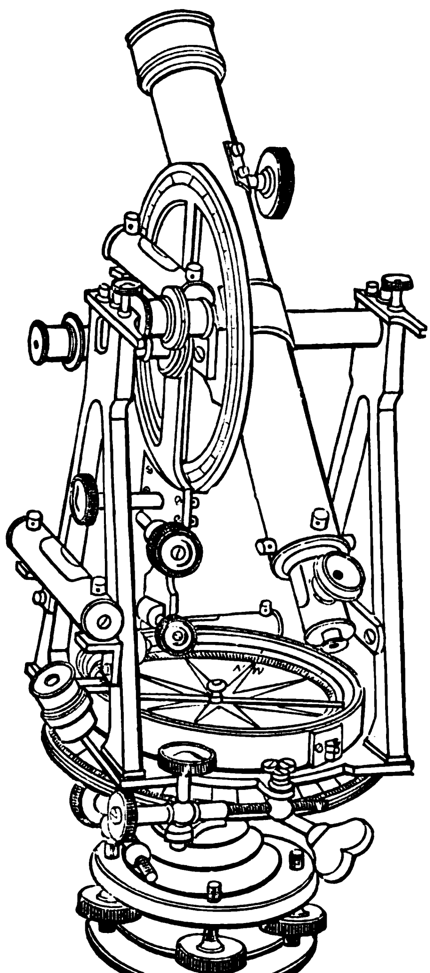


Fig. 477.—Hoskold's Miner's Transit Theodolite.

There is a clamping screw fixed to the vernier plate for the purpose of keeping the two plates together when tightened, or of allowing the upper plate to move whilst the bottom one is fixed. A tangent or slow-motion screw gives the upper plate a slow motion after the clamping screw is fastened. Similarly, a clamping screw tightens the collar below the bottom plate, and a slow-motion screw is placed for moving the whole limb through a small space so as to adjust it more perfectly after tightening the collar. Two spirit levels are placed upon the horizontal limb at right angles to each other and a compass is sometimes placed upon it in the centre and between the supports for the vertical limb. As, however, this only allows of a small compass, the more modern plan is to fix a magnetic needle in an oblong shaped box, and this trough compass is sometimes fixed on the telescope on the side opposite the main spirit level and sometimes it is made to slide under the lower plate. This arrangement allows of the use of a larger and more reliable needle. Two parallel plates with four milled-headed screws similar to those on the spirit level are placed below the lower plate as a means of adjusting the levels accurately before making an observation with the instrument. The vertical limb is divided upon one side every 30 or 20 minutes and two verniers are placed so as to read the vertical angles to single minutes. The vertical limb has a clamping screw and a slow-motion screw; the former on being tightened holds the

The method of traversing on the surface is similar to that underground, but for extensive surveys, the principal points should be determined by a system of triangles proceeding from an accurately measured base of considerable length. The details are afterwards filled in, but it would occupy too much space to fully describe surface surveying here. It may be said in passing that in filling in the details, offsets are taken with a rod or tape from the traverse lines, usually at right angles to them, to all the points required to be shown in the survey, such as houses, hedges, fences, the edges of streams, &c.

Fig. 478 shows a plain theodolite, which, however, is not much used now, it having been superseded by the transit. It differs in construction from the transit by having only a half-vertical circle, with a single vernier, and in the arrangement of the fittings connected with the telescope. There are two verniers on the horizontal circle, but a single microscope is used to read them by passing it in a groove from one vernier to another.

Fig. 478.—5-INCH PLAIN THEODOLITE.

The standards or A-frames in the plain theodolite are firmly attached to the vernier-plate clear of the compass-box. The pivots of the transverse axis rest on coupled bearings on the tops of the standards. The vertical arc is fitted over the transverse axis, which has a turned flange, to which the arc is firmly screwed. The arc is divided to 30′, and reads with a vernier to minutes. The vernier is sometimes read with a microscope jointed on the transverse axis, and sometimes with a loose magnifier, for economy. Difference of hypothenuse and base is generally divided on the back of the arc. The vertical arc has a clamp and tangent placed at the back, not seen in the engraving. Along the bar of the vertical arc a stout plate is attached by screws. From this a pair of Y's with clips and eye-pins support the telescope, as in the Y-level. The diaphragm of the telescope is cross-webbed, and a simple cap is used to cover the object-glass end instead of a ray-shade. The principal level is fixed to collars round the telescope, and rests under it for compactness: it can be adjusted by capstan-headed nuts.

The vernier-plate carries only one level at right-angles to that of the telescope. The telescope is therefore set to zero by the vertical arc, and the two levels are then used as the pair upon the vernier-plate of the transit.

The telescope of the plain theodolite cannot be moved round an entire circle in a vertical plane, but it may be reversed in the clips for ranging lines.

In Stanley's new model transit theodolite, Fig. 479, the principles of construction are the same as in the ordinary transit theodolite, but the distribution of materials and details is very different. The illustration is of a 5-inch instrument.

E to exclude dust. The foot of the screw A has a ball joint which rests on a flat surface, upon which it is sprung tightly by a spring-plate above the ball.

The upper plate of the tribrach is fitted with a stage, Fig. 481. A dovetail-slide B B is fitted upon the base of the stage, adjustable for wear by a slip-piece with two screws at the narrow part of it. The slide is adjusted to position in the direction of its dovetail fitting by the screw B′, so as to move the whole instrument above it for centring in this direction. A slide acting in the same manner, with dovetail fitting-pieces at each end only, A A, moves the slide for an equal distance for centring transverse to the slide B by the milled head A′. The motion given the screws A′ and B′ permits the perfect adjustment of the theodolite over a point on the ground by means of a suspended plummet, after the instrument is set up to nearly its true position by movement of the tripod legs. The range of motion is about $\frac{3}{4}$ inch—an amount quite sufficient for final adjustment, but which does not materially affect the equilibrium of the instrument upon

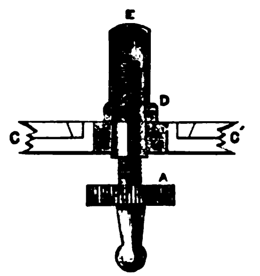


Fig. 480.—Foot-screw Stanley Transit.

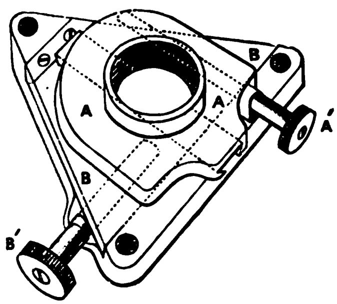


Fig. 481.—Stanley's Tribrach Mechanical Stage.

its rigid tripod, as it has in this case a broad, solid base even in the extreme positions of the slide.

For examination or adjustment of the theodolite the tripod stand should be at first firmly fixed with its legs extended to an angle of about 70° to the ground, which should be solid and hard. As the telescope has to be brought to the height of the observer's eye, it is well to mention his stature in ordering an instrument. The tripods are made for tall men, and are often awkward and unsteady if the legs are extended to bring the telescope down to the height of a person of short stature. They may always be cut down and re-made. When the tripod is set up, the toes should be each separately pressed down, so that future slips are impossible. This being done, the instrument is taken from its case and grasped firmly by the body part under the horizontal circle, and is placed on the tripod at once, and screwed firmly but not too tightly down upon its bearing-surface. With a transit theodolite the upper part is generally detached and packed separately in its case. Where this is so, after the body part is fixed on the tripod, the cleats on the top of the standards must be opened out, and the upper part of the instrument, lifted by its telescope, be slowly lowered into its bearings, being particular at the same time that the clips under the telescope embrace their stay-piece on the standard. The cleats must then be closed over the pivots. The instrument being set up to position, all levels may be adjusted to the centres of their runs, and every part clamped sufficiently to make the instrument firm, but in no case using violence to produce a strain in any part. The clamps or other fittings are afterwards separately released as they are required for examination or for the adjustment of the separate parts to which they relate.

For the correction of faults that may be included in the above operations, the parts of the instrument must be separately examined.

Examination of the Transit Axis.—The best means of adjusting this axis in a theodolite is by a *striding-level.* Where this is not provided with the instrument—and it is often left out for economy—the axis is generally better left as it is adjusted, in this particular, by the maker. To adjust the transit axis the vernier-plate bubbles are set exactly true by reversing angles of observation. The cleats are opened and the striding-level is mounted above the instrument resting upon the pivots. The telescope is placed exactly over an opposite pair of parallel plate-screws, or parallel with two screws if the base adjustment is on the tribrach principle. The striding-level is then carefully observed and reversed on the pivots. If there is any difference in the run of the level-bubble the transit axis is adjusted by raising or lowering the movable V on which one pivot rests by turning the capstan-nuts until it is quite correct. This adjustment is almost superfluous, as the axis is generally set right at first and is not subject to change.

For larger theodolites of 12 inches and over, the transit axis is much better adjusted by means of an artificial horizon. By the use of this instrument in the northern hemisphere the pole star is first observed directly by the telescope, and then by its reflection from the horizontal surface of clean mercury placed on the ground at 12 feet or so from the instrument. If the star and its reflection cut the webs equally in directing the telescope by movement of its transit axis only from the one to the other, this axis must be truly horizontal. If the vernier-plate be then turned a quarter of a revolution, and the exterior axis a quarter of a revolution, the telescope transited and observation be repeated, the verticality of the principal axis may be adjusted with perfect certainty. The principle axis should be moved one-eighth revolution all round, and the bubble examined at every position to assure perfect adjustment. With the plain theodolite, Everest's, and some others, the transverse axis is fixed to position by the maker.

Examination and Adjustment of Webs.—The ordinary manner of webbing the diaphragm of a theodolite is shown in Fig. 428. Horizontal angles are taken by the upper intersection of the nearest to vertical webs. A single web is placed horizontally for taking vertical angles: it is necessary that this should be nearly true. When the theodolite has its axis vertical, as shown by the vernier-plate bubbles being in the centre of their runs, if one end of the horizontal web be set to cut a small distant object by sight in the telescope, the same object should keep on the web while the tangent-screw of horizontal circle is moved a distance sufficient to traverse it, the hand being always taken from the screw while the observation is made. If it does not, the collimating-screws should be lightly tapped with the back of a penknife in the direction to set it right. These screws have a slot in the body of the telescope, under the loose covering plate, sufficient to permit of this small adjustment.

Adjustment of the Telescope to Vertical Collimation.—The eye-piece is first focussed as before against a piece of white paper held obliquely in front of the object-glass until the webs are sharply seen. The axis of the telescope is then examined for vertical collimation error. The method of doing this has been already described for a telescope placed in Y's, as it is in the Y-level, and the plain theodolite. The only difference with the transit theodolite is, that instead of turning the instrument in its Y's, the telescope is *transited,* as it is termed, over on the transverse axis exactly half a revolution, or 180°, as seen by the vernier reading; and the horizontal circle is moved also half a revolution, so that the telescope points again on the same distant point which is used for an object. If the webs still cut the same point or small object, the webs are in vertical collimation, feet, another at 150 feet, and every other 50 feet, or contour lines may be marked at every change of 10 feet or other number according to circumstances. By levelling underground, these points of equal altitude in the different roads are ascertained, and afterwards connected on the plan by faint dotted lines. Or, if contour lines are not marked on the plan, the levels may be written neatly and clearly about the plan on all main roads. They show at a glance the probabilities of water taking certain directions; whether the coal has to be hauled up or downhill to the shaft, and direct the manager's attention to parts of the road which probably may be improved. A record of this sort on one seam becomes a very valuable guide in directing the operations on another, the workings of which may follow over the same ground, and in the case of abandoned collieries, plans containing such information are of the utmost assistance to a neighbouring venture.

The Mines Act, 1887, renders it not only necessary to survey the workings and extend the plans in accordance therewith at least every 3 months, but also to show the general direction and rate of dip of the strata, together with a section of the strata sunk through, if the last be reasonably practicable.

To plot the survey by the protractor and parallel rule, place the protractor as stated on one of the lines representing the true north and south meridian of the plan; if the angles of the survey were taken from the true north make use of the parallel meridian line which is most convenient. The protractor would be placed with its zero point on the north end of the line and with its centre on a continuation of the same line where it crosses one of the parallel lines at right angles to the meridian. A continuation of the same meridian line southwards will coincide with 180° on the protractor. The numbers of the sets should be then pricked off, each being twice marked, supposing a good protractor with folding arms to be used. A magnifying glass is held in one hand (the survey-book lying open on the plan), whilst with the other the vernier of the protractor is set at the angles, corresponding with those of the survey-book. The best plan is to mark all these off with only one of the prickers the first time, and then to go through them either backwards or forwards a second time and use the pricker on the opposite folding arm as before remarked. Should any error arise it will be discovered from the fact that a straight line between the two prick marks of the same set ought to cross through the centre point of the protractor, which, in laying on the plan as stated, should be at the intersection of two of the lines referred to. The parallel rule is then used and placed in a line with the two punctures representing the first set, and taken in a truly parallel course by means of its rollers across the paper to the point from which on the plan the first set was taken. A pencil line is carefully ruled of a reasonable length; by looking at the survey-book some judgment of this may be formed, but it is better to draw it long enough as the extra length not required may be rubbed out afterwards. The parallel rule is now carefully moved back to the punctures at which it was first set, and if it corresponds with these the line will be truly drawn. If not, take the line out with india-rubber and repeat the operation. The operator must be careful not only to place the lines truly parallel with the set-marks but also not to reverse any of them and rule them on the plan in the opposite direction to which they should be. The scale may be now laid down to the line drawn and the length taken from the survey-book having first been subjected to the proper deduction for the vertical angle (if any) the horizontal length is marked on the line by a puncture, a pencil ring made round it, the number of the set written in pencil near the ring, and the excess of the line beyond the ring is taken out with india-rubber. The parallel rule is then placed truly by the punctures representing the next set and brought in a carefully parallel line to the ring at the end of the line forming the first set. A pencil line is again ruled from this ring in the proper direction and the method of procedure corresponds with that described for the first set, and so on throughout the survey, taking care that if a remark "from —— in —— set" appears in the survey-book that the plotting proceed from the point indicated. The pencil used should be finely pointed and fairly hard—an H.B. is very suitable for the purpose. As these pencil lines represent the centre of the road (it being impossible to fix a surveying instrument right in the side of the road) it is customary, instead of inking in a single line on the plan representing the centre line, to rule two lines representing the sides of road, and these should be inked in at equal distances from the centre line which is shown in the plotting. Indian ink and a proper drawing pen are used for the purpose of inking in. Fig. 488 shows the plan of the survey before alluded to.

It is highly desirable that, wherever practicable, surveys should have proof of their accuracy by what is called "ties." A traverse survey commencing at some well-

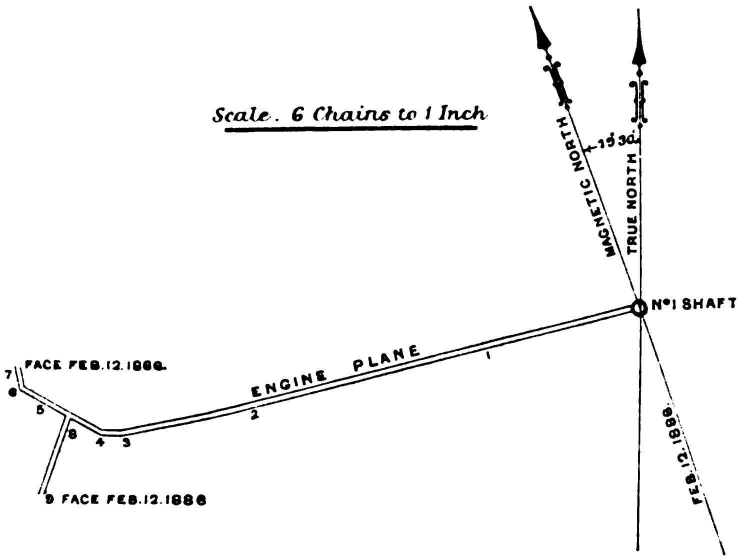


Fig. 488.—The Plotting of a Survey.

defined point on the surface may always have a "tie," that is, it may be closed at the starting point, and although it entails a little more labour where a doubling back and returning along a different route is resorted to, it may be well worth the labour. If such a survey be accurately made, on plotting it, the end of the last "set" line will exactly reach to, and coincide with, the beginning of number one "set" line. Similarly there may, where convenient, be intermediate checks or ties, taken from any one point of the main lines to others, and these checks on plotting an accurately made survey strengthen the proof of accuracy in the whole. If a slip has occurred in conducting the survey, on plotting, it will be discovered by the "set" lines not truly fitting where they should, and the error now known to exist must be discovered or a fresh survey made.

Again, a "tied in" survey may be tested by Euclid, for, "The sum of all the interior angles of any rectilinear figure, together with 4 right angles, are equal to twice as many right angles as the figure has sides." This is not so thorough a test as the plotting, because it checks only the angles taken and not the chainage, still it may sometimes be useful. For instance, it may be applied where the plotting proved an inaccurate survey, and if the angles are found to be correct as surveyed, the error or errors must be in the chainage.

Underground workings do not afford the same facilities to "tie" surveys as are obtained on the surface, for however tortuous their course, the roadways must be traversed. Frequently, however, a "tie" can be obtained by following a roadway from its junction with another road until by means of branches the first-mentioned junction is returned to.

Advantage should be taken of these checks wherever they can be obtained.

Fig. 489 shows an underground survey "tied in" in the manner described.

	166 to 0 in 1	
22	19° 5′	(166)
21	21°	(96)
20	14° 45′	(60)
19	5° 40′	(105½)
18	342°	(61)
17	332° 55′	(217)
16	350° 55′	(56½)
15	359° 15′	(239½)
14	20° 25′	(41)
13	31° 35′	(78)
12	249°	(221)
11	247° 47′	(157½)
10	240° 20′	(103)
9	214° 33′	(29½)
8	176° 55′	(243)
7	163° 12′	(102½)
6	178° 58′	(127)
5	170° 11′	(261)
4	134° 27′	(35)
3	113° 26′	(67)
2	99° 51′	(84½)
1	109° 25′	(169)

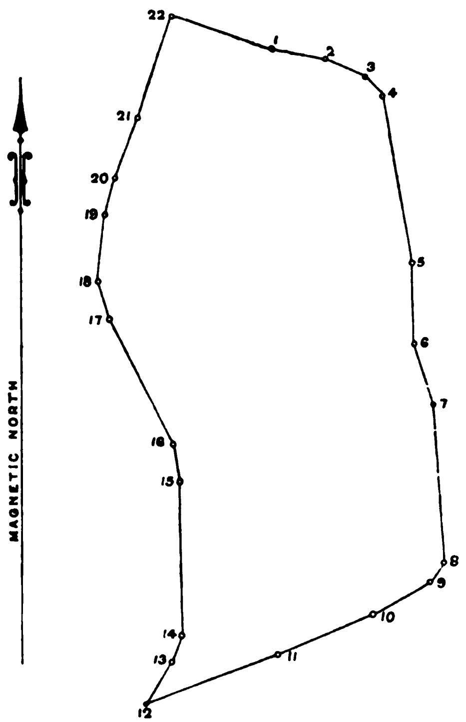


Fig. 489.—"Tied in" Survey.

The plotting of this survey proves it correct, but as an example for testing other surveys, the accuracy of the angles taken is proved below by calculating all the interior angles at the numbers of the "set" lines.

180° − (109° 25′ − 19° 5′) = 89° 40′	Interior angle at	22
180° + (21° − 19° 5′) = 181° 55′	,,	21
180° − (21° − 14° 45′) = 173° 45′	,,	20
180° − (14° 45′ − 5° 40′) = 170° 55′	,,	19
180° − (365° 40′ − 342°) = 156° 20′	,,	18
180° − (342° − 332° 55′) = 170° 55′	,,	17
180° + (350° 55′ − 332° 55′) = 198°	,,	16
180° + (359° 15′ − 350° 55′) = 188° 20′	,,	15
180° + (380° 25′ − 359° 15′) = 201° 10′	,,	14
180° + (31° 35′ − 20° 25′) = 191° 10′	,,	13
249° − (180° + 31° 35′) = 37° 25′	,,	12
180° − (249° − 247° 47′) = 178° 47′	,,	11
180° − (247° 47′ − 240° 20′) = 172° 33′	,,	10
180° − (240° 20′ − 214° 33′) = 154° 13′	,,	9

shown the surface levels of the different pits crossed, and all other prominent objects. A careful scrutiny of the ground, quarries, cliffs, railway cuttings, and an examination of the fossils obtained from the rocks, &c., may then follow; the classification of the strata noted, with the amount of dip or rise in the direction of the line of section, and the direction of dip. By this means it is possible to calculate the thicknesses of the different rocks, and to plot them below the surface line previously shown on the drawing paper. In colliery districts, the pit sinkings and workings enable the surveyor to show the position of the seams of coal, the different strata, and also the faults met with, all of which require to be drawn.

The section is outlined in pencil, and afterwards inked in with Indian ink, the main lines being shown in black, the faults with some other striking colour. Coal-seams are shown with bold, thick black lines, and the different strata are distinctly coloured. When the colours are all dry, the names of the coal-seams may be neatly printed over them, and either a reference printed in a corner of the paper identifying the strata represented by the different colours, or the names of the strata may be printed on them without using a reference. A title or heading is then printed along the top of the paper explaining the nature of the section and its scale or scales drawn or given.

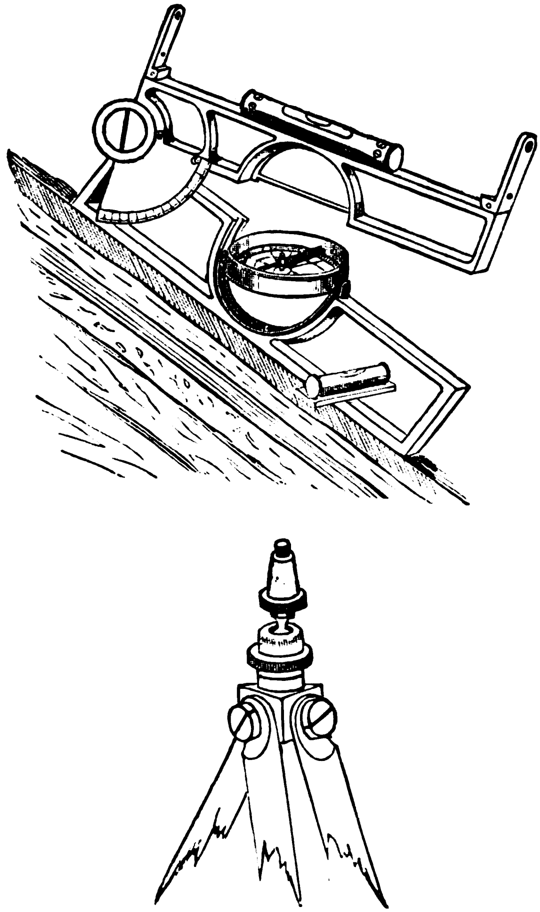


Fig. 495.—Louis's Improved Davis's Clinometer.

A very convenient instrument for ascertaining the necessary particulars is Louis's improved Davis's clinometer, made by Messrs. John Davis & Son, of Derby, and shown in Fig. 495. In it a compass is placed in the lower portion of the clinometer frame, where it is carried by means of pivots on a brass arc, which may be revolved in the frame. This allows of the instrument being laid on strata of any dip, the rate of which will be ascertained by raising the upper portion of the clinometer frame until the air-bubble of the spirit-level placed on top is in the centre of its run, as shown in the drawing. The angle of dip may now be read by means of the graduated arc, whilst its direction is obtained by giving motion to the compass-box till it assumes a horizontal position in the under portion of the frame, and then reading the needle-bearing. The strike of strata is usually determined by first ascertaining their maximum dip, and the compactness of this instrument allows it to be readily turned whilst readings are taken to fix the exact direction of dip in any one place. The bubble of the lower limb is mounted on a swivel, and this enables the instrument to be levelled both ways without being reversed. The compass-box is reversible in the under portion of the frame. The size of the clinometer is $6\frac{3}{4}$ inches long $\times$ $\frac{1}{2}$ inch wide $\times$ 3 inches deep, and weighs 1 lb. 2 ozs. It is provided with a tripod with ball-and-socket joint, 3 feet 10 inches long, and weighing 1 lb. 8 ozs. When

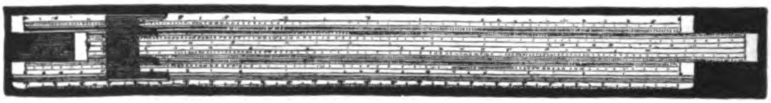


Fig. 496.—Celluloid Slide-Rule with Ordinary Cursor.

attached to the tripod it may be used as a Dumpy level where great accuracy is not required, or for ascertaining the inclination of a sloping adit, or undulations of the ground.

The number of calculations required in large colliery offices makes it highly desirable to use a calculating slide-rule in order to save labour. Although slide-rules are somewhat complex, there is no difficulty in using them if the instructions

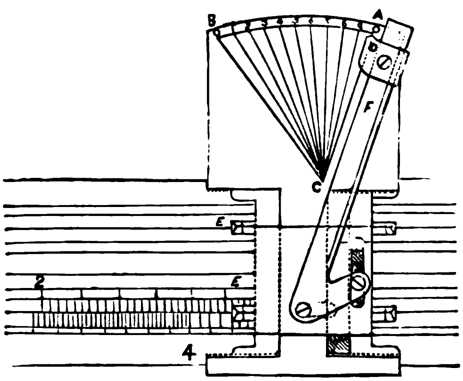


Fig. 497.—Goulding's Improved Cursor.

supplied by the inventors are carefully followed. A great number are made for special purposes only, some being more useful in one branch of engineering than others.

Fig. 496 shows a celluloid slide-rule with ordinary cursor, introduced by Messrs. J. Davis & Son, Derby, which enables a variety of technical calculations to be expeditiously made in addition to ordinary multiplication, division, evolution, and involution. The body of the rule is of wood, in which strips of hard white celluloid are inlaid, and on these celluloid strips the scales are engraved, being thus in black lines on a white ground. This rule is of the Gravet type, 10 inches in length, and the markings remain without relative change from variation of temperature. The upper part of the rule is divided into two parts, containing scales perfectly alike. The lower scale is exactly like the upper ones, but twice the length. The slide is divided the same as the rule, so that when the 1 of the slide is at 1 of the rule all other divisions will coincide. The cursor is the small metal frame fitting in two small grooves in the edges of the rule, and is used for pointing and subdividing the spaces on the rule.

being securely clamped to the lower, with the vernier at zero, the instrument was directed towards the church, and the lower clamping-screw tightened with the webs of the telescope correctly bisecting the top of the church spire. This being done, the upper plate was loosened, and an angle from the church to the sun obtained at that time, 9.15 A.M. Reading, 269° 52′. At 3.15 the upper clamping-screw was slackened, and the telescope, with the vertical limb undisturbed, directed to the sun; on its edge appearing in contact with the webs of the instrument, the reading of the angle was recorded as 41° 45′. The angle contained between the two readings therefore was (360 − 269° 52′) + 41° 45′ = 131° 53′.

For the correction we have—

Log. of $\frac{3}{2}' = 90''$	=	1·954243
„ sec. 51° 36′ 15″ (N. latitude of Pontypridd)	=	10·206845
„ co-sec. $\frac{1}{2}$ of 131° 53′	=	10·039467
		2·200555 = Log. of

158·7, or say 2′ 30″. Therefore the approximate direction as found by the observation is too far to the *left* by 2′ 30″.

∠ from church to 1st observation 360 − 269° 52′	=	90° 8′
„ „ 2nd „	=	41° 45′
Difference		2) 48° 23′
Difference halved		24° 11′ 30″

but being 2½ minutes too much to the left, the correct reading should be 24° 9′. The magnetic bearing from the same point of observation was taken to Pontypridd church by the same instrument at the same date as S. 43° 12′ W. from which deduct 24° 9′

Making the magnetic declination 19° 3′

Question 132.—Describe the underground and surface levelling staves, and state how you would proceed to level underground with the level and staff.

The staff used in underground operations is usually made about 9 feet long, and is in three pieces, which are connected together like the joints of a telescope, and these close down to 3 feet 6 inches. It is graduated into feet, tenths, and hundredths. The figures appear in an inverted position as seen through the telescope, but the surveyor soon gets accustomed to this. To avoid this some surveyors prefer having an additional lens placed in the telescope, others have the staff figures arranged so as to appear the right way up when seen through the telescope. The staff used on the surface is of the same description as that ordinarily used underground, being made in three pieces, the two upper sliding into the lower like the joints of a telescope, and being graduated into feet, tenths, and hundredths. The total length when out is 14 or 16 feet, and it closes to 5 or 6 feet. Staves of 18 and 20 feet are occasionally used. The method of proceeding with an underground levelling is as follows:—

An assistant holds the staff in an upright position at the shaft, and the levelling instrument having been set up truly level in the road which it is desired to level, a sight of the levelling staff is taken and the result entered in the book. The levelling staff is then moved along the road to where its inclination changes, or to where, owing to a change in the direction of the road, it is impossible to see farther, and a sight from the instrument gives the reading there. The fore-sight reflected rays could not have penetrated far from the shaft owing to obstructions in the galleries, while on sunless days not the faintest glimmer of light could have relieved the absolute darkness of the mine.

SAFETY LAMPS.

Humboldt's Lamp.—An ingenious lamp was designed by Humboldt in 1796, which burned without drawing the feed air from a surrounding atmosphere. It was only used for experimental purposes, as it very soon went out.

Clanny's First Lamp.—In 1813 a society for the prevention of accidents in coal mines was formed in Sunderland, and in the same year Dr. William Reid Clanny, of that town, devised and exhibited at the meetings of the society a lamp which was afterwards tried in an inflammable mixture of air and gas at the Herrington Mill pit, in the county of Durham, on October 16, and again on November 20, 1815. The flame of this lamp was insulated, and the air supply necessary for its combustion was blown through a stratum of water placed below by means of a pair of bellows. A second layer of water was arranged above the flame through which the products of combustion escaped. The water thus separated the air around the flame from that outside the lamp. In an explosive atmosphere it went out. This lamp was too unwieldy to be useful, and after being used experimentally was quickly superseded by later inventions.

The Davy Lamp.—In August, 1815, Sir Humphry Davy visited the collieries near Newcastle-upon-Tyne to investigate the nature of mine gases, and on November 9 of the same year read his celebrated paper on fire-damp before the Royal Society of London. This paper was published in the Philosophical Transactions of that Society, and contains a fascinating account of the experiments undertaken by Davy, which led to his safety-lamp, especially interesting in view of later inventions in the same line.

A flame is always a burning gas whether supplied directly as in the case of ordinary coal gas, or made from a liquid as in the case of oil lamps, or from solids as in the case of candles, or produced from the burning of wood or coal. In all cases the material, whether yielding one gas or more, takes a gaseous form before the appearance of flame. Before a gas can burn it must be first heated to what is called its temperature of ignition.

Davy found that fire-damp requires an admixture of a large quantity of atmospheric air to render it explosive. He experimented with small metallic tubes and sieves of brass and iron wire gauze, which he found, under certain conditions, prevented the passage of flame. Small metal rings have a remarkable power of reducing the size and illuminating power of the flame. Smaller rings were tried and altogether prevented the passage of flame. Metallic tubes one-fifth of an inch in diameter and 1½ inches long were found to be efficient. A spiral of wire at ordinary temperature, if quickly dropped into the flame of an alcohol lamp, will extinguish it, see Fig. 509. A spiral of thick copper wire is suitable for the experiment. The alcohol flame is not extinguished by the exclusion of air as in an ordinary extinguisher, but because it is cooled down by the wire spiral. If the latter be heated before being introduced into the flame, it loses its extinguishing power.

If gas be lighted at a Bunsen burner or at an alcohol lamp, the flame from either of which will not deposit soot on objects in contact with them, and if a piece of fine iron wire-gauze about 6 inches square be held at one corner in the hand, first above and then lowered on to the flame, there is no appearance of flame above the gauze any more than there would be above a solid iron plate.

$\frac{2}{3}$rds of the Davy lamp, it being fastened to two of the upright bars, on which it may be made to slide up and down. Besides its defectiveness in respect to passing the flame at a rather low velocity, another great objection to the Davy lamp is its insufficient light, and most of the more modern lamps give a better light, glass being introduced at the bottom instead of gauze for this purpose.

An examination for fire-damp with the Davy lamp should be made at first without reducing the wick-flame. With the flame at its full size, the upper portion of the lamp is filled with the products of combustion, to the exclusion of a large volume of ignited gas, and a small volume burning within does not so readily pass to the outside, whereas a reduced wick-flame has a tendency to facilitate passage of the flame to the surrounding atmosphere. The reduced wick-flame is only necessary for low percentage of fire-damp. The lamp should, of course, be kept in an upright position, so that the flame and the smoke do not impinge against the sides of the lamp. The gauze cylinder should not exceed $1\frac{1}{2}$ inches in diameter, for the larger the lamp the greater will be the force of an internal explosion, and consequently the greater the liability to drive the flame through the gauze from within. Thus, what are called Scotch lamps, which are really large Davy lamps with gauze cylinders about 3 inches in diameter, can hardly be classed as safety-lamps at all.

The weight of an ordinary unshielded Davy lamp is about $1\frac{1}{2}$ lbs., while that of a "tin can" Davy (see Fig. 515) is about $2\frac{1}{4}$ lbs., without wick or oil in either.

Owing to the greater velocities of air currents in mines now than formerly, and to the discovery that an unprotected Davy lamp is not safe in an explosive mixture having a velocity of 6 feet per second, the use of the lamp under such conditions is prohibited by law unless it is specially protected.

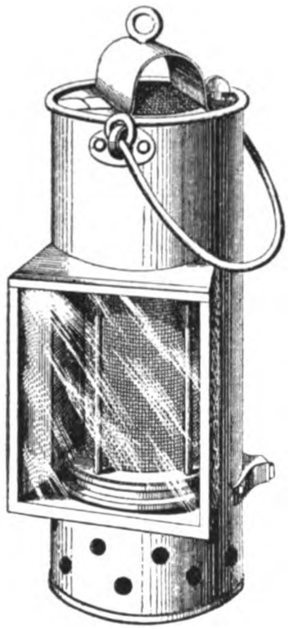


Fig. 515.—Davy Lamp in Case.

In the North of England this has led to the use of the Tin-can Davy shown in Fig. 515. This consists of an ordinary Davy lamp enclosed in a tin case provided in front with a flat pane of glass. The tin is perforated at the bottom for the admission of air. The case should extend from the bottom to the top of the lamp. Near the top, which is open, the handle is fitted. The lamp is locked to the case, and thus made it was one of the safest tested by the Royal Commission on Accidents in Mines, 1886. In their Report, the Commissioners say that "The Davy lamp enclosed in a case of any form which completely shields the gauze from the direct action of the current, may apparently be trusted in currents not exceeding in velocity 2,000 feet per minute (about 33 feet per second), but for higher velocities the greatest care is necessary in designing the case and its relations to the enclosed lamp."

Cases with inlet and outlet shut-offs may be arranged which would form additional safeguards. Cases without such appliances render the light steadier and somewhat improved. At best, however, the Davy gives a weak light, owing to the opaque wire surrounding the flame.

In different districts Davy lamps are made with movable metallic shields of different heights, which partly surround the gauze, and the dimensions of makers' lamps differ slightly.

External glass cylinders reaching to different heights up the gauze are also used, and lamps so shielded are usually known as "Jack lamps." With short glasses there is sometimes no provision for the air to pass underneath, but the long glasses usually rest on an india-rubber ring with projections, or on four metal pegs, so as to allow of a passage for the air to feed the flame under the glass. The glasses may be moved up and down outside the gauze, unless they extend throughout its whole length.

To be efficient in a current of 10 feet per second the glass cylinder surrounding the main gauze must reach above the double gauze formed by the smoke gauze cap.

Fig. 516 shows an arrangement of Davy as a combined examiner's and shot-firing lamp, sometimes used where there is not a specially devised shot-firing lamp. Here the glass cylinder extends about half-way up the gauze, and above it is fixed a metallic shield, partly surrounding the upper portion of gauze above the top of the glass. For the purpose of firing the shot the glass is lifted sufficiently to allow the wire to be inserted between the meshes of the lamp-gauze, in order that its temperature may be raised in the flame. When the wire has been heated and withdrawn, the glass is lowered, and, together with the upper shield, it forms a great protection from the force of the air currents.

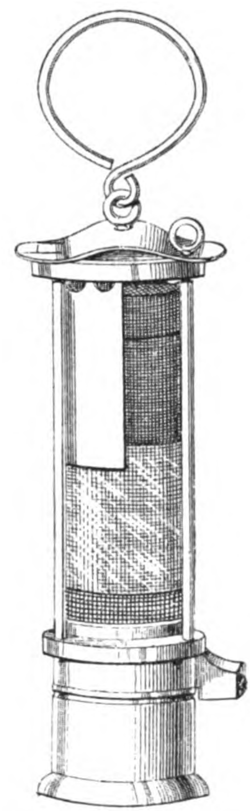


Fig. 516. Davy Lamp with Glass Cylinder and Half-round Upper Shield.

The daily examination of the mine before work begins is often made by the help of ordinary safety-lamps. The Davy, protected by a bonnet and a movable glass on the outside of gauze (to comply with the requirements of the 9th general rule of the Mines Act, 1887), is usually preferred by the examiners to other forms of lamps. In order to make the cap more distinct and thereby aid the observation it is usual to reduce the flame in size by drawing down the wick with the pricker in low percentages of firedamp. The luminosity of the flame being then less, its elongation, or the pale blue light of the cap, becomes more distinctly visible. An estimate may thus be made from the appearance of the lamp flame of the quantity of firedamp present in the air. The amount of firedamp present must exceed 3 per cent. for the cap to be distinctly visible with an ordinary gauze Davy lamp. Lower percentages cannot be detected by it, nor can 3 per cent. with certainty. With 5 per cent. of firedamp present, the cap reaches to the cover of the lamp, where it spreads out and fills the upper part of the wire gauze cage for a depth of 2·6 inches from the top. The result of these experiments will be modified with Davy lamps having gauze cylinders of varying diameters.

The *Stephenson* Lamp.—In 1812, George Stephenson, the eminent engineer, obtained an appointment at the Killingworth Colliery, about 7 miles to the north of Newcastle-on-Tyne. In August, 1815, he requested Mr. Nicholas Wood, who commenced his apprenticeship as a colliery viewer at Killingworth Colliery, in the year 1811, to prepare a drawing of a lamp from a description which he gave him. There is no doubt, therefore, that Stephenson was experimenting with lamps simultaneously with Sir Humphry Davy. The result was the invention of a safety-lamp, the characteristic feature of which was a glass cylinder about two inches in diameter, surmounted by a cap of perforated copper. The feed-air was conveyed under the glass cylinder by means of small tubes. On Oct. 21, 1815, Stephenson's first lamp, which had only one tube to admit air, was tried at a blower in Killingworth pit, and burned well. On Nov. 4 of the same year a second lamp was tried. This had three capillary tubes to admit air, and in consequence it burned better than the first. On Nov. 30 of the same year, Stephenson's third lamp was tried in the mine and found to be safe, and to burn well. In this lamp the air was admitted under the glass by a double row of small perforations. These lamps had no iron wire gauzes.

The question of priority as between the Davy and the Stephenson safety-lamp has been a subject of bitter controversy. It is clear that the subject occupied the attention of the two great men at the same time, and each simultaneously produced a lamp on much the same principle. Great credit is therefore due to both inventors. To Sir Humphry Davy's researches, however, we are indebted for the discovery of the principle of safety found in the iron wire gauze, which still constitutes the means of security in modern lamps, and which Stephenson himself adopted as a means of protecting his glass cylinder, in the place of a perforated cover over the glass previously used. The application of glass to safety-lamps was condemned by Sir Humphry Davy, on account of its liability to breakage.

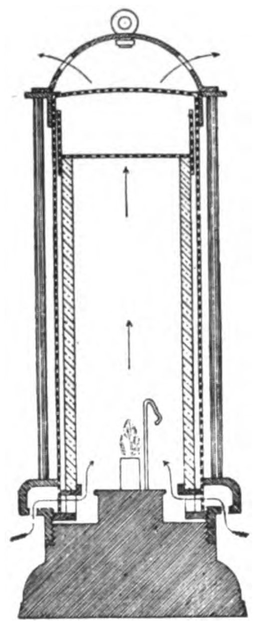


Fig. 517.—The Stephenson Safety-lamp.

The Stephenson or "Geordie" lamp, as it is sometimes called, as at present made is shown in Fig. 517. The cylinder of glass there shown is about 5 inches in height, and rather under 2 inches in diameter. It is covered by a cap of perforated copper, and placed within the wire gauze cylinder, which is slightly over 2 inches in diameter, and about $6\frac{1}{2}$ inches in height. A smoke gauze cap, which is a short wire gauze cylinder, placed over the top of the main gauze, is sometimes added to the main gauze cylinder, which it then overlaps for about $1\frac{1}{2}$ inches. The glass cylinder generally rests loosely on a brass ring, and can, by a jerk, be shifted vertically along the gauze, the amount of play differing in lamps from about $\frac{1}{4}$ to $1\frac{1}{4}$ inches. In some varieties of the Stephenson, the glass cylinder is fixed in its place by the gauze being slightly nipped in a ring, or by its reaching to the top of the gauze cylinder. The lamp weighs about 2 lbs. without oil or wick. The air to supply the flame is admitted through a number of small holes of about $\frac{1}{20}$th of an inch in diameter, formed in the under side of a hollow collar, and then through the main gauze which is kept in place by a ring below as indicated by the arrows. The ring of metal keeps the bottom of the glass cylinder about $\frac{1}{4}$ of an inch above the bottom of the main gauze. It is perforated with large openings to admit the air under the glass after it has passed through the main gauze. Unless these inlet perforations at the bottom of the lamp are kept free from dust and oil they soon become choked, and prevent the lamp from burning well. Occasionally the metal top to which the handle is attached is a perforated dome, but more often a bridged top is fitted, either of which allows the products of combustion to escape after passing out of the main gauze.

In an explosive atmosphere which is still or has only a low velocity, owing to the air entering from below not containing sufficient oxygen for the support of the flame, and, owing to the glass above the flame being filled for the most part with the products of combustion, the flame is soon extinguished. This may happen without igniting the mixture at all, or there may be a feeble burning at the bottom of the lamp. The quantity of gas burning is too small to raise the temperature of the gauze to a dangerous degree, but it may break the glass at its lower edge. With a broken glass a Stephenson becomes practically a Davy lamp.

In a strong current of inflammable gas and air, however, the force across the gauze above the glass cylinder causes a current to descend on one side of the glass, while the products of combustion are ascending on the other. Under such circumstances the gas will ignite at the top as well as at the bottom of the lamp, thus causing the gauze to become red-hot, and igniting the external gas mixture. In currents having a velocity of from 9 to 12 feet per second the Stephenson lamp becomes unsafe.

The *Clanny* Lamp.—Dr. William Reid Clanny's first lamp, which is often mentioned as the father of all safety-lamps, has been already referred to. After Sir Humphry Davy's discovery, iron wire gauze was applied to the lamp which had been devised by Dr. Clanny as well as to that of George Stephenson. Previous to this, it is probable that Clanny had adopted some modification of his lamp which he explained in a paper read by him on Aug. 19, 1839, before the members of the South Shields Committee. The Transactions of the North of England Institute of Mining Engineers, vol. xvii., p. 37, contains a description of the introduction of a short cone or cylinder of safety gauze, within the glass cylinder, and reaching to a little below the top of it so as to surround the flame. An illustration of the Clanny lamp as at present made is shown in Fig. 518. It consists of a lower cylinder of stout glass surrounding the flame, and an upper cylinder of wire gauze of less diameter. The feed air passes through the lower part of the wire-gauze cylinder, then down the inside of the glass cylinder, the products of combustion ascending inside the cold air-currents, and escaping through the upper part of the wire gauze. The oil-holder, with pricker, is the same as in the Davy lamp. Protecting bars are placed vertically around the glass and others around the gauze above. The main gauze is surmounted by a smoke gauze cap, and at the bottom is flanged to fit against a brass ring over the glass. The greatest care is required in fitting the glass cylinder. At its top and bottom sometimes washers of soft leather, indiarubber, asbestos or other suitable material are interposed between the glass and the metal rings, asbestos being the most effective. There is a great difficulty in maintaining the joints air-tight at all temperatures to which they are exposed. If no washers are used and a firm connection is made, the difference in the expansion between the metal and the glass is liable to cause a breakage when the lamp is burning; if the connection is made too loosely, a dangerous opening may be left between them. If washers are used the application of heat may cause them to shift their position and require frequent renewal unless asbestos is used. The glasses must be carefully selected, only those with truly parallel bearing surfaces and free from chipping should be used. A good washer cannot remedy defects in the glasses or

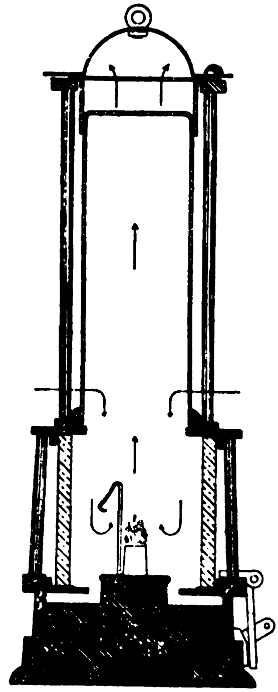


Fig. 518.—The Clanny Safety-lamp.

flame, the products of combustion passing through the cone-gauze in the interior of the lamp, and up through the top shield.

The outer shield has five horizontal rows of circular holes punched in it, the inner six horizontal rows of slits. The openings in one shield are opposite the solid portions of the other so as to prevent direct entry of the air which takes the course shown by the arrows. Of the three gauzes used the outer one is a cylinder without a top, the middle one is a Clanny type, and the inner one is curiously constructed, it being made up of an outer cylinder with a top which carries an inner short one carrying a metal chimney to improve the draught. The products of combustion have to pass through two gauzes and through openings in the top of the bonnet. Bars are placed round the glass cylinder to protect it from injury and connect the upper and lower rings.

When tested in an explosive current containing 8 per cent. of gas with a velocity of 47 feet per second, this lamp was found quite safe and went out in five seconds without exploding the external mixture. It is highly probable, therefore, that the lamp is capable of successfully resisting explosion in the mine in any velocity of current likely to be met with. The weight of the lamp without oil or wick is about 3 lbs., so that it is slightly heavier than some other lamps which are perfectly safe under ordinary conditions. The light given by it is good and steady in a still clear air, but strong currents reduce its illuminating power. It is not extinguished by a moderate amount of tilting or oscillation.

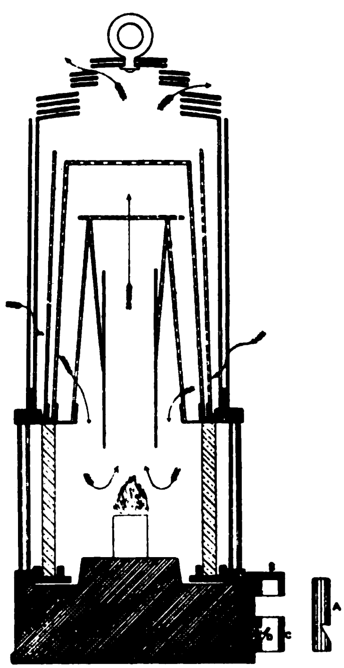


Fig. 519.—The Morgan Safety-lamp.

The lock for this lamp is a lead plug which differs in form from the ordinary lead rivet, and may be more quickly dealt with. There is an opening through the projection C on the oil chamber and a recess in the upper projection B. When the oil chamber is properly screwed on, the two vertical holes are opposite each other. In the lower is a small spring catch, D. The lead plug, A, is cylindrical, but has a portion cut out for the spring to fit into as shown in the illustration. The plug A is made to fit accurately in the openings C and B. To lock the lamp it is passed vertically upward through the lower opening until it is prevented from going further by the closed end of B, when the lower extremity is level with the bottom of the opening in C. In its upward progress it forces the spring D back out of the way, and when fully inserted the spring falls out under the cut portion of the plug. This prevents it from being drawn back out, while the covering at B arrests its further upward movement. To be unlocked the lead plug must be cut.

The *Protector* is also a good lamp. In it colzaline is burned instead of the vegetable or animal oils used in other kinds of lamps described. It makes no smoke or soot, will not clog in the gauze, does not require any pricker, and very readily shows the presence of inflammable gas. Inside the Protector lamp-bottom is a sponge which, in trimming the lamp, simply requires to be saturated with the spirit, the superfluous colzaline being poured out again. A permanent asbestos wick is placed in the upper part of the wick-tube, and the liquid absorbed in the

the lower openings any explosive mixture near the roof or in holes above the roof is discovered, whilst in sluggish currents, the air may be admitted through the lower openings in the tubes. When both top and bottom openings in the tubes are closed at the same time, the supply air is shut off entirely, and the lamp goes out immediately afterwards. This may become necessary when testing for an explosive mixture. The glass instead of being cylindrical in form is larger at the base, the top being the same size as the coned bonnet and gauze above it. A conical gauze cap is used in the Hepplewhite-Gray instead of the gauze diaphragm in the Gray shown in Fig. 522.

A shield outside the gauze cap extends from the glass to a point slightly above the gauze, and terminates in an outlet cone which reaches to the level of

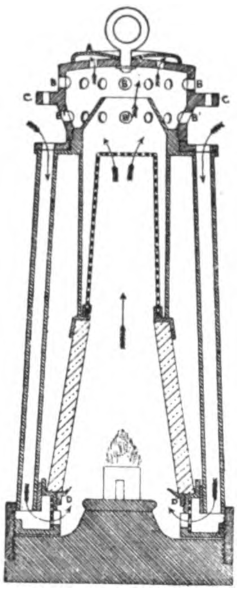


Fig. 523.—Hepplewhite-Gray Lamp.

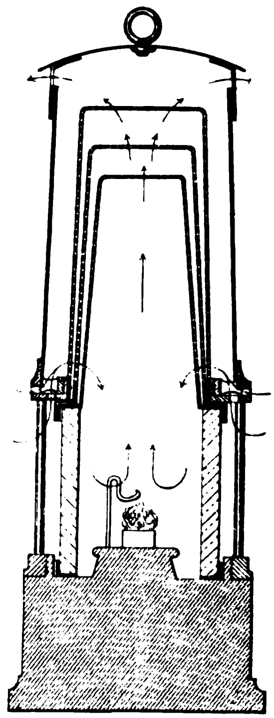


Fig. 524.—Marsaut Lamp.

the shield-plate C., its upper extremity being mid-way between the two horizontal rows of perforations B, B′, by which the hood is pierced for the escape of the products of combustion. The shield plate C is of a size to cover the inlet openings. The circular row of holes in the crown of the lamp are of the same size as those in the shield-rim. A stiffened cover-plate is placed over, to protect the products of combustion at their point of exit from direct currents. By these additions and improvements the discharge is regulated as well as the inlet so that they bear what is considered a proper proportion to each other. The restriction of the discharge when the lamp is in use causes the top of the gauze to be kept in a bath of carbonic acid gas, so that if an internal ignition of gas takes place it will not continue in the lamp. The first test for fire-damp should be made with this lamp when either one or both of the shutters over the holes near the base of the inlet tubes are pushed up, leaving the holes open. The searcher then slowly and steadily raises the lamp, without lowering the wick flame until the presence

The *Mueseler* lamp is constructed on a similar principle to the Clanny. It has a short cylinder of thick glass round the flame, above which is a gauze cap, Fig. 525. Immediately above the flame and extending within the gauze cap is a central conical metal chimney, the top of which in some forms of Mueseler lamp is covered with wire gauze, but in the Belgian pattern is open at the top as well as bottom. The chimney is carried by an attached ring of gauze fixed to its outer circumference between the top of the glass cylinder and the bottom of the gauze cap.

The feed air, as indicated by the arrows in the drawing, passes under ordinary circumstances first through the lower part of the gauze cap and

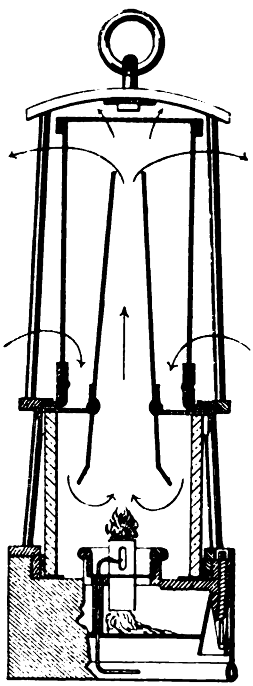


Fig. 525.—Mueseler Lamp.

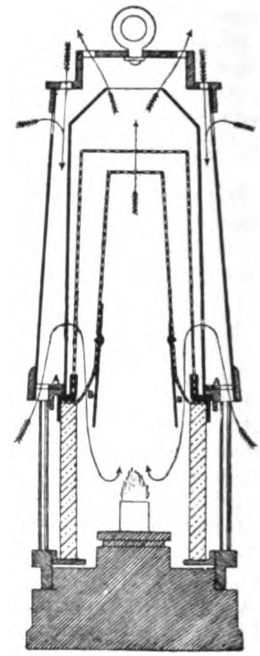


Fig. 526.—Ashworth-Mueseler Safety-lamp.

then downwards through the horizontal gauze ring, continuing its course afterwards between the metal chimney and the glass cylinder. The inlet air has thus two obstructions to its passage, the outlet only one, if the chimney is not covered with gauze. The object of the chimney is to create a strong upward draught and to insure the inlet air being drawn down close to the inside of the glass cylinder and thus keep it cool. A great drawback is its liability for the light to go out if the lamp happens to be held with its axis at a slight angle to the perpendicular, as in that position the flame drives back the inlet air on one side and baffles the current. Fig. 525 shows the Mueseler lamp as made in Belgium, where it is constituted the legal lamp for use in fiery mines. The base of the chimney is ·85 of an inch above the top of the wick-tube. The gauze diaphragm which supports the chimney is 1·9 inch above the wick-tube. The height of the chimney is 4·6 inches, its diameter at the bottom being 1·15 inch, narrowing to 1 inch at the junction of the main cone, ·25 above the base, and diminishing to ·4 inch at the top. The diameter

In another section of the Commissioners' Report appear the following remarks:—" This lamp seems very safe in all currents in which it was tested. In air moving with any velocity up to 3,500 feet per minute the flame burns very steadily when the lamp is either erect or inclined. The flame is scarcely affected by violent oscillations of the lamp or by rapid motion up and down in a vertical direction, and it is not extinguished by inclining the lamp until the latter is nearly horizontal."

The *Clifford* lamp, as shown in Figs. 528—530, is new since the publication

VERTICAL SECTION

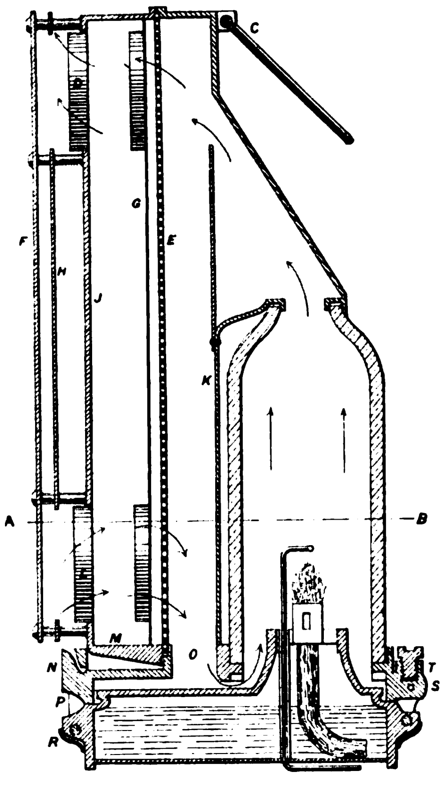


Fig. 528.

TRANSVERSE SECTION ON LINE A.B

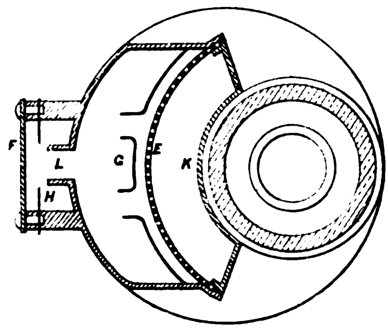


Fig. 529.

PLAN OF OIL CHAMBER AND ATTACHMENT

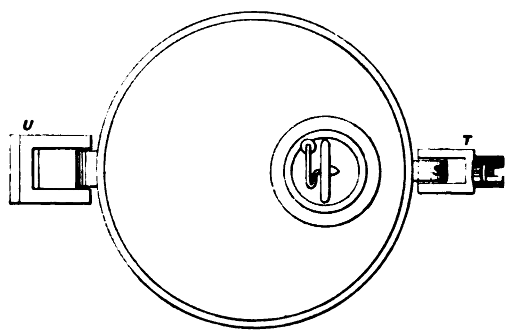


Fig. 530.

THE CLIFFORD LAMP.

of the Commissioners' Report. It is to be called the Phœnix lamp, is novel in construction, and is designed to give great resistance to the passage of flame in high current velocities. In its construction, the usual form of having a glass cylinder surmounted by a cylindrical gauze has been entirely departed from.

Fig. 528 is a vertical section; Fig. 529 a transverse section on the line A B; and Fig. 530 is a plan of the oil chamber and attachments with the upper portion of the lamp removed. The glass cylinder surrounding the flame is firmly held in position by asbestos washers. The upper portion is contracted in diameter and is covered by a metal hood, which forms part of a segmental pillar or box, glasses. The products of combustion ascend the metal chimney C, which leads directly from the inner glass, through the top of the main gauze A, which leads from a ring above the outer glass, through openings in the top of the metal shield which encloses the whole.

A heavy mineral oil having a specific gravity of not less than ·83, with a flashing point of 250° Fahr. closed, Abel's test, is burnt in a cone similar to those used in paraffin lamps, and is said to be as safe as animal or vegetable oils, or to the mixture prescribed by the Mines Commission, and to burn without smoking. A flat wick is employed, and the height of the flame is regulated by a simple rack and pinion movement controlled from the outside by means of a small key fitting the square shaft of the pinion. A pricker is required to keep the wick of the usual types clean, but as no carbon cap forms on the wick of the Thorneburry lamp, the services of a pricker are dispensed with. A very superior light is given by the lamp, which will burn for 14 hours in actual use in the mine with the flame at a uniform normal height.

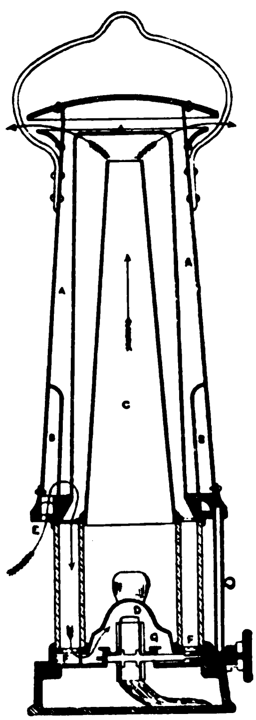


Fig. 533.—The Thorneburry Safety-lamp.

Objections have been made against the use of lamps with two concentric glasses on account of the inner glass being so near the flame that the additional heat from an internal explosion is almost certain to crack it, thereby rendering the lamp much less secure, and also on account of the difficulty in making accurate and tight joints with them. In cleaning the lamp the glasses have usually to be taken out and much time is occupied in refitting the parts of a large number of lamps. To overcome the latter objection the glasses in the Thorneburry are left undisturbed when the lamp is taken apart to clean, and, with regard to the former, the cylinders are of specially-prepared virgin glass properly annealed. This, it is said, unless subjected to sudden changes of temperature, may melt but will never crack by the application of heat.

An internal explosion of gas extinguishes the lamp flame, after which the ignited gas may remain for a few seconds burning at the wire gauze ring F protecting the air inlet, but this in turn soon becomes extinguished owing to the nitrogen and carbonic acid gases given off by combustion in the cone. Slow currents do not cause dimness as in ordinary lamps, and it bears the swinging about incidental to the use of a safety-lamp in the mine. It is simple in construction, containing no more parts than an ordinary Mueseler, which facilitates cleaning, and the gauze and chimney can be seen without unlocking the lamp. The lamp has been tested in horizontal and inclined explosive currents, the velocities of which varied from 10 to 50 feet per second, with the result that it was always automatically extinguished without firing the external mixture.

An objection to the lamp is its weight, which when ready for use is 3 lbs. $10\frac{1}{4}$ oz. It also gets very hot if allowed to stand in a still atmosphere.

Marshall's Automatic Extinguishing Lamp.—This lamp, shown in Fig. 534, is new since the Report of the Mines Commission, 1886, and is remarkable for the ingenuity of its design. It requires no bonnet, so that the gauze cylinder is fairly open to view, and a partially unobstructed examination of the roof can be made with it if long glasses are used. An outer "sleeve gauze" cylinder fits closely within the glass or chimney before the lamp can be opened; a check screw X is screwed into the oil vessel through a hole Y at the last turn of the oil vessel, so that one turn of the oil vessel is required before the check screw X can be extracted, thus extinguishing the flame; a screw-hole Z, shown in the enlarged illustration, Fig. 536, for the check screw when it is not in use. If desired, this screw, which is more difficult to open without a proper key than an ordinary screw lock, can be discarded for lead rivet locking, or the use of the extinguisher and the mechanism actuating it may be dispensed with in favour of a lead rivet lock, thus considerably reducing the cost of manufacture and complication in fitting up when cleaned.

When the explosive mixture ignites within the lamp, it burns inside the gauze-covered inlet ring surrounding the wick tube, and therefore must burn the string loop U, which is consumed in a few seconds. The wire T is thus released; the spring S instantly turns the cap R over the chimney top, and at once extinguishes all fire by shutting in and throwing down the carbonic acid gas. As an additional precaution an ordinary screw-lock is fixed where shown at the side.

The two rings, K, are made in one casting with connecting pieces between them. They support the chimney in place, and prevent it from tilting, while the horizontal gauze, G, causes the tension and allows the inner glass to expand with the heat. A close joint between the top and bottom ends of the inner glass, B, and the metal rings, is thus always maintained under all changes of temperature. The rings, K, and chimney with extinguisher attached, are so combined by a screw ring as to form one fitting only. The outer glass is kept cool by the feed air.

There is no smoke gauze cap over the main gauze, which is of copper, as is also the round wick tube. All gauzes used are of ordinary mesh, having 784 apertures to the square inch. The upper half of the chimney is made of copper, while the lower half is of brass. The outer glass A is nearly $\frac{1}{4}$ of an inch thick, the cylinder being $2\frac{3}{4}$ inches in diameter outside. The inner glass B is $1\frac{3}{4}$ inches in diameter outside. The outer glasses are 3, $3\frac{1}{2}$, 4, and $4\frac{1}{2}$ inches long; with the longer glasses, the chimney and upper parts are shortened. The length of the inner glasses varies from $1\frac{3}{4}$ inches to $3\frac{1}{4}$; they should be well annealed. A stock of string loops must be kept ready for use, but a new one is not required every day. They are easily prepared for use by being tied upon a tapering peg nailed to a bench.

To clean the lamp, it is taken apart by unscrewing the bottom ring plate, when the whole of the parts come out together. In fitting up, a small "setting hook" is used for attaching the string loop U to the extinguisher wire T and hook V, the lamp being held upside down for the purpose. When the loop has been attached to the wire T, and is pulled slightly, the cap comes off the chimney. The loop is then secured to the hook on the lever in the position shown in the drawing.

In the unlikely event of the extinguisher failing to act, or in the still more unlikely event of the gas flame passing between the glasses, the collier can turn the lamp bottom upwards, and shut in the carbonic acid gas, which effectually extinguishes the fire-damp flame in from 2 to 6 seconds in currents of any velocity.

The weight of the lamp with oil is about $4\frac{1}{4}$ lbs., and it well stands the jerks, oscillations and swinging inseparable from its use in the mine without disturbing the light, which remains clear and steady in strong currents of air.

In explosive currents of great velocity the lamp is very safe and has been tested in currents varying from 15 to 50 feet per second without failure. The lamp was exposed to horizontal explosive currents, to currents inclined upwards, and to currents inclined downwards, and at the expiration of a few seconds, the combustible connection with the spring was burnt, the extinguisher operated and put out all flame, without an external explosion taking place, and without cracking he inner glass, which is able to resist a much greater heat than that necessary to supported and are kept steadily in place by a screw ring which is passed over the gauze cylinder and screwed into the top of the gauze-covered inlet ring.

The bottom of the shield is riveted to a ring which screws to the lamp frame. This screw ring supports four pillars, which are bent inwards above the shoulder formed on the shield to carry the top of the lamp and the handle. The close joint between the lower end of the upper half of the shield and the inlet ring, together with the flange on the latter, effectually separate the ingoing air from the outgoing gases. A round wick-tube is used, and the oil chamber is screwed on to the upper portion of the lamp in the usual way, but the locking device is a great improvement on the screw lock. It consists of a pillar lock P, which locks both the oil vessel and the shield. A hasp, R, is fixed to the pillar P, and to lock the lamp this is dropped over the projecting boss O. This can only be done when the pillar P is raised or lowered to the proper level, and then the upper end passes through a hole in the flange, which carries the shield so as to prevent it from being unscrewed, while at the same time the lower end enters partly into a hole bored in the rim of the oil-chamber. The projection O has an eye in it for the reception of the lead rivet when the hasp is turned down sufficiently for the purpose. If desired, a padlock may be substituted for the lead rivet, or a screw lock hasp can be hinged to the pillar P, it being slotted to pass over the projecting boss O, and the point of the screw locking sideways through the eye in the projecting boss O. Screw locks and padlocks are, however, easily picked and re-locked. To open the lamp, the lead rivet is removed which allows of the hasp R being raised clear of the projecting boss, and then the pillar lock P can be raised and lowered so as to permit the oil chamber and shield to be unscrewed. The oil vessel and shield rims can have a series of holes or longer slot holes in them to compensate for wear of their screw-threads. A single glass is used, and in order to overcome the danger of its working loose when screwing on the shield, a screw-lock S is provided in the lower ring of the lamp. The point of this screw when locked is against the side of the angle ring used for fixing the glass cylinder, thus removing a great source of danger found in some safety-lamps. An asbestos washer is placed above and below the glass cylinder.

Resting on the flange which supports the shield is a vertically movable steel band N, which in one position is in close contact with the lower half of the shield to a point just below the inlet holes where the band ends. This forms the "shut-off" to be used when gas testing, if there should be an internal explosion. The shut-off is raised by hand until it covers the inlet holes so as to prevent further entry of the gaseous mixture, following which the lamp is extinguished, or the same effect is produced by turning the lamp bottom upwards. The locking of the shield and oil vessel may be left until the last moment so as to give the examiner or the collier an opportunity to unscrew the shield and examine all the gauzes which are then exposed to view, and if necessary to tighten up the main gauze. With a little ingenuity the gauze and glass may be cleaned and examined without removing them. When the shield and oil vessel are unscrewed, a hand brush can be passed through the chimney to clean the gauze, or the chimney and main gauze can be removed from the top when the shield is unscrewed without removing the glass, which may remain in until it is broken or cracked. An advantage claimed for this lamp is that if tilted the flame burns low, and will therefore not crack the glass; upon righting the lamp the flame recovers itself. The lamp is very small and handy. In the smaller of two sizes made, the glass cylinder is 2 inches high and 2 inches in external diameter, the thickness of glass being $\frac{3}{16}$ths of an inch. The weight of this lamp without oil is $2\frac{3}{4}$ lbs., and the height is about $8\frac{1}{2}$ inches to the flat top. A size larger has a glass cylinder $2\frac{1}{4}$ inches high, $2\frac{1}{4}$ inches in outside diameter, and weighs without oil about $3\frac{1}{4}$ lbs. The smaller lamp has been tested in explosive currents of great velocity, which it withstood without failure.

and sucking it. The same thing may be done with an unbonneted Clanny, if the wick be first raised so as to abnormally enlarge the flame by sucking the pipe against the gauze immediately above the glass. Possibly the knowledge of these facts made it seem useless for many years to introduce better locks, but now there are many good lamps, the shields and locks of which are not so easily tampered with.

For the most part, however, they afford little more security than the original device of the screw-lock. Many of them can be opened by easily extemporized mechanical means, while others are rather more difficult. It is easy to understand the temptation to tamper with the lock which the miner feels, whose lamp burns faultily or is accidentally extinguished. The legitimate method of opening and of readjustment may be available only on the surface, and that possibly a mile away, involving the loss of an hour's work, besides the toil of going to and fro. But even this, while it explains, will of course never justify the picking of a locked lamp in a fiery mine where the momentary exposure of a naked flame may cause a fearful explosion involving the loss of hundreds of lives, and a vast destruction of property. The difficulty would be met, and the temptation indefinitely lessened, by having relays of lighted lamps at stated places which would be available to the miner in case of need.

Most of the improved locks are supposed by the inventors to require the locking bolt to be withdrawn by the aid of an air-pump, or powerful magnet or water-pressure, it being previously kept in position by a spring. Some even of these, however, can be overcome without the proper appliances.

Purdy's Lamp Lock.—Purdy's lamp, of which the Royal Mines Commissioners in 1886 speak highly, has a pneumatic lock of ingenious device, which the inventor thus describes :—" A small cylinder is fitted at the side of the oil vessel, having a piston and rod, or bolt, which passes through and projects above the rim of the oil vessel. This rod or bolt is held up by a spring, after the principle of a cornet valve. When the top of the lamp is screwed on to the bottom, the piston bolt becomes inserted in a recess in the upper neck of the lamp, which comes opposite to the bolt at the moment of contact of both parts of the lamp. For the purpose of unlocking, a hollow screw is fitted in the side of the cylinder, which can be withdrawn only so far as to admit a small hole in the side of the hollow screw becoming accessible to an air-pump. When it is desired to draw down the bolt and unlock the lamp, the hollow screw is partly withdrawn and inserted in the mouth of the air-pump, by one stroke of which the air is exhausted, and by a simultaneous turn of the upper part of the lamp the two parts are disconnected."

Craig and Bidder's Lamp with Magnetic Lock.—This is one of the oldest, if not the first, of the magnetic locks for safety-lamps, it having been patented in 1869, and is described in the Transactions of the North of England Institute of Mining Engineers, Vol. XIX., p. 15-16. One arrangement of the lamp lock is shown in Figs. 539 and 540. The gauze frame is screwed on to the oil vessel A, by the ring C, in the usual way. This ring has a hole, B, bored partly through the under side of the metal, so as to receive the bolt D, attached to a piece of soft iron F, and pressed upwards by the spring J. At the last turn of the oil vessel screw, the bolt is forced by the spring into the hole B. In the brass bottom of the oil chamber M are two circular pieces of iron, HH, let into it, which correspond to the two poles KK, Fig. 541 of an electro-magnet LL. To unlock the lamp, the circular pieces HH are placed on the top of the corresponding circular projections KK, Fig. 541 of the magnet LL. The electric circuit can be completed by means of a knob placed conveniently for the operator, which, when contact is made, renders the projections KK the poles of a powerful extending vertically from the bottom of the lamp through the oil vessel to a level with the wick, and insulated from the lamp and the oil. A fine platinum wire, D, is stretched across and secured to the upper part of the rods CC, one of which can be partly rotated, thus carrying the platinum wire to touch the wick or to be withdrawn from it. This pole is prevented from being turned too far by means of a cam-shaped collar which comes into contact with the outer wick-tube. The poles CC terminate below in two flat metallic heads BB, which fit on two corresponding metallic flat heads AA fixed on a table top. The heads AA are attachable to wires leading from a small battery. To light the lamp, after it has been trimmed and locked, or before locking, it is placed on the table with the heads BB resting on the metallic portions AA, thus completing the circuit through the platinum wire D, and the flowing electric current heats the platinum wire sufficiently to light the wick. On removing the lamp the platinum wire is moved clear of the influence of the lamp flame, which can be adjusted in the ordinary manner.

The electricity used is of low tension, capable of producing only feeble sparks, and the battery used is small, portable and inexpensive. The appliance is shown in the Fig. on a Clanny, but is applicable to almost any kind of lamp, and as the platinum is raised to a white heat any kind of oil may be used. The apparatus is intended for lighting lamps at lamp stations in the workings by an official in charge, and not inside at points nearer the working places on account of the danger of sparking when the lamp is connected to the battery. The lamp stations, therefore, should not be made any nearer to the working faces than is quite safe for the use of naked lights. Whenever a lamp becomes extinguished in the workings it should be carefully examined before being re-lighted by the official in charge at the lamp station. If this can be done without unlocking, as may be the case with some unbonneted lamps, there may be great convenience in an apparatus of this sort, but with lamps in which the shield and oil vessel are locked at the same time by a lead rivet, a proper examination can only be made by taking the lamp to pieces, and little, if anything, is gained by the use of such a lighting arrangement, especially in small workings: indeed, its use may be fraught with danger, as there would be a temptation to light and afterwards use a lamp without the trouble of examination.

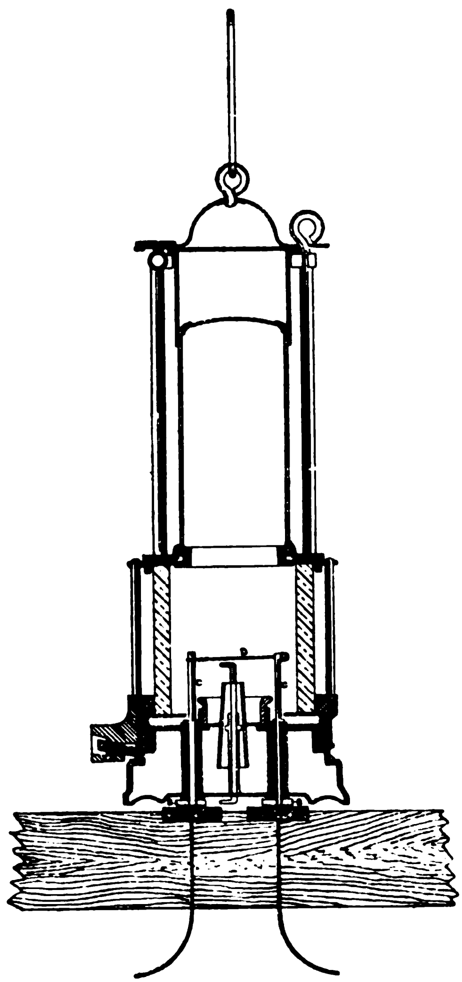


Fig. 551.—Illustrating Hann's Method of Lighting Safety-lamps while Locked.

glass. It is constructed with longitudinal metal joints, and is finished at each end by a short brass tube. Strips of mica are also used in Pendleton's lamp. Although the mica does not crack when the flame is allowed to reach it, if great heat is generated in the lamp it becomes brittle and, owing to the joints, untrustworthy. It is not likely, therefore, that mica will supersede glass for the cylinders surrounding the flame.

Cleaning Lamps.—At small collieries having only a few safety-lamps in use the cleaning is usually done by hand, without the aid of mechanical contrivances to unscrew or remove the internal fittings. An ordinary hand lamp brush is used to rub the dirt from the gauzes. If the gauzes are smeared with oil the brushes are sometimes dipped into powdered magnesian limestone, or the gauzes are steeped now and again in a solution of caustic potash. An examination of the wires is made during the cleaning by holding the gauze cylinders to the light at intervals. Dirty patches thus revealed to view are rubbed with the brush, while the discovery of twisted wires or dents in the gauzes leads to their removal.

For the glasses and any metallic portions not cleaned by brushes and suitable compositions cotton waste is used or suitable cloths. On fitting the parts together again, the washers, where necessary, are renewed so that there is no slackness of glasses. Safety-lamps are sometimes taken home to be cleaned at the close of each shift by those using them, after first being unlocked in the lamp-room on the surface. In some instances only the tops are taken home by the men, the bottoms being left for the lamp-keeper to deal with. Where the cleaning is performed in a properly appointed cabin it may be done by a qualified staff of lamp-keepers, or by each user cleaning his own lamp; but this last plan is open to grave objections, as the number of brushes and the accommodation in the lamp-room are limited, and the men, already tired with their day's work, and irritated at having to stand about and await their turn to use the brush, hurry over the work as quickly as possible instead of cleaning their lamps thoroughly.

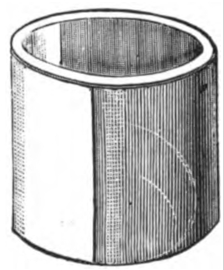


Fig. 553.—Patent Reflector Glass.

Even when the workmen clean their own lamps there is at least one lamp-keeper at each colliery to replenish the oil-vessels, renew the wicks and replace worn washers or broken glasses, except in the very rare instances in which the men themselves supply all safety-lamps and afterwards maintain them in a perfect state of repair. In order to comply with the Mines Act, 1887, there must always be an examiner appointed to see that the lamps are in safe working order, which they cannot be if any parts are omitted, and that they are securely locked before being taken into the workings. The user should also examine the cleaned lamp he receives from another person to make sure for himself that all its parts are in proper order and that the lamp is well trimmed.

Some collieries have as many as 5,000 safety-lamps in daily use, comprising lamps of different types for gas-testing, shot-firing, or general use. These large numbers must seriously tax the energies of even an efficient staff of keepers. Their labours, however, are lightened by the use of special machines for cleaning safety-lamps, by means of which lamps are rapidly unscrewed, cleaned, and put together again. Fig. 554 shows Wolstenholme's safety-lamp cleaning machine, supplied by Messrs. John Davis & Son, Derby. It is provided with a table 4 feet 6 inches by 2 feet 6 inches and is self-contained. The motive power may be obtained through the belting from the fast and loose pulley, shown at K, or a self-contained engine of the "Brotherhood" type can be fixed to revolve the shaft; or if the colliery is provided with electric current, a little selfon their arrival in the morning may not have far to go out of their way, and should not be blocked by other buildings or obstructions to the daylight. The lamps, moreover, will often have to be cleaned by artificial light, which must be good. The staff of workmen must be both sufficient and efficient. They should be careful and experienced men, who will ensure the lamps being properly tended and securely locked before being sent out. The walls of the cabin are fitted with narrow shelves properly numbered on the outside edge to receive correspondingly numbered lamps as they are brought back. An entry of each owner's name is kept in a book opposite his number. A bench in the cabin is necessary for cleaning on, whether this is done by hand or by machinery. The number of windows will depend upon its size. There may be two, three, or more in order to pass the lamps out to the men quickly. The passage outside leading to and from each window should be formed to compel the men to file past one by one and be barred by a self-registering turnstile securely locked when not in authorized use. The number of men as ascertained thus each day should correspond with the number of lamps taken out. At times, unfortunately, it becomes of great importance to know the exact number of men descending the shaft in any particular shift, as well as their names and places of residence. When the workmen take both top and bottom portions of their lamps home in order to clean them, it becomes more difficult to know who are at work in any particular shift.

Examining and Testing Lamps.—Modern lamps are more complicated in their construction than the first type, and the number in use is vastly increased. The work of examining these lamps at large collieries is, consequently, considerable. In many cases the lamp is half covered with a bonnet, hiding from the attendant's view the most important parts. The large number which must be taken to pieces before they can be examined, and afterwards restored to working condition, entails much labour, and it is possible that some small but important part may be omitted in the operation. It is very difficult for the eye to detect defects in the adjustment, and impossible—without resorting to other means—to know whether the joints between the glass cylinder and the gauze are perfect under the varying temperatures at which a lamp is worked. If a thoroughly reliable and simple means of testing could be devised, it would be well to test each lamp before it is taken into the pit. At some collieries this is done daily in the lamp-room by placing the lamp (lighted), after examination, in a mixture of gas and air previously ascertained to be a properly explosive one. This test should be slow and deliberate, and the lamp allowed to remain some time in the mixture.

The Royal Commissioners draw particular attention to this point. They say: "With the safer and more complicated lamps it is still more difficult to detect by eye defects in adjustment. Indeed, in many instances important parts cannot be seen after the lamp has been put together. During the course of our experiments explosions have been occasionally traced to imperfections so small that we have had some trouble in finding them even when certain of their existence. Experiments 723–725 of the Woolwich series are examples of this, as will be seen by comparison with those following them. We consider it, therefore, absolutely necessary that such lamps should be regularly tested in an explosive gas mixture before they are allowed to descend the shaft." In another part of their report:—"A lamp may be of the safest pattern, and yet small defects in the fitting of its parts may entirely deprive it of its power of affording protection. In preparing a large number of lamps for use in a mine it may happen even with the greatest care on the part of the lamp-man that a lamp in an imperfect condition may be allowed to pass. The detection of these imperfections by simple inspection is in many cases almost impossible. And we are convinced that the only way of avoiding the introduction into a mine of a dangerously imperfect lamp diffusion of two gases into one another depend upon their relative specific gravities. Thus the diffusion of hydrogen into marsh gas would take place more quickly than that of marsh gas into hydrogen; marsh gas into air more quickly than air into marsh gas; and air more quickly into carbonic acid gas than *vice versâ*. Each gas would exchange places with air according to a diffusion equivalent which can be calculated. In the case of air of uniform temperature, although diffusion would take place, the volumes passing in opposite directions would be equal, and the pressure inside the vessel would remain unaltered. If the vessel is filled with air and surrounded by a mixture of air and marsh gas at the same pressure as the air inside the vessel, the mixture passes more rapidly through the porous substance into the vessel than air passes out of the vessel. As a result of the inflowing volume being greater than that of the air flowing out in a given time, the pressure within the vessel is temporarily increased. Air mixed with fire-damp is lighter than air alone, the mixture becoming lighter in proportion to the percentage of fire-damp. The pressure within the vessel then varies according to the percentage of fire-damp contained in the external atmosphere. By observing the greatest pressure reached in the vessel, having previously noted the pressures produced by surrounding atmospheres known to contain certain percentages of fire-damp, an approximation to the percentage of fire-damp present at any observation is supposed to be obtained. If carbonic acid gas be present in the atmosphere instead of fire-damp, the mixture is heavier than air alone, and the diffusion which then takes place diminishes the pressure inside the vessel. Ansell's indicators are designed to show variations of pressure resulting from the property of the diffusion of gases, such pressures being made apparent in different ways.

The instrument is made in different forms suited to give warning either of a slowly accumulating quantity of fire-damp at a goaf, or of sudden irruptions of the gas, the increase being indicated either by telegraphic signals or by a syphon-gauge.

The particular form of Ansell's indicator, however, which has been most used, consists of a pocket aneroid barometer of the most delicate construction enclosed in a chamber the brass back of which has been removed and replaced by a plate of unglazed earthenware. This plate is sufficiently porous to allow of diffusion between the air inside the aneroid and that outside, but it prevents the entrance or escape of gas or air by mere mechanical pressure. The pressure on the outside of the vacuum chamber in the aneroid acts on a spring, the motion of which is multiplied by a system of levers and so causes a hand to travel over a dial face, which is graduated to inches as in the ordinary aneroid. The porous plate is protected by a brass cover to prevent diffusion taking place at undesirable times. To test for gas, the instrument is taken into the mine, and on reaching the place of trial it is first held by the ring handle until it has become of the same temperature as its surroundings. The brass cap is then removed from the porous plate, and the instrument is held in the suspected atmosphere. In about 45 seconds the maximum pressure is attained, and the index hand must then be read, because after the maximum point is reached, effusion or the passage of gas through the porous plate takes place, and an intermingling proceeds rapidly until the air on both sides of the partition has become similarly mixed with gas and is of uniform composition. All indications of pressure in the instrument then cease to appear and the index-hand travels back to zero. If, in this instance, say 2 per cent. of fire-damp was indicated by the instrument when the maximum pressure was reached, and it be now taken to another place in the mine, where there is, say, 3 per cent. of fire-damp present, only 1 per cent. will be indicated by the index-hand, and in time that indication will disappear because the intermingling here will result in the instrument being eventually filled with air containing 3 per cent. of fire-damp. The 1 per cent. shown on its face must be added to the 2 per cent. previously in the instrument to give the proper amount for the second observation.

The Forbes Damposcope.—This is the invention of Professor George Forbes, and is intended to determine the percentage of gas in the atmosphere by measuring the length of that part of a tube containing the air to be tested which gives the maximum resonance to a musical note of standard pitch. The length of the column of air in the tube depends upon its density, and this fact is made use of as a means of calculating the density of the air in the mine. The instrument consists of a brass tube 1 inch in diameter, and between 6 and 7 inches long. A tightly-fitting piston can be moved up and down the tube by means of a rack and pinion. The graduated pinion-head is 3 inches in diameter, and its complete revolution causes the piston to move 1 inch. An index attached to the tube over the graduated pinion-head enables readings of the piston movements to be taken. The tube has an open end, in front of which is fixed a tuning-fork, capable of vibrating 512 times per second. This tuning-fork is first set in vibration when the instrument is in pure air, and the piston is moved backwards and forwards until in the position which gives the strongest resonance. The index reading is then recorded, or, if preferred, it can be adjusted to zero. The tube is then filled with the atmosphere to be tested, the tuning-fork again vibrated, and motion given to the piston, until that position is found which again gives the most intense resonance. The graduated pinion-head is again read and compared with the previous reading, thus showing the increase or decrease in the length of the column from the open end of the tube. If the temperature has remained unchanged, and the air tested contain only marsh gas, the percentage of gas is calculated. A thermometer is fitted to the instrument, which, it is thought, to some extent enables corrections to be made to readings if changes of temperature occur, or percentages of gas corresponding to the value of the scale are recorded for different temperatures in a table based upon earlier experiments with air containing known percentages of gas.

The indication of gas by this instrument is subject to the same sources of disturbance and error as the Ansell aneroid, notwithstanding the attempts to apply corrections. There is also some difficulty in adjusting the piston in the exact position which produces the strongest resonance, and untrained observers will probably adjust differently. The necessity to ascertain and make allowance for the causes of disturbance before a proper estimate of fire-damp can be obtained by these instruments, makes them too complicated for ordinary use in the mine.

The Hardy Detector.—This is another acoustical indicator in the use of which the composition of the atmosphere is supposed to be determined from observations of the beats produced by two sounds nearly in unison, so that it does not require to be used by a trained observer with a sensitive ear, as in the case of the Forbes damposcope. When two streams of pure air are directed at the same time from two distinct organ-bellows to two organ-pipes, giving the same note, the sound produced is of a single note. If, instead of fresh air, one of the bellows be supplied with a mixture of air and gas, one organ-pipe does not give its natural sound, but a note modified by the column of air and gas in the tube. The two pipes thus acted on simultaneously give two sounds which are not in unison, and, therefore, distinct beats are produced. These beats are more or less frequent in proportion to the gas in the air, and are easily recognized and counted by an unpractised observer.

The Hardy detector consists of two separate bellows and two organ-pipes. One of the bellows and its organ-pipe are enclosed in an air-tight compartment containing pure air; the other bellows receives its supply from an atmosphere to be tested, which it directs to the other pipe. An experiment may be made in a few seconds. The organ-pipes both sound the same note in pure air. If the mixture under examination be composed of air and marsh gas, the following results are obtained :—

photometer through glass plates, which close the wire-gauze tube M and the metal tube N.

In the base of the instrument is a small magneto-electric machine. By turning a handle on the outside of the box which encloses the apparatus, an electric current is produced, and this passes through the two precisely similar platinum wire spirals, heating them to moderate redness. An increase in the velocity of rotation of the handle increases the electric current in the wires, and correspondingly raises their temperature and increases their brilliancy. The rate of rotation may vary within certain limits, but should be such as to heat the spirals to dull or moderate redness.

Fig. 562 shows the indicator being employed in the mine, where it has accurately shown the presence of fire-damp ranging from 0·25 upwards. When constructed, the scale of the instrument is divided to suit two precisely similar spirals, which in pure air are rendered equally bright by the same current. In its new condition the instrument is capable of giving accurate results for high and low percentage tests. After being in use a comparatively short time, however, the wire spiral which is exposed to the gas becomes permanently altered in its electrical resistance. The scale divisions then give erroneous percentages of gas present in the atmosphere tested, and it becomes necessary to set the zero of the scale afresh in air free from gas. The scale admits of being shifted to a slight extent, and when adjusted, the indications of gas it gives, although not strictly accurate, are sufficiently so for practical purposes. A difficulty, however, arises from the fact that the above alteration in the spiral is constantly increasing, and thus requiring re-adjustment. It may not be easy to obtain fresh air for the adjustment, if that becomes necessary in the mine, and, if the wire spiral undergoes considerable change during one round of testing, the adjustment of the zero may be impossible owing to the limited range of motion allowed to the scale. In that case two new spirals must be inserted in the place of those in the instrument, and this usually causes serious delay.

Fig. 562.—Liveing's Indicator in use in the Mine.

An objection to the use of the Liveing Indicator in the mine arises from the fact that it furnishes no light, and is so much dead weight to be carried about by the firemen or other official, in addition to a safety-lamp for lighting purposes. After being in use some time there has been difficulty in obtaining, by means of the magneto-electric apparatus, a current of sufficient steadiness and constancy to ensure accurate readings. In unskilful hands the handle may be too rapidly rotated and thus produce a current capable of fusing and rendering useless the exposed platinum spiral. A less rapid rotation of the handle may bring about the same result if much gas is present.

Lewis and Maurice's firedamp indicator (the invention of Sir William Thomas Lewis and Mr. A. H. Maurice) is shown in Figs. 563–566.* E is an air chamber, containing about 2 cubic inches of air; in it the mixture of

* See Transactions, South Wales Institute of Mining Engineers, vol. xv., pp. 39—43.

the platinum wires during a test, would depress the liquid in the gauge, but this is obviated by bringing the enclosed air in J under the same heating influence as that in E. Both may expand, but they do so equally, so that the gauge is not affected when a test is made. Thus when the cap is unscrewed from the bottom of the instrument, the level of the liquid may be disturbed in the same way as that in a thermometer, but whatever level it assumes, after screwing on the cap with the test sample of air enclosed, the index finger is made to agree with it by sliding it along the scale.

The indicators of Monnier, Coquillion, and Le Châtelier belong to the same class as that last described, all depending upon the contraction in volume or the change of pressure effected by the combustion of the marsh gas present in an enclosed volume of air. The complete combustion of the fire-damp present in the air would reduce the original volume of the mixture of air and gas by very nearly twice that of the fire-damp present. The decrease of volume, instead of being measured, is usually indicated by the decrease of pressure it causes in a closed vessel connected with a suitable gauge.

Professor Monnier's instrument is too costly and too complicated to come into general use, but is a very ingenious invention.

M. Coquillion's consists of three glass tubes connected together, the central one being of small bore and graduated. Above it is a glass tube of larger diameter closed by an air-tight cover in which is a small stopcock. Two insulated platinum wires traverse the tube, and these are connected inside by a fine platinum wire. The glass tube at the other end of the graduated tube is attached to a small pear-shaped india-rubber bag. The glass portions have an outer brass casing with slits on opposite sides for reading the scale. Water is poured into the instrument until it fills the bag and rises to a certain mark on the graduated scale.

To make a test with the instrument the stopcock is first opened, and on the bag being squeezed with the hand the water rises up the tube and expels the air contained in it; on releasing the bag the water returns to its former position and the upper tube of the instrument becomes charged from the surrounding atmosphere. To ensure the effectual expulsion of the original volume of air, and that none be left to dilute the atmosphere to be tested, this process is repeated several times, after which the stopcock is closed. The position of the surface of the water in the graduated tube is then read and the fine platinum wire is raised to a white heat by a current of electricity produced by a battery or a magneto-electric machine, which continues for two or three seconds. This operation is repeated two or three times at intervals. The fire-damp in the instrument is thus consumed; the resulting aqueous vapour is condensed when the instrument cools, causing a decrease in the pressure in the tube above the water. The atmospheric pressure on the bag then causes the surface of the water in the graduated tube to rise. The height of the water is now read and compared with the previous reading, so as to enable the observer to calculate the percentage of marsh gas present in the atmosphere.

It takes five minutes to make a single experiment with this instrument, so that it could never be used largely in the mine. Moreover, the indications are rendered inaccurate by the exposure of the air containing the products of combustion to water. The carbonic acid gas produced by the combustion of the fire-damp is soluble in water and may be partly or wholly dissolved according to the proportion present and the length of exposure. The results, therefore, are doubtful.

Le Châtelier removes this source of error by substituting mercury for water. He further quickens the period of the experiment by surrounding the test-vessels with a water-jacket. The products of combustion are thus more quickly cooled, and the periods of waiting for the true pressure readings are lessened.

These forms of apparatus are not well adapted for underground use, because head, held in place by a bowstring, against a gong in the chamber, and the sound can be heard by the observer. The apparatus is large and heavy, and is adapted for laboratory, not underground use. The sample of air is captured in the mine by means of a diaphragm hand-pump and a 6-gallon rubber bag, and is brought to the gas-tester. The Shaw gas-tester is an extremely sensitive instrument, and in its normal condition is capable of giving indications of great accuracy and delicacy. It is said that the addition or subtraction of only 0·1 per cent. of fire-damp will cause the bell to ring or remain silent. The lowest kindling proportion of the inflammable gas which is to be measured when mixed with air must first be determined by the apparatus. A calculation of the proportion of inflammable gas present in the sample of air and gas tested can then be made by the operator first reading off the proportion of air or gas actually added to and mixed with the sample under examination.

The apparatus is complicated and costly, and will probably require constant attention to be maintained in a state of efficiency. The collection and transport of samples of mine air are attended with considerable inconvenience and delay, and there is also risk of leakage or change of composition in the samples. By the time such samples reach the testing apparatus the condition of the atmosphere in the mine may have undergone a change, and no inference can be drawn of the state of any one point at the particular moment the test is being made. The apparatus is certainly very ingenious, but simpler and more portable forms of fire-damp indicators are to be obtained.

The Flame-test class of Detectors are certainly the most convenient forms of apparatus. They can be used at once in the mine with safety, and experiments can be made with them with ease and rapidity. The presence of carbonic acid gas, or of aqueous vapour within the limits in which these occur in the coal mine, do not appreciably interfere with flame-tests, and the results obtained are not appreciably affected by the variations of pressure and temperature of the air of the mine under ordinary conditions.

The Davy lamp has long been a means of detecting gas by means of the flame-cap. A very useful adjunct to the Davy and other lamps has for several years been in use in *Garforth's Ball Detector* (see Fig. 568).

The Royal Commissioners thus speak of Mr. Garforth's invention:—

" For extracting the gas from crevices Mr. Garforth uses a small hollow india-rubber ball provided with a metal nozzle. The ball being compressed in the hand so as to expel the greater part of the air contained in it, the nozzle is inserted into the crevice or cavity and the ball is then allowed to expand to its natural size. It of course becomes filled with the gas mixture existing in the place under examination, and this gas may be introduced into a safety-lamp through a gauze-protected channel by again compressing the ball. For thus testing the gas a bonneted lamp is used, through the oil cup of which passes, close to the wick tube, a narrow pipe containing gauze diaphragms, and terminating below in a tube which just fits the nozzle of the india-rubber ball. The external end of this tube is closed by a spring valve, and when the nozzle is introduced it opens the valve and the gas can be forced up the pipe and on to the lamp flame. This simple and extremely portable apparatus seems to answer its purpose perfectly, and the addition to the lamp of the pipe above-mentioned does not affect either its security or its illuminating power."

This detector may be very useful in obtaining samples of gas mixtures from holes above the roof too small for the insertion of a Hepplewhite-Gray lamp, but it is incapable of detecting percentages of firedamp below 2 per cent. in the air, and does not measure or register any proportion of gas and air. Its use is likely to decrease now that several lamps are made to take their air supply from the top.

visible when there is from 1 to $1\frac{1}{2}$ per cent. of fire-damp present up to 6. Percentages of gas lower than 1 are likely to be confounded with a halo or false cap always visible over the benzoline flame, but it is said that $\frac{1}{4}$ per cent. of fire-damp can be detected by the lamp.

Professor Clowes gives the following as the cap heights observed in experiments made by him with the Ashworth lamp, having a flame 3 mm. high (see Fig. 571).

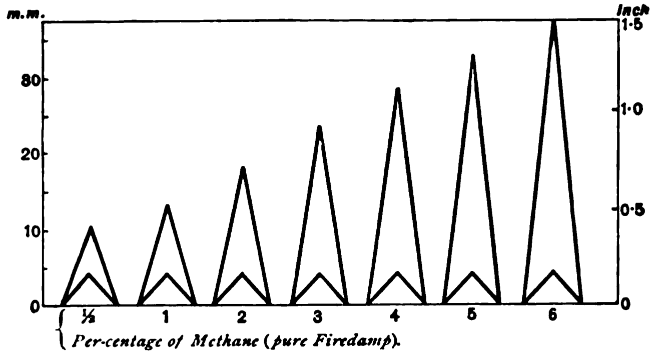


Fig. 571.—Flame and Cap (Benzoline). (Actual size.)

The heights of the test-flame and of flame caps are as follow :—

Percentage of Methane (Fire damp) present.	Ashworth's Benzoline Flame, height 3 mm. or 0·12 inch. Height of Cap. mm.	inches.
0·5*	7*	0·28*
1·0	10	0·40
2·0	14	0·56
3·0	20	0·80
4·0	25	1·00
5·0	30	1·20
6·0	35	1·40

The presence of fire-damp causes the full-size benzoline flame to spire more distinctly than the oil-flame. This test is made by raising the flame until it is on the verge of smoking. If this be done and the lamp raised into a gaseous mixture an elongation of the flame follows. The elongation becomes more pronounced as the percentage of gas present increases, but any lengthened flame at once commences to smoke. This spiring is due to diminished supply of oxygen to the flame, but it cannot be made use of with certainty for detecting the presence of gas.

The Chesneau Lamp.†—This is a lamp invented by M. Chesneau to burn a large alcohol flame for gas testing (see Fig. 572). The reservoir for the alcohol is of brass, formed with a circular crown for the admission of air through double

* This indication is very doubtful; it cannot be trusted.
† See "Transactions of the Federated Institute of Mining Engineers," vol. 4, pp. 617, 618.

This is a suitable safety-lamp, furnished with special fittings to receive an appliance for delicate and accurate gas-testing. The appliance and its adaptation were designed by Professor Clowes and other gentlemen associated with him.

The form of safety-lamp employed is open to selection, but Professor Clowes prefers the Ashworth-Hepplewhite-Gray lamp, because he considers it to be one of the best existing forms of gas-testing safety-lamps.

The invention is largely the outcome of very careful experiments undertaken in 1891, by Professor Clowes, with the view of ascertaining the delicacy and accuracy of gas-testing safety-lamps then existing, and of deciding which is the most suitable test-flame for the purpose.

Professor Clowes first designed a special form of apparatus, or "test-chamber," which is particularly suitable for the observation of the caps which appear over

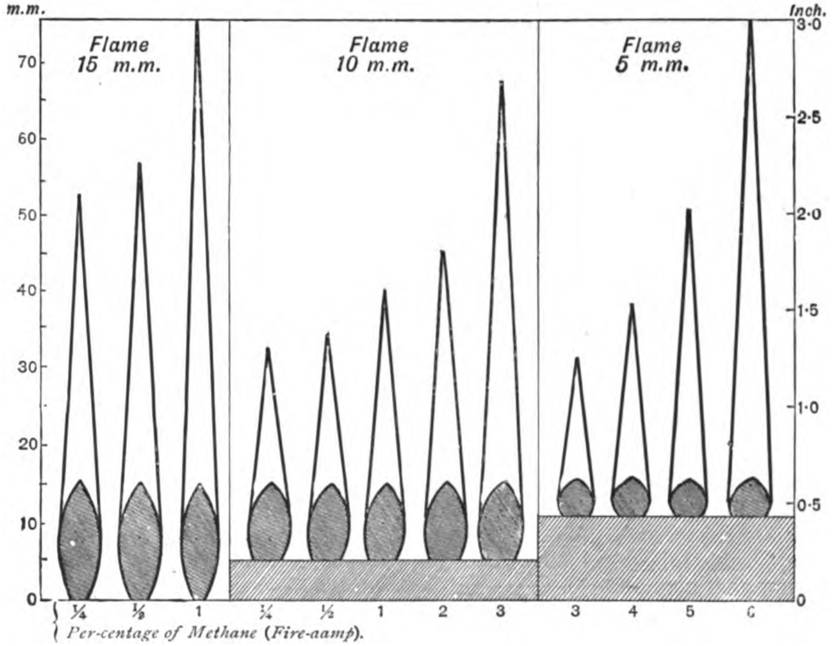


Fig. 577.—Actual Height of Caps over Hydrogen Flame

different test-flames. This apparatus will be fully described later on. By the aid of this "test-chamber," he was enabled to observe and measure with accuracy the caps which were produced over the ordinary flames of oil, benzoline, and alcohol, in some of the lamps already referred to, when they were exposed to mixtures of air with fire-damp. Similar observations were afterwards made upon the caps seen over a naked hydrogen-flame. The result of the latter series of observations is shown in Fig. 577. A comparison of the effects produced convinced Professor Clowes that the hydrogen-flame was superior to all other flames for gas-testing purposes. Its advantages are the following:—(1) Owing to its non-luminosity, the flame never interferes with the perception of the caps, and the observation of the caps can, therefore, be made without shielding the hydrogen-flame from the eye. (2) It is the only safety-lamp flame which is fed without

pp. 265, 367, 375; vol. 6, p. 177, and vol. 7, p. 2. Also Journ. Soc. Arts, Feb. 17, 1893; and Cantor Lectures, Soc. Arts, Jan., Feb., 1894; Proc. Roy. Soc., vol. 52, p. 484.

size, inasmuch as the hydrogen flame will only be used at such times as a reduced oil flame does not indicate the presence of gas. The cylinders retain their store of gas without leakage or loss for an indefinite length of time. The tests made when the cylinder is nearly discharged are quite as effective as those made when it is just replenished, but in this case the standard flame drops rapidly after it has been set.

The oil-chamber is supplied with a mixture of colza oil and water-white paraffin, the paraffin being introduced to prevent charring and crusting of the wick, while the admixture is safe in use and gives a better light than colza oil alone.

The weight of the brass lamp with oil and all fittings, except the hydrogen cylinder, is nearly 4 lbs.; the simpler form of brass lamp weighs 3½ lbs., and the hydrogen cylinder about 14 ounces. The lamp fully charged weighs 1 lb. 11 oz. when made in aluminium, and the hydrogen cylinder with aluminium fittings weighs 12 ounces.

The lamp is not intended for common use in the mine by the working miner, but only for those whose duty or desire it is to make examinations for gas. It may be most usefully applied to ascertain the state of the return airways, and for the regular examination of the working places before each shift begins work. A few of these lamps at a colliery will probably be found to be sufficient. For detecting percentages of gas above 3 and up to 6 the reduced oil flame is recommended. Higher measurements than 6 per cent. are unnecessary, for from 6 to 7 per cent. of gas becomes explosive and dangerous. For testing and measuring percentages below 3 down to 0·25, or even to 0·1 per cent. when requisite, the hydrogen flame is used by attaching the hydrogen supply and changing from the oil to the hydrogen flame. The hydrogen flame is pale, so that it requires no screening from the eye, and in fact assists the eye in detecting the cap. Even to a colour-blind observer it is quite distinct from the cap, but in order that the latter may be seen more clearly, a dead-black permanent surface has been produced upon the interior of the back of the glass behind the flame. A standard of 10 millimeters (0·4 inch) is used for measurement of gas from 0·25 to 3 per cent.; if higher percentages are to be indicated a standard hydrogen flame of half that height is employed. Gas may be detected and measured when it is present in proportions from 0·1 right up to 6 per cent., by means of the hydrogen flame alone, if the flame is reduced to 5 mm. or 0·2 inch in height for percentages from 3 to 6 per cent.; the indications for the lower percentages may also be increased, if desired, by using a flame 15 mm. or 0·6 inch in height (see Fig. 577); this increase is, however, unnecessary, since the caps over the standard flame, with even 0·1 per cent. of gas and upwards, are clearly seen by even an inexperienced observer.

Since the lamp was first used it has been noticed that the reddish-purple tinge of the hydrogen flame changes to pale blue in the presence of carbonic acid gas in the air. The change of colour is distinctly perceptible when only 2 per cent. of this gas is present, and it becomes more and more pronounced as the proportion increases. The flame is, therefore, a much more delicate indicator of carbonic acid, or "choke-damp" than the ordinary oil flame; the oil flame begins to diminish in size only when 10 per cent. of this gas is present, and is extinguished by 15 per cent.

This lamp, therefore, fitted with the hydrogen gas-testing apparatus, is not only used for ordinary illumination in the mine, but may be employed without any other apparatus to ascertain the presence of low percentages of gas in the air of the mine. The Pieler lamp will measure percentages of gas varying from 0·25 to about 2, but when it is used it is carried in the mine as an extra lamp beside that used for illumination.

The Ashworth-Hepplewhite-Gray safety-lamp flame is scarcely affected in air-

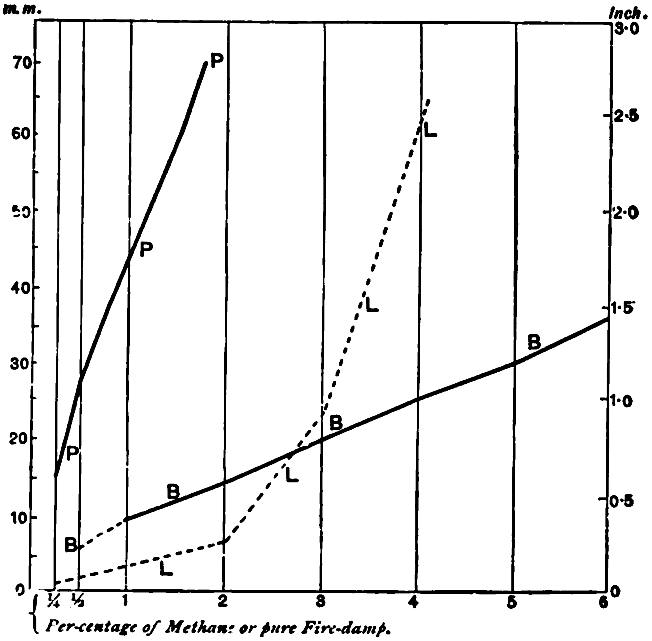


Fig. 587.—Fire-damp Indications to Scale.

P = Caps over Pieler Alcohol flame (one-half height).
B = Caps over Ashworth benzoline flame (full height).
L = Liveing's electrical indicator, relation between light emitted by covered and exposed spirals.

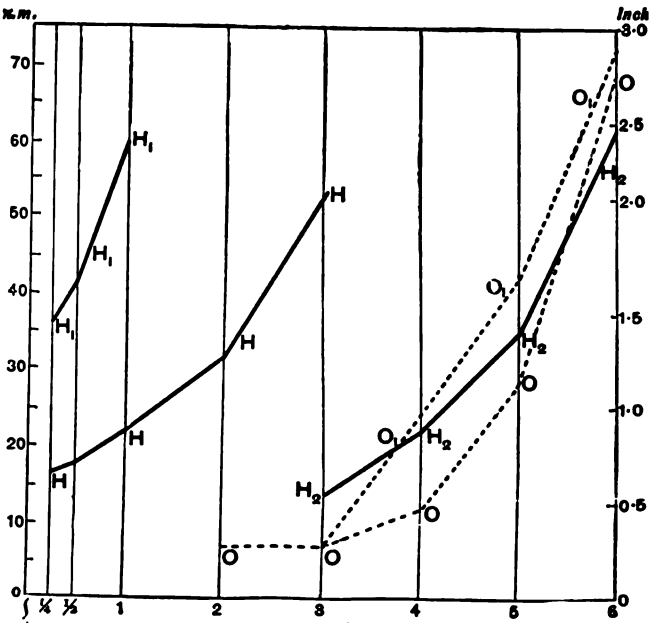
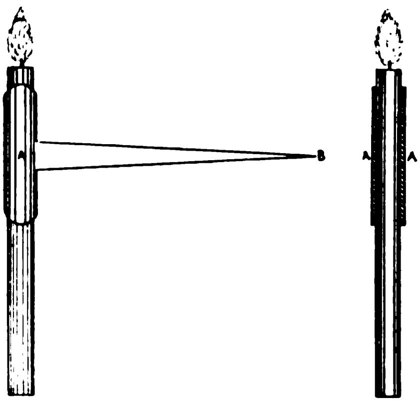


Fig. 588.—Cap-heights with Hydrogen Oil-lamp. (Actual Size.)

H_1 = 15 mm. hydrogen flame. H_2 = 5 mm. hydrogen flame. O = Pale blue oil-flame.
H = 10 mm. hydrogen flame. O_1 = Maximum oil-flame.

made into candles in the tallow-chandler's shop, from whence they are sent out to each pit, there to be given out to the colliers immediately before they descend the mine. About 20 candles go to the pound, and about 4 suffice for an 8-hour shift, the precise number depending upon the state of the air. The candle is generally held in a candlestick, so constructed that it can be carried through underground galleries in the hand or in the cap, see Figs. A and B, 590, and 591.

The candlestick is made of iron or copper, and consists of two similar halves, Fig. 590. The socket A is roughly curved to press against the surface of the enclosed candle, while the shank is flat and made to diminish in size, until it ends at the point B. The two similar portions are soldered or brazed together near the point B, but are free to open at the socket, the two halves of which may be separated sufficiently to allow of a candle being passed in between them, after which they close over upon the candle, and hold it firm. When work is begun underground the candle may be placed in a loop sewn in the cap (see Fig. 591), or the pointed shank may be stuck into the timber, roadside, or floor, where it is shielded



A. B.

Fig. 590.—Candlestick used in Somersetshire.

Fig. 591.—Candle carried in Leather Cap.

from the current of air. In travelling through small airways or around the faces of workings in a seam 1 foot thick, there is a great advantage in carrying the candle in the cap, so as to have both hands at liberty to assist in crawling. In strong air-currents, however, the candle-flame is blown from an upright position, and in burning rapidly melts the tallow on one side. This causes great inconvenience, as the hot grease streams down the candlestick to the cap, sometimes reaching the face, and if the candlestick becomes highly heated, as it will if the user neglects to raise the top of the constantly diminishing candle above the socket when required, the candle itself drops right out of the socket to the floor. Where a candlestick is not used, the candle is carried by means of a ball of clay, which may be moistened from time to time as it becomes dry, and thus will adhere to the timber, road-side, or floor, when it becomes necessary to free the hands. In the metalliferous mines of Cornwall, where the air-currents are not strong, the miners frequently use a ball of moistened clay to secure a tallow-candle firmly in their hats.

In the North of England boy-drivers are frequently employed on the main trolley-ways or horse-roads. These main roads are usually intake air-courses with strong currents of air flowing along them, and if a driver uses a candle the flame is often protected from the wind by what is called a "mistress." This is a rough wooden box-shaped case, consisting of a back shade of thin board about 8 inches × 4 inches, and having side and end pieces fixed to it. The space in front is larger than the board forming the back, so that the sides and ends

CHAPTER XVI.

SUNDRY AND INCIDENTAL OPERATIONS AND APPLIANCES.

Coal-Dust—Watering the Underground Roadways—Explosives and Blasting Operations—Gunpowder—Gun-cotton—Tonite—Nitro-glycerine—Dynamite—Useful Work performed by Explosives—Blown-out Shots—Johnson's Tamping Plug—Charging, Stemming, and Firing Shots—Shot-firing Safety-lamps—The Lauer Detonator—Experiments with Wooden Plugs for Tamping—The Water Cartridge and Accessories—Sand and other means of Protecting Cartridges—Tamping with Wet Moss—Roburite—Bellite—Carbonite—Securite—Ammonite—Ardeer Powder—Westphalite—Lime Cartridges—Wedges for Coal-getting—Macdermott's Rock and Coal Perforators—Ingersoll Hand-power Rock Drill—Ingersoll Machine-power Rock Drill—Gillott and Copley Rotary Coal-cutting Machine—Bower, Blackburn, and Mori Electrical Coal-cutting Machine—Stanley's Coal Heading Machine—Caging Appliances and Drop Staples—Pit Horses, their Food and Work—Fleuss Apparatus for Breathing in Noxious Gases—Fleuss Lamp—Exploring for Water—Underground Dams—Water-blasts—Underground and Surface Fires—Testing the Roof—Driving through Faults—Watt's Steam Indicator—Richards's Indicator—Use of Indicator Diagrams—Continuous Diagrams—The Thompson Indicator—Schäffer and Budenberg's Double Indicator—Bourdon's Pressure Gauge—Schäffer and Budenberg Bourdon Gauges—Steel Tube Gauges for Very High Pressures—Duplex Gauges—Graduating Ordinary Pressure Gauges—Graduating Steel Tube Pressure Gauges—Bourdon Vacuum Gauges—Schäffer Diaphragm Gauge—Testing Vacuum Gauges—Lightning descending Shafts—Dunford and Emen's Patent Automatic Tub-greaser—Self-lubricating Pedestals for Colliery Tubs.

COAL-DUST.

Closely connected with the lighting and ventilation of collieries is the effect of coal-dust in promoting or increasing the disastrous effect of explosions. It is only of late years that attention has been to any extent directed to this question, but the experiments by Mr. W. Galloway (whose name will always be honourably associated with this matter), and those of the Royal Commission and others, prove beyond doubt that the existence of fine coal-dust in the underground roadways and workings of fiery mines is a dangerous element, and that it has played an important part in colliery explosions. Coal-dust consists of minute particles of coal, and when these are mixed with the air in the workings they appear to be liable to combustion. It is doubtful whether explosions of coal-dust occur on any large scale where the mine yields no fire-damp, although a blown-out shot may fire the coal-dust. It is stated, however, that if a small quantity of fire-damp be present and fired in this way, its effects are much intensified and the explosion is extended along the roads containing coal-dust, which thus feeds the flames of the explosion. Although there may only be a small quantity of gas, the flames, when thus started over a dusty road, are carried to a point far beyond that due to the simple explosion of the inflammable mixture of fire-damp and air. These facts are now fully recognised, and in dry fiery mines, where coal-dust is largely produced, it is becoming a general practice to remove the dust, and to moisten the roof, floor, and sides of the roads. Mr. Galloway says that 1 per cent. of fire-damp mixed with coal-dust and air forms an explosive mixture. Sir Frederick Abel states that from 2 to $2\frac{1}{2}$ per cent. of fire-damp in the mixture is necessary.

Since the publication of the official report on the circumstances attending the Camerton Colliery explosion, which occurred on the 13th Nov., 1893, it seems well established that coal-dust as found in the mine is itself explosive in the entire absence of inflammable gas, and under such conditions as may ordinarily be appears that this may be done by the use of specially made " spray-producers." From these the water issues in the form of spray or mist of such exceeding fineness as to be absorbed by the intake air to the point of saturation. This condition of the air is maintained by placing the spray-producers at suitable intervals along the roadway. A vertical branch pipe having an internal diameter of $\frac{1}{2}$ an inch is laid from the main road pipe to the roof and carried to the centre of the roadway. The pressure of the water may be from 100 to 150 lbs. per square inch, and issues downwards through the spray-producers, which are placed at intervals of about 50 yards along the roadway. Each branch pipe is provided with a regulating cock.

At collieries where compressed air is used, a better result in the system of damping the intake air will be obtained by the use of compressed air in conjunction with water conveyed in the pipes as just described. A main pipe is necessary to convey the compressed air along the main intake, but probably that is already laid to supply some machinery in the workings. At each spray-producer a $\frac{1}{2}$-inch branch pipe leads out of the main compressed air pipe and may be laid parallel to the water pipe branch. The compressed air pipe is connected to the water pipe by a nozzle in the interior of an ordinary T-pipe, which forms the junction of the air and water. The water is driven out by the air through an adjustable spray-producer, which is regulated by means of a nut and screw. Spherical valves are placed in both the air and water pipes to prevent the water from passing into the air pipes and also the air from escaping into the water pipes should any accident occur to the water main. The air and water issuing at the spray-producer become mechanically mixed, in infinitesimal globules, and as the pressure of the air gives considerable impetus to these, the moisture is carried further than with the globules of water unmixed with air. Consequently, where compressed air is used, the spray-producers may be placed at greater intervals apart, probably from 100 to 400 yards, to suit the circumstances of each colliery. A further advantage of the air-and-water system arises from the fact that a much lower pressure of water can be used. Indeed, it is only necessary for the water to have sufficient pressure, when throttled down, to find its way into the chamber where the compressed air meets it and drives it out. In shallow pits, where no head of water can be obtained, this may be a better means of laying dust than adopting some expensive plan to get sufficient head of water.

The working faces may be damped as well as the main roads by either the water, or the air-and-water system. The pipes are laid to the face and the spray-producers fixed there. In compliance with the Act of Parliament, no shot is to be fired in the presence of dust, but if the dust is destroyed by damping, for a radius of 20 yards from any shot-hole and no fire-damp be present, the competent man appointed for the purpose may fire the shot.

EXPLOSIVES AND BLASTING OPERATIONS.

Probably no mining subject has had more attention during the past few years than that of blasting. By the Mines Act, 1887, various new restrictions are imposed on the use of explosives, both as to the actual process of blasting, and the places where it is permitted.

The use of explosives is intimately connected with the lighting of mines, and their freedom, or otherwise, from gas and dust. Thus, wherever safety-lamps are used, shots can only be fired by or under the directions of a competent person appointed for the purpose. Similarly, wherever a mine is dry and dusty the same restriction is imposed. Further, where safety-lamps are used, the competent person may not fire a shot unless " he has examined the place itself, and all contiguous accessible places of the same seam within a radius of twenty yards, and has found such place safe for firing;" and if gas has been reported in the ventilating district at any of the four inspections recorded last before a shot is to be fired he must also examine the place or places, and see "that such gas has been cleared away, and that there is not at or near such place sufficient gas issuing or accumulated to render it unsafe to fire the shot," or if this is not the condition of the place as regards gas the shot can only be fired if it is "so used with water or other contrivance as to prevent it from inflaming gas, or is of such a nature that it cannot inflame gas."

Then, if the place is dry and dusty (without reference to gas), such place and all contiguous accessible places within a radius of twenty yards, must be thoroughly watered, or have treatment equivalent to watering, but if watering would injure the roof or floor, then the explosive may be used with water or other contrivance so as to prevent it from inflaming gas or dust, otherwise the explosive must be "of such a nature that it cannot inflame gas or dust."

A further restriction is applicable to any part of a main haulage road, or place contiguous thereto, showing dust adhering to the roof and sides. Not only must the place of firing, and within a radius of twenty yards, be watered, or have treatment equivalent to watering, but the explosive must be used with water, or other contrivance, so as to prevent it from inflaming gas or dust, unless it is of such a nature that it cannot inflame gas or dust. An alternative to this is, that one of the conditions mentioned must be observed, and all workmen—except those engaged in firing the shot, and others (not exceeding ten) employed in attending furnaces, boilers, engines, signals, or horses, or inspecting the mine—removed from the seam or seams on the same level.

Thus it will be seen that, with one exception, greater stringency is now required in the matter of blasting than formerly. Under the Act of 1872, where gas had been found issuing so as to show a "blue cap" on the flame of a safety-lamp, shots could only be fired in coal-work when persons ordinarily employed were out of that part (which was decided to mean the "ventilating district") of the mine. Now, if certain safeguards are employed, shots may be fired with all the men in the mine.

Where an explosive is required in the actual getting of coal, as in those cases where the reduced percentage of large coal and its enhanced cost with hand labour alone do not result in a fair profit, *Blasting powder* is still used, notwithstanding many objections. Its composition is given in Chapter XI. of this work.

The explosive force of gunpowder depends on the sudden formation of gases, chiefly nitrogen and carbonic acid, which, at the high temperature at which they are evolved, amount to about 2,000 times the volume of the powder employed. The granular form of the gunpowder increases the rapidity of its combustion, as the flame is better able to penetrate it, and thus kindle every grain almost at the same time. For this reason, in mining, where it is desirable that the combustion should be comparatively slow, the powder is coarse-grained. The temperature at which it explodes is about 600° F. The products of its combustion are given elsewhere. The combustion being comparatively slow, the pressure resulting from the expansion of the gases has more time to act on the mass, and, in coal-getting, to rend it without much shattering effect, thus admitting of larger coals being obtained than by the aid of a quicker explosive.

A great drawback to the use of gunpowder in mining is the large amount of flame emitted from the shot on ignition. This emission of flame is a thoroughly recognised element of danger in mines producing fire-damp, especially where they are dry and dusty; and even without fire-damp many mining engineers, managers and others, are firmly convinced that there is considerable danger in using powder where a place is dry and dusty. Without actually prohibiting its use, the Legislature has endeavoured to minimise the risk in both cases by the stringent regulations already referred to.

Cartridges made of compressed powder have been used during the last few years, the object being to obtain within a minimum volume a given store of power. These cartridges are very convenient, being purchased ready for use, and obviate the danger attending the making of cartridges from loose powder, which operation is often left for the miners to do at their homes. Where the shot-hole can be bored truly circular there is no practical objection to the use of compressed-powder cartridges. Numerous accidents have, however, happened with them, probably caused by the cartridges having exploded owing to the heat generated by excessive friction while being forced into holes either too small or not truly circular.

The Mines Act, 1887, prohibits explosives being taken into a mine except in cartridges, unless such mine is exempted from the clause by order of the Secretary of State. Although not expressed, probably it is the intention of the Act that the explosive shall not only be taken into the mine, but also be used in cartridges —unless the mine is exempted. The grounds for claiming the exemption are not stated in the Act, but apparently this provision is intended to meet those cases where it is more practicable to use powder in the loose state, and because of a difficulty in boring holes round enough for cartridges, which difficulty undoubtedly exists in certain kinds of rock, and even in fireclay containing nodules of ironstone.

Many modern explosives are nitro-compounds, and these may be reduced to two, viz., gun-cotton and nitro-glycerine. All the nitro-compounds have low points of ignition, ranging from a few degrees below 200° C.

In the manufacture of *Gun-Cotton*, cotton waste is steeped in a mixture of sulphuric acid and nitric acid. During this process three equivalents of hydrogen are removed by the oxidising action of the nitric acid, and replaced by three equivalents of nitric peroxide. After steeping, it is washed in water, and cleansed from acid, following which it may be stored, or converted into other compounds. Although its constitution is different from what it was when placed in the bath, its appearance remains the same.

Gun-cotton explodes at 400° F., and sometimes at a much lower temperature, and as gunpowder explodes at 600° F., gun-cotton may be fired on gunpowder without igniting it. It has double the explosive force of gunpowder, and has the advantage of yielding no smoke. Time, moisture, and exposure do not alter its qualities. It is exploded by detonation. Cotton-powder is gun-cotton reduced to a fine state of division.

Tonite is cotton-powder, with the admixture of a nitrate or similar body.

In the manufacture of *Nitro-Glycerine*, a similar chemical change takes place to that obtained in the manufacture of gun-cotton. In this case the nitrification of a liquid is effected, viz., glycerine. Glycerine is mixed with nitric acid, and the mixture allowed to fall or drop in a narrow stream into water, when the nitro-glycerine immediately separates.

The action of the nitric acid removes three equivalents of hydrogen, and replaces them by equivalents of nitric peroxide, and without apparent change in the material.

Nitro-glycerine in its liquid state is inconvenient to use. Moreover, it is highly dangerous while in this condition.

It is susceptible to ignition in two ways. If burned at the wick of an ordinary spirit-lamp, and the experiment be conducted by a skilful operator, it burns quietly and harmlessly. This experiment is too dangerous to be entrusted to any but skilled hands. Similarly, gun-cotton may be comparatively safely lighted by applying a flame to it, when it will burn rapidly without explosion.

inserted underneath the oil-chamber, and pushed up the brass tube till the end is level with the flame, and in that position it may be exposed at the opening, E. The cylinder, D, is drawn down by means of the pricker, H, the mouth applied to the blow-pipe, K, its cover removed by pressing the lever, L, after which the flame is blown on to the end of the fuse at E, and causes its ignition. The pricker, H, is released, immediately upon which the reaction of the spiral spring, G, causes the return of the cover, D. Care should be taken by the shot-firer not to withdraw the fuse from the lamp until certain that sparks will not be thrown from the end of the fuse on its removal from the lamp. An emission of sparks only ceases after a Bickford fuse has burned for a length of from 2 to 3 inches.

Messrs. Bickford, Smith & Co., Limited, of Tuckingmill, Cornwall, are the manufacturers of Bickford's safety fuses, which they make in numerous qualities adapted to every requirement of blasting. The safety fuse is employed to fire the charge in every kind of mining, quarrying, and subaqueous blasting, whether with powder or any modern explosive. It is usually supplied in coils of 24 feet, and must be stored in a dry place, and free from contact with oil or grease, which substances are prejudicial to its composition.

To those engaged in coal-mines, the *patent colliery fuse* will be of the greatest interest. This fuse is one of Messrs. Bickford's more recent specialties; it is designed to obviate the dangers attending the use of safety fuses in fiery or gaseous collieries, as recommended in the Report of the Royal Commissioners on

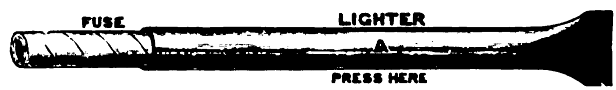


Fig. 602.—The Bickford Patent Safety Lighter.

Accidents in Mines (1886), and to meet the requirements of the Coal Mines Regulation Act, 1887. The chemical and other means employed in the manufacture of this fuse, render it practically smokeless, a desideratum in close places, but its speciality is that it does not emit spark or flame or highly heated gases laterally during combustion, and it may therefore be safely used with gelatinous or water cartridges in fiery mines. Closely allied with this patent is another recent invention of Messrs. Bickford, namely, their *patent safety lighter*, introduced to remove the difficulty of passing wires into safety-lamps, or of using special shot-firing lamps. This little instrument, shown in Fig. 602, effectually prevents the emission of spark or flame when the fuse is fired. The lighter consists of a tin tube having one end closed and the other open for the reception of the fuse. A small glass tube containing sulphuric acid is placed within the tin tube, the glass being in contact with a little chlorate of potash and sugar. One end of the fuse is inserted into the open end of the tin tube, and the mouth of the lighter is then closed firmly, but not too tightly, around it by pressing its open end with the Bickford nippers (see Fig. 603), the slots in the handles being made for this purpose. The fuse is then lighted by squeezing the centre of the tube with the nippers (see illustration). The grip of the nippers breaks the glass tube and releases the sulphuric acid which then ignites the chlorate mixture, and this in turn ignites the end of the fuse. The chemical action which takes place inside the tube produces no objectionable external effect, as all sparks are kept within the tube.

Messrs. Bickford also make an instrument which may, under certain conditions, be found a useful alternative to the patent colliery lighters, called the *Pistol shot-firer*. To use this, one end of the fuse is fitted with a specially-adapted

"The principal imperfections of this process are as follows :—

"1. A great number of shots cannot be fired so nearly simultaneously as by electricity, for this system is scarcely practicable for firing more than two shots at once.

"2. The Lauer detonators are 40 inches long; they are cumbersome and can only be carried with difficulty in thin seams.

"3. In straight drifts, screens must be erected about 40 yards from the face to avoid projected stones.

"4. It is feared that the manufacture of these detonators will not always be satisfactory. Several collieries in Austrian Silesia, where this system had been adopted throughout, have abandoned it, in consequence of defective supplies (this information is given by the director of the Karwin collieries).

"5. This system possesses certain dangers to which miners are unaccustomed. An unskilful workman may cause the explosion of a shot before he reaches the shelter."

The detonator is sunk into the cartridge, and the stemming is rammed round the tube enclosing the wire until it is embedded in the stemming.

Firing shots by means of electricity, although adding slightly to the cost, is doubtless the safest method that can be adopted. An electric fuze or "exploder" is required. It consists of some explosive compound which is capable of being acted on by an electric current so as to produce an explosion. One method of firing is by inserting in the exploder a fine platinum, iron, or alloyed metal wire, and connecting this with other wires in the circuit of a powerful voltaic battery. The fine wire not having sufficient conducting power offers sufficient resistance to the electric current to heat the wire to redness and thus cause an explosion of the compound in contact with it. Another method is to make a sudden discharge of static electricity take place between the terminals of two wires embedded in the charge. The passage of the sparks between the terminals causes the explosion of the fulminate, which then fires the shot.

An electrical machine for exciting and accumulating electricity, and conducting wires, are also necessary. If it is desired, a large number of holes may be fired simultaneously. The usual practice is for the whole apparatus to be placed in the hands of a competent man, and an assistant, who travel from place to place, and charge and fire the shots if, after the prescribed examination of the place and its vicinity has been made, it is found safe to do so. Some electric shot-firers test the caps before firing them.

In firing by electricity it is of the utmost importance that the wires or "cables" be left unconnected with the battery until everything else is in readiness and the persons present have removed from the position of the shot to a place of safety.

In charging a hole with cartridges of dynamite or similar compounds, one or more being used according to the charge required, each cartridge should be pushed quietly home with a wooden rammer. If safety-fuze is used, a suitable length is taken, and one end,—cleanly cut,—is inserted in a detonator-cap down to the fulminate, and the cap well closed over the fuze by pressing with circular nippers. In wet places, or if water is used in tamping, the junction of the cap and fuze should be well greased. A small cartridge, called a "primer," is then opened for the reception of the detonator-cap. The cap, with the fuze attached, is inserted in the primer to a depth of about three-fourths of its length, and the paper covering securely tied round the fuze with string. The primer and fuze are then ready for being placed on the charge in the hole, after which some loose stemming is put in and the hole stemmed as may be necessary.

Where gunpowder is now used for blasting, it is usually in cartridges, and in damp places the cartridge-paper should be waterproof. If there be water entering which we are justified in claiming for the patent gelatine-water-cartridge do not only rest upon the invention of Mr. Miles Settle, but upon the employment of the gelatine explosives of Mr. Alfred Nobel in conjunction therewith. The joint use of the two inventions is indeed a most fortunate combination.

"The adoption of the improved water-cartridge and the gelatine compounds by Mr. Settle in his Madeley and Darcy Lever collieries, which are undoubtedly among the most fiery in Great Britain, is a sufficient proof of the safety of this method of blasting. In the collieries referred to, many thousands of water-cartridge shots have been fired with absolute success during the past three years. In the Fair Lady Pit, at Leycett, Madeley, where, on 6th January, 1880, a disastrous explosion occurred, caused by a blown-out gunpowder shot, and resulting in the loss of sixty-two lives, no blasting of any kind was allowed until Mr. Settle introduced his water-cartridge. Since the adoption of this system, blasting has been regularly carried on for upwards of fifteen months with perfect safety, and to the entire satisfaction of H.M. Inspectors and of the workmen employed in the pit.

"It may be interesting to add that, between the 1st May, 1886, and the 31st August, 1887, nearly 300,000 shots have been successfully fired with the patent gelatine-water-cartridge.

"As proved by the experiments conducted under the direction of the Royal Commission on Accidents in Mines, all nitro-glycerine compounds do not possess in an equal degree the safety or advantages of gelatine-dynamite and of gelignite, which latter is another form of the same explosive. Dynamite, although it has been employed with the water-cartridge in several experiments with comparative safety, has in other instances shown sparks and flame, and, moreover, possesses the disadvantage, that unless each charge is protected by a waterproof covering it is liable to dangerous exudation of nitro-glycerine owing to the action of water.

"In view of the danger attending the use of dynamite, gun-cotton, tonite, gunpowder, and other explosives with the water-cartridge, Nobel's Explosives Company, Limited, consider they are justified in prohibiting the employment of the Settle water-cartridge in conjunction with any explosive compounds other than Nobel's gelatine-dynamite and gelignite."

The following description of the requisite appliances has also been issued by Nobel's Explosives Company:—

"ACCESSORIES.—*Water-Cartridge Cases* (Fig. 608).—These are manufactured from specially-prepared waterproof material. The water-cartridge case or bag in general use measures 18 inches long by 2 inches in diameter; but several other sizes may, under certain circumstances, be found to be more suitable, and these can be procured on application.

"*Gelatine-Dynamite or Gelignite Cartridges.*—These cartridges constitute the explosive charge, and are supplied of varying lengths, according to the strength of the shot required. Gelatine-dynamite and gelignite, when used in connection with the water-cartridge, may be considered to be about four hundred per cent. stronger than ordinary blasting powder. About 4 oz. of gelatine-dynamite, or about 4 oz. of gelignite, are therefore equal to 1 lb. of compressed blasting powder; but, as the results obtained in water-cartridge blasting are superior in many respects to those obtained by ordinary powder blasting, the regulation of the charge can most readily be determined by experience. No other explosive than these two gelatinous compounds should be employed with the water-cartridge. One cartridge alone should be used, and in Fig. 607 B is shown the method of fixing the electric detonator fuse therein.

"In exceptional cases, where it is absolutely necessary to employ more than one

"*Nobel's Electric Detonator Fuses* (Fig. 607 A).—These are thoroughly reliable, and differ from the fuses of other manufacturers in several respects. They are usually prepared with wires 54 inches long, but, if desired, they can be furnished with wires of various lengths attached. Fig. 607 B shows the electric detonator fuse inserted into the cartridge, the detonator being pressed in overhead and the

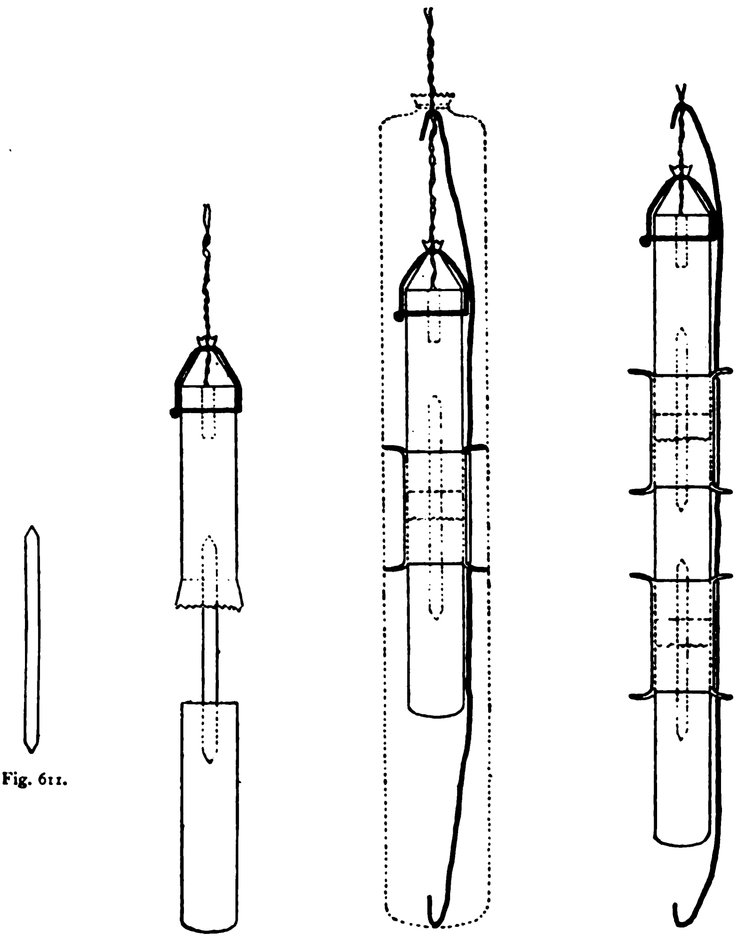


Fig. 611. Fig. 612. Fig. 613. Fig. 614.

SETTLE'S PATENT WATER-CARTRIDGE.

Fig. 611.—WOODEN SKEWER FOR CONNECTING CARTRIDGES TOGETHER.
Figs. 612, 613, 614.—CARTRIDGES CONNECTED TOGETHER TO FORM ONE CHARGE.

cartridge-paper tied over it, the twine being then twisted round the cartridge, to keep the whole in position.

"*Supports.*—These are made of tin, and are so formed as to suit any size of cartridge. Fig. 610 C, D show the tin support, and from Fig. 609 the method of fixing the supports on the cartridge of gelatine-dynamite or gelignite will be readily understood."

to the pit. The Roburite Company issue full instructions, and, if these be strictly followed, injury to the workmen from handling will not arise. A smaller quantity of roburite is required for a given work than of some other explosives, a shot-hole $1\frac{1}{2}$ inches in diameter doing the same work as a $2\frac{1}{2}$-inches water cartridge, and thus effecting a saving in expense. With roburite the flame is quenched by the gases resulting from the explosion, the gases not supporting combustion.

In the beginning of the year 1889 two medical practitioners and the professor of chemistry in Owens College were appointed to investigate the effects of roburite on the health of the workmen in the Park Lane Colliery, Lancashire. As a result of the investigation, the committee came to the conclusion that, with care, the roburite need neither be spilt nor come into contact with the hands of the operator, but in some mines proper precautions are not observed, and cases of undoubted nitro-benzine poisoning have been brought to the notice of the committee, due to improper manipulation of the cartridges. The committee also found that roburite, when properly confined, undergoes complete combustion, leaving no trace of nitro-benzine derivatives unburned, but that there is a chance of incomplete combustion occurring, owing to the explosive not meeting with sufficient resistance.

The following are the general conclusions and recommendations of the committee:—" That although roburite itself is a strong poison, and undoubted cases of poisoning have arisen from the use of it in coal-mines, yet, if stringent care is exercised on the part of the managers, shot-firers, and colliers, the use of roburite will not add to the harmful conditions under which the miner works. As indicating the directions in which additional precautions are in our opinion most necessary, we venture to add the following recommendations to our report:— (1) That the entire manipulation of the cartridges should be entrusted to special shot-firers, who should be instructed in their use. (2) That the effective tamping of the cartridges should be insisted on. Experiments would show by what means the complete combustion of the roburite could be invariably secured. (3) That every care should be taken to ensure the removal of the fumes from the working-faces before the return of the miners, *e.g.*, by continually bringing the brattice cloth up to the working-face. (4) That the products of explosion should be rapidly mixed with a large volume of air. We lay stress on the necessity for the carbonic oxide being diluted with a large quantity of air before it can be breathed with impunity by those who enter the mine."

Bellite is a new explosive, and is said to be as explosive as No. 1 dynamite, although it is not a nitro-glycerine compound. It is the invention of Mr. Carl Lamm, and is composed of four parts by weight of ammonium nitrate, and one part of dinitro-benzol. The powder has a yellowish colour, is almost dry to the touch, and, as it is said to act like the best slow powders, is very useful for coal-mining.

Carbonite is said to resemble gunpowder in its action more than any of the new explosives, except that it is flameless when exploded. It is a nitro-glycerine compound, containing about 25 per cent. of nitro-glycerine absorbed in a mixture of wood meal, and saltpetre, with a little carbonate of sodium, and is made up into cartridges of $\frac{7}{8}$ inch or upwards. It is of a brownish colour, and being of a plastic nature it may be pressed into irregularly-shaped bore-holes so as to completely fill them. The cartridges may be stored for a considerable time without losing their efficacy. When struck with a hammer or stone it will not explode, and will only do so by a detonator. The power of the explosive, which is about twice that of gunpowder, is not reduced when placed in wet shot-holes. Many trials have been made with carbonite, and they prove that it is free from flame, even when exploded in coal-dust, under circumstances where the explosion of gunpowder and some nitro-compounds resulted in great flame and violent interior explosion.

Securite is a granulated powder of light yellow colour, with an odour of bitter almonds, and is the invention of Mr. Schoenewez, a German. It is made out of the bye-products of coke-ovens and gas-works, is said to be efficient, and 35 per cent. cheaper than dynamite. It is a mixture of about 17 parts of meta-dynitro benzol, 80 parts of nitrate of ammonium, and 3 parts of oxalate of ammonium.

The products of combustion act as diluents of the oxygen in the air, but, it is stated, are otherwise possessed of no deleterious properties, so that no discomfort arises to miners using the explosive.

It is said to be about four times as forcible as blasting-powder. In wet holes it must be used in waterproof cartridges. It is claimed for it that it cannot be exploded by ordinary concussions or blows, nor by a burning or a glowing body. It is exploded by detonation. It is said to break down the coal in a similar manner to ordinary powder, and to destroy all flame and sparks within the range of the products of its combustion.

Ammonite is a mixture of about 88 parts of nitrate of ammonium and 12 parts of dinitro-naphthalene.

Ardeer powder is composed of about 34 parts of nitro-glycerine, 11 parts of charred kieselguhr, and 55 parts of sulphate of magnesium and nitrate of potassium.

Of these six explosives, carbonite and ardeer powder are nitro-glycerine compounds, the others being nitrate of ammonium compounds. They were experimented with by a committee of the North of England Institute of Mining Engineers and were found to be less liable than blasting powder to ignite inflammable mixtures of air and fire-damp. They do not, however, ensure absolute safety when used at places containing inflammable gaseous mixtures, for all the high explosives produce flame on detonation.

Westphalite is the latest safety explosive which has been introduced to this country, and from the few experiments which have been made with it it has given satisfactory results.

Reports of scientific men in Germany and France, who were appointed to examine into the question of explosives, state that these substances fired in a dangerous mixture of air and fire-damp ignite the mixture only when their temperature of explosion exceeds about 2,200 degrees Centigrade. The temperature of explosion of ordinary gunpowder is about 2,231° C., of nitro-glycerine about 3,170° C., of dynamite 2,940° C., and of cotton 2,636° C. Therefore it is possible for all these substances to produce an explosion of fire-damp. But if other substances can be added to these explosives so as to reduce their temperature of explosion to 2,000° C., or less, their use will be attended with perfect safety. Experiments appear to prove that an equal weight of either the carbonate or the sulphate of soda to dynamite renders the latter incapable of exploding a fiery atmosphere. Similar results follow from mixing a considerable quantity of finely powdered coal-dust with the dynamite. By experimenting further it is probable that suitable substances may be found, which, when added to any compound, will render its use safe.

The use of *Lime cartridges* as a substitute for blasting has been tried within the last few years. The system is patented by Messrs. Sebastian Smith & Moore.

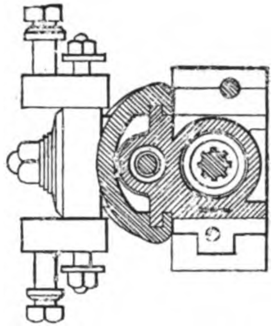


Fig. 628.

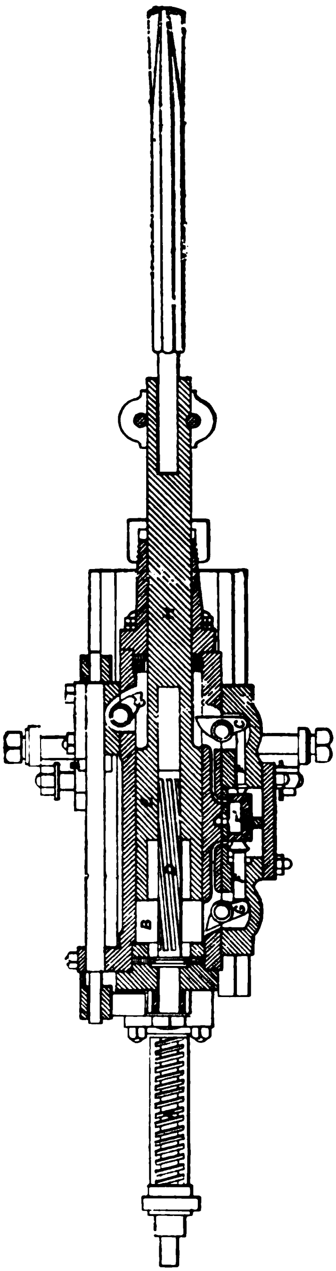


Fig. 626.

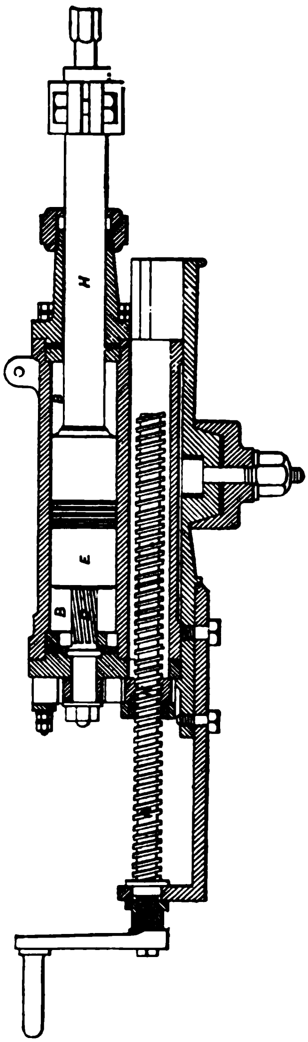


Fig. 627.

Figs. 626–628.—The Ingersoll Machine-power Rock Drill as formerly made.

One of the best is the Ingersoll Machine-power Rock Drill, shown in the drawings, Figs. 626–628. It consists of a cylinder B, in which works the piston E, H being the piston rod. The valve-gear is actuated by the piston through the medium of tappets. It consists of a slide valve E′, two valve spindles F, F, and the two tappet levers C, C.

The piston, as it moves backwards and forwards, alternately encounters and depresses the limbs of the tappets which project into the cylinder, as shown on the drawings; and as these limbs move simultaneously, the motion thus communicated is transmitted by the valve spindles to the slide valve. The tappets are so adjusted relatively to the stroke of the piston as to admit of a cushion (of steam or air) being formed at either end of the cylinder; this cushion prevents injury to the cylinder covers and preserves the drill from excessive vibration.

A spirally grooved bar D, recessed into the back end of the piston, gives a rotary motion to the drill. The piston has a cap screwed into it fitted with studs to run in the grooves of the bar.

Fig. 629.—The Ingersoll Rock-Drill in Work.

A ratchet-wheel is fixed to the end of the bar, and into its teeth a pawl is held by a spring. By means of the pawl, the piston is compelled to turn during the back stroke, whilst allowing the spiral bar to rotate during the forward stroke. Besides the automatic rotary motion of the drill now described the feed motion in the Ingersoll rock drill is produced automatically, so that as the borehole advances the cylinder equally moves forward and there is no diminution in the force with which the drill strikes the rock. As the drill penetrates the rock, the piston approaches the forward end of the cylinder, and strikes against a tappet lever C′ which partly rotates the rod on which it is carried. By means of pawls and ratchet teeth, this rod turns a nut upon the back end of the cylinder. The feed screw A passes through this nut, which in the act of rotating upon the fixed feed screw causes the cylinder to advance.

The drills used are of different sizes, according to the work to be done. A $2\frac{1}{2}$-inch cylinder drill is capable of boring a hole from 1 inch to $1\frac{3}{4}$ inches in diameter 8 feet deep. The motive power may be either steam or compressed air, the usual working pressure being 45 lbs. per square inch.

For shaft-sinking the drill may be carried on a tripod, or on a special shaft-sinking frame. For driving headings it may be supported on a stretcher-bar, as shown in Fig. 629. This is a wrought-iron hollow tube, provided at one end with a claw, and at the other a strong adjusting screw lock and nut for fixing it in position. For more important tunnel work the apparatus is mounted on a tunnel-car, specially designed for the purpose. This carries four drills.

The rate of progress by rock-boring machinery is from 3 to 12 times greater than by hand-labour, according to the hardness of the rock.

The only objection we have heard against the above described form of the Ingersoll rock drill is that when heavily driven, the tappets are apt to break, followed almost certainly by injury to the cylinder through the broken pieces jambing the piston against the walls of the cylinder.

Since the foregoing description and remarks were written, a new type of Ingersoll machine-power drill has been devised. In it the use of tappets is dispensed with and also the stems which acted in conjunction with them to move the valve across its face. The valve is now formed by a segment of a circle, and is brought in direct contact with the piston, where it acts in the place of a pair of tappets by receiving a gentle push from the inclines on the piston, instead of the blow

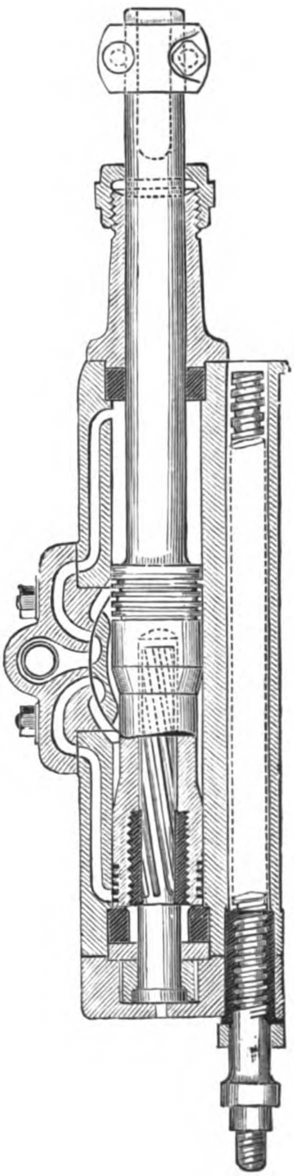


Fig. 630.

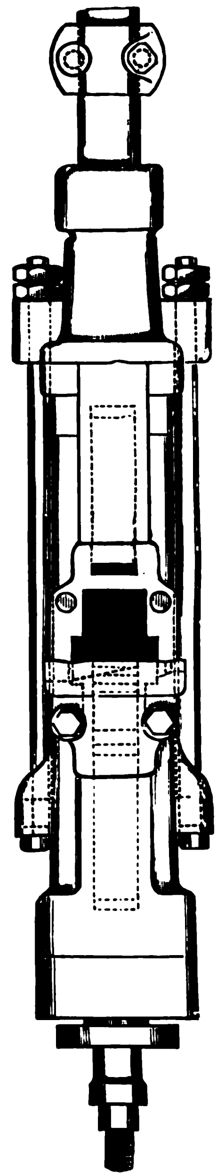


Fig. 631.

THE INGERSOLL MACHINE-POWER ROCK DRILL AS AT PRESENT MADE (1890).

formerly given by the piston to the tappets, and thus admits the motive-fluid fore and aft alternately into the cylinder, while at the same time the fluid keeps it firmly up against the face.

The following particulars are supplied by Messrs. Legros, Mayne, Leaver & Co.:—

"Fig. 630 shows a section, and Fig. 631 a plan of this new patent drill, which the reverse of the old style, in which the tappet causes the pawl to be thrust sharply into the ratchet, and the spring disengages it, whereas, in the present form, the effect is ensured by a more gentle motion, and with fewer parts, the knuckle-joints, separate tappet, and feed-regulator being dispensed with. It is now intended to extend the side bolts so as to take the back as well as the front cylinder-cover, and in the automatic-feed drills these have to be set slightly on one to allow the spindle to occupy its proper position.

"In the hand-drill shown in Fig. 625 the automatic-feed is obtained by the cone at top of the working rod acting on the feed ratchet by means of a lever, spring, and pawl, the cone in effect taking the place of the tappet in the power-drill."

COAL-CUTTING MACHINERY.

The labour of coal-getting is arduous even under the most favourable circumstances, and in thin seams it is especially severe and trying, owing to the constrained and unnatural attitude of the collier during the time he is at work, and more particularly at the work of "holing."

It is considered allowable, with good and skilful hewing, to take a height not exceeding 9 inches for the holing when carried to a depth of 3 feet.

Where the holing is altogether in the coal, and the seam thin and hard, hand-hewing is placed at a great disadvantage, because the coal obtained as a result of the holing is much less than that obtained from the same depth of holing on thick seams, the labour of holing in which is much less. Coal-cutting machines have been designed, and are in use to some extent, for undercutting the coal. Where circumstances admit of their use they save the collier the heaviest part of his toil. There is, however, great difficulty in the general adoption of coal-cutting machinery, on account of the undulating character of the seams, and the resulting irregularities of roof and thill over very limited areas. These irregularities form great obstacles to the smooth working of coal-cutting machinery, besides which, some districts are so much intersected by faults breaking the continuity of the coal-seams that machinery for getting the coal is quite out of the question. Other coalfields are more favourably situated, and on economical grounds it may be judicious to adopt coal-cutting machines.

The following description of the Gillott & Copley Rotary Coal-Cutting Machine is furnished by Messrs. John Gillott & Son, of Barnsley:—

"The machine described below has now been over 13 years in practical operation, under a variety of circumstances. It is designed for holing or undercutting, and is more especially adapted for collieries worked on the 'Longwall,' or some similar system, where a considerable length of face can be operated upon.

"It is driven by compressed air, and works at the low pressure of from 20 to 30 lbs. per square inch. This is of great importance, as when this pressure is exceeded it is attended with considerable difficulty and a largely increased cost.

"It can be made to cut level with the floor, in a parting between two coals three feet or more above the trams, or at any other height; and is applicable for any seam of coal where a height of not less than 20 inches can be afforded for it to travel along the coal face.

"It will cut in fire clay seating, *hard or soft coal*, or take out a pricking between two coals. It is self-propelling, is made to suit the gauge of any colliery tramway, and travels on the rails as ordinarily laid by the colliers. No fixing is required to keep the machine up to its work.

"The machine is made principally of steel, thus combining the greatest strength with the least weight in the smallest space. It is made in three sizes; in the largest machines the frame, which is of crucible steel, is about 5 feet 6 inches long by 2 feet 4 inches wide, and on this are fixed two cylinders 9 inches in diameter, with a 9-inch stroke, working on to a forged steel crank-shaft, which by a simple

STANLEY'S COAL-HEADING MACHINE.

In carrying on the many operations incidental to coal-mining, circumstances may arise which necessitate long headings or narrow places in the coal being rapidly driven. The use of a machine, such as that designed by Messrs. Stanley Brothers, enables a much greater rate of progress to be attained than is possible by hand-labour. The machine runs on wheels in the heading. Attached to the inner end of a central shaft are a radial and two horizontal arms. Cutters are fixed on the horizontal arms, which, as the central shaft is rotated, cut an annular groove in the coal and form a core within it, which is removed from time to time as the work proceeds. There are two sets of gearing for working the machine. That in front causes the shaft and arms to revolve. This is effected by admitting compressed air to the two cylinders carried on the framework which give motion to a shaft connected with the central shaft by means of cog-wheels, through which the motion is transmitted. The back set of gearing provides for the forward motion of the cutters in operation. The central shaft is threaded so that the back gearing, which consists of a cog-wheel with threaded gun-metal bush fitted into its boss, works on it. The back cog-wheel is secured to the frame, and may be driven by a sliding cog-wheel working on the crank-shaft.

On setting the back gear into motion, the frame advances on the central shaft; when the machine has been moved forward, the back gearing is thrown out, the front gearing thrown in, and the central shaft and arms advance in the frame, at the same time that they are revolving. The machine is provided with two telescopic screw-pins by means of which the frame is held securely in position while the cutters are at work. The usual size of machine weighs two tons, and the cutters form a heading five feet in diameter, at the rate of about three feet in an hour. The arms can only advance between three and four feet; and when out their full length the engines are stopped; the back gear thrown in, and the frame advanced and secured in position again for a further cutting. Different types of the machine are made to suit the varying qualities and classes of coal seams and their thickness.

CAGING APPLIANCES AND DROP STAPLES.

A serious loss of time occurs at collieries in changing tubs where the shafts are deep, if cages having two or more decks are used, and the ropes are lapped on drums in the ordinary way, unless some caging appliance be used. The inconvenience and loss are enhanced if there are intermediate loading stages in the shaft between the surface and the bottom.

In order to avoid having more than one loading stage in the shaft, drop staples are often used for lowering the coal from an upper to a lower seam, or the same object is attained by means of a self-acting inclined-plane driven across the measures between seams of coal.

Drop staples are fitted with guides or conductors precisely the same as the winding shaft. Two single-decked cages are used in the staple by means of which the full tubs are lowered, while the empty tubs are raised. The weight of the full tub or tubs is sufficient to effect the change. A single rope is generally used, which is attached to the upper cage in the usual way, passed over a clip-pulley, and the lower end attached to the cage at the bottom. The clip-pulley is held securely in its place by framework a few feet above the seam. Rails or flat-sheets are laid at the loading and unloading stages to facilitate the entry or discharge of tubs at the cages. The clip-pulley is provided with a brake-ring, by means of a brake acting on which, through levers, the velocity of the cages is controlled and regulated by the upper-stage attendant, and an up and down signal are provided. Both at top and bottom shaft gates are placed for the pro-

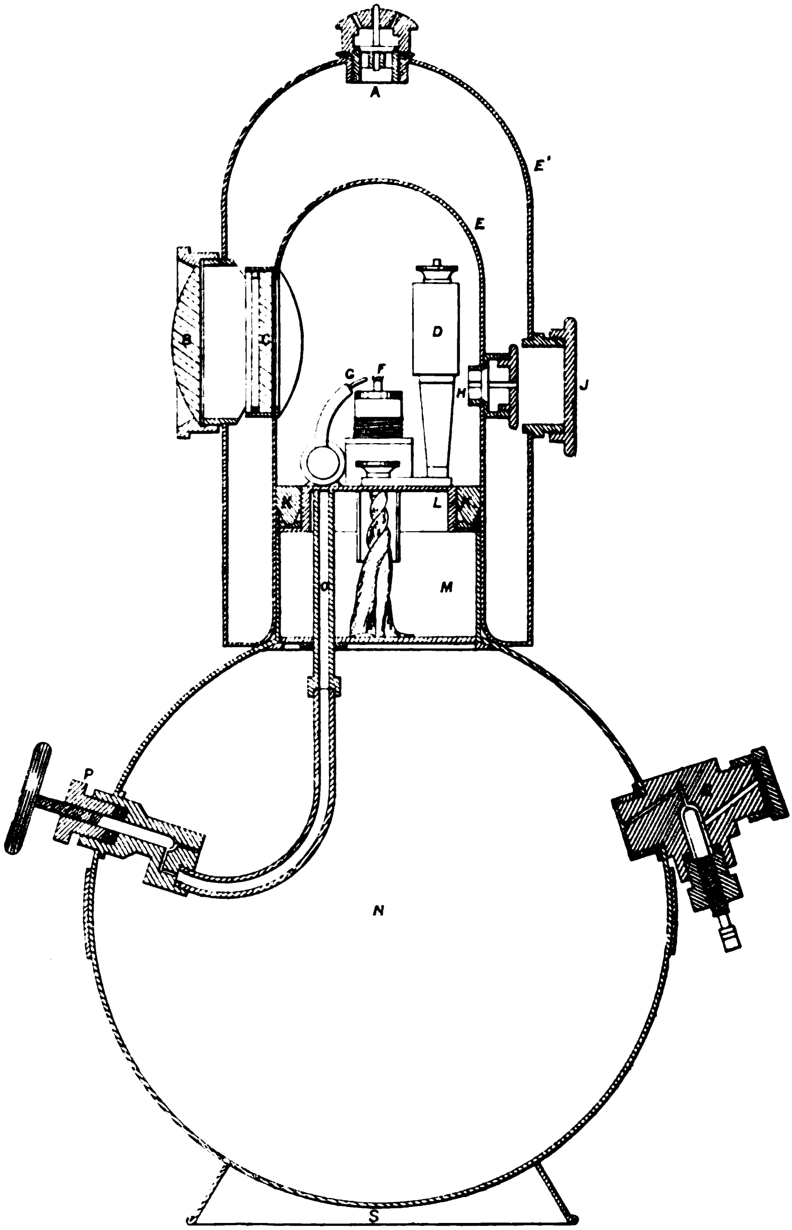


Fig. 653.—The Fleuss Lamp.

The supply of oxygen being under the control of the wearer may be urged as an objection to the apparatus. If he become nervous and supply himself too freely he will exhaust the store before it is safe to do so; no other inconvenience would arise, as an overdose of oxygen is not injurious. This objection would be is conveyed by boxes to the water pipe or pipes A in the dam, and the pieces having all been built up, the wedging is proceeded with from the inside. Fir wedges, 12 inches long by 3 inches broad and an inch thick at the head, are first

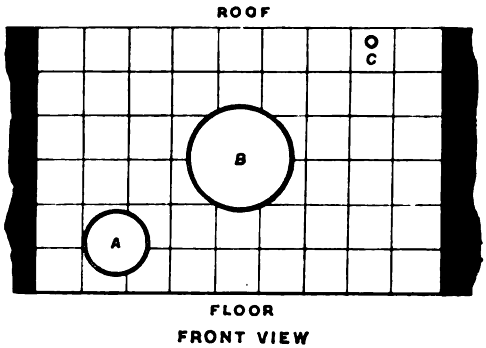


Fig. 655.

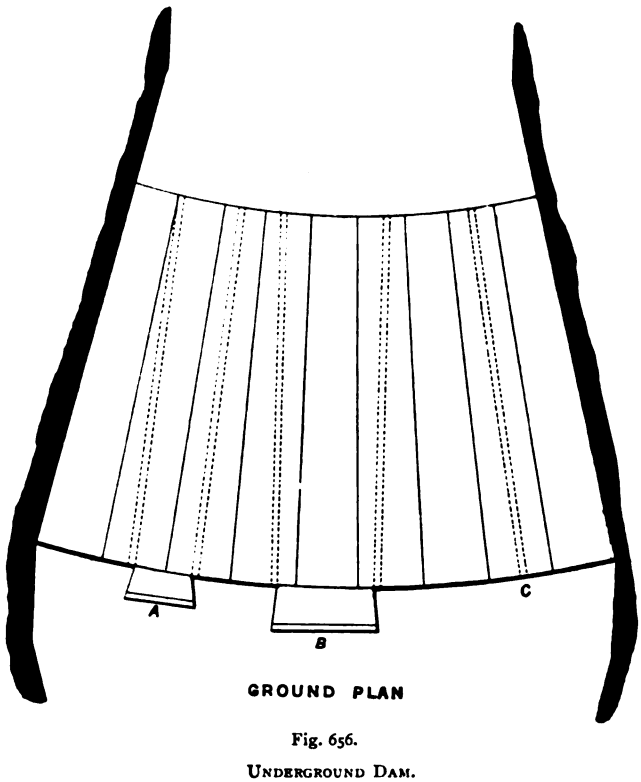


Fig. 656.

UNDERGROUND DAM.

driven in, and after these have been driven at all the joints and round the pipes, smaller wedges of oak may be driven, an iron chisel being used to prepare places for their insertion. When no more wooden wedges can be got in a few steel wedges may be made to enter, more especially between the wood and the stone or coal. After the wedging is completed, the workmen drive a plug of wood into 10 equal divisions and then to set off half of one space from each extremity and arrange the remainder equi-distant. Lines are drawn at right angles with the atmospheric line from each intersection or divisional point, then with the scale of the diagram each ordinate is measured and figured above and below the atmospheric line. The total sum of each set of ordinates is then obtained by simply adding the several lengths together, and the mean is got by dividing the total by the number of ordinates—in this case 10. In Fig. 668 the mean pressure of the steam is 15·575 lbs. on the square inch, and the vacuum obtained is equal to an atmospheric pressure of 9·69 lbs. on the square inch. The effective pressure on the piston then is $15{\cdot}575 + 9{\cdot}69 = 25{\cdot}265$ lbs. per square inch, and the horse-power of the engine may be found from this, knowing also the length of stroke and the number of strokes the engine is going per minute. It will be observed that the remarks on Fig. 668, R 110, S $27\frac{1}{2}$, V 26, mean that the engine is going 110 revolutions per minute with a steam pressure in the boiler of $27\frac{1}{2}$ lbs. and the vacuum gauge marks 26 inches or 13 lbs. pressure. It will be noticed that the highest record of full supply of steam in the cylinder is 24·2 lbs. showing a loss due to friction and radiation in passing from the boiler to the cylinder, also that the difference in the vacuum pressure is owing to the temperature in the cylinder being higher than in the exhaust steam pipe at the condenser end.

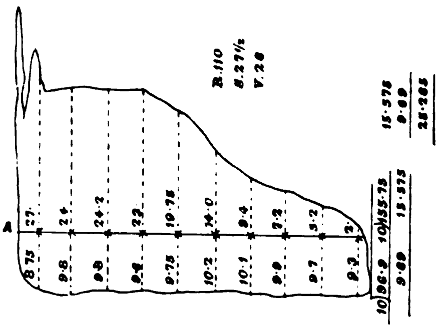


Fig. 668.—INDICATOR DIAGRAM.

To be strictly accurate in working horse-power of engines a diagram should be taken on both sides of the piston, even where the action of the steam on the one side is repeated on the other, and a mean of the two diagrams adopted as representing the effective steam pressure, but for the sake of clearness the method is here shown with a single diagram.

In diagrams taken from non-condensing engines the line traced will not, of course, at any point descend below the atmospheric line, but will in fact be above it owing to the back pressure of steam on the other side of the piston. Therefore, in measuring the ordinates on a diagram obtained from a non-condensing engine, measure with the scale from the atmospheric line to the upper line traced on the diagram and also from the atmospheric line to the lower line of the diagram. Having obtained the average of each set of ordinates, subtract the one from the other, that is deduct the average back pressure from the full pressure to obtain the average effective pressure. The average back pressure is seldom less than 3 lbs. even in good engines.

Goodeve, in his *Text Book of the Steam Engine,* thus writes of the indicator diagram of a single-acting engine :—

"In the single-acting engine two diagrams must be taken, one from the top and the other from the bottom of the cylinder. These diagrams are quite unlike in form, for the action during the down-stroke is not repeated during the up-stroke as in a double-acting engine, and our first task will be to comprehend the reasons of the particular conformation observed. For this purpose reference is made to a diagram taken from a Cornish pumping engine, having a cylinder 70 inches in diameter and making 4 strokes per minute, under a mean pressure of 15·1 lbs. per square inch. The figure is re-act disadvantageously upon the parallel motion as does that in the Richards indicator.

In Fig. 671 the indicator piston is connected to the main horizontal rod of the parallel motion by a connecting rod, passing inside the piston rod, which is made hollow so as to allow of this. The attachment to the piston is by means of a small double ball joint. This gives a joint which is quite free without having any play and allows of a means of adjustment by taking up any play resulting from wear on the joint. The parallel motion is carefully designed to ensure that the pencil point describes a straight line and that the motion of the pencil point is precisely proportional to the displacement of the indicator piston throughout the stroke. To suit the higher speeds for which the indicator is intended, the passage of the cock and connection is made $\frac{1}{2}$ inch, against $\frac{3}{8}$ inch in that of the Richards indicator. The piston motion is multiplied in the proportion of 1 to 4, as is also that of the Richards as made by Messrs. Schäffer and Budenberg. The stroke of the piston is only $\frac{3}{4}$ inch as against $\frac{7}{8}$ inch, the maximum height of the diagram being consequently 3 inches, against $3\frac{3}{8}$ inches, in the Richards. In all other respects the Thompson and Richards indicators are similar. The former is also made in a small size, which is intended more especially for taking diagrams at high speeds. In it the parallel motion is made very light, so that the oscillations of the pencil at the highest speeds are considerably reduced; a stronger spring than in the large instrument being invariably used. This instrument complete with fittings weighs only 5 lbs., as against 11 lbs. in the case of the large Thompson, and a still greater weight in the ordinary Richards indicator. Its greater portability gives it an additional advantage over the other instruments. The large Thompson indicator has been successfully employed at a speed of 400 revolutions per minute, whilst the small

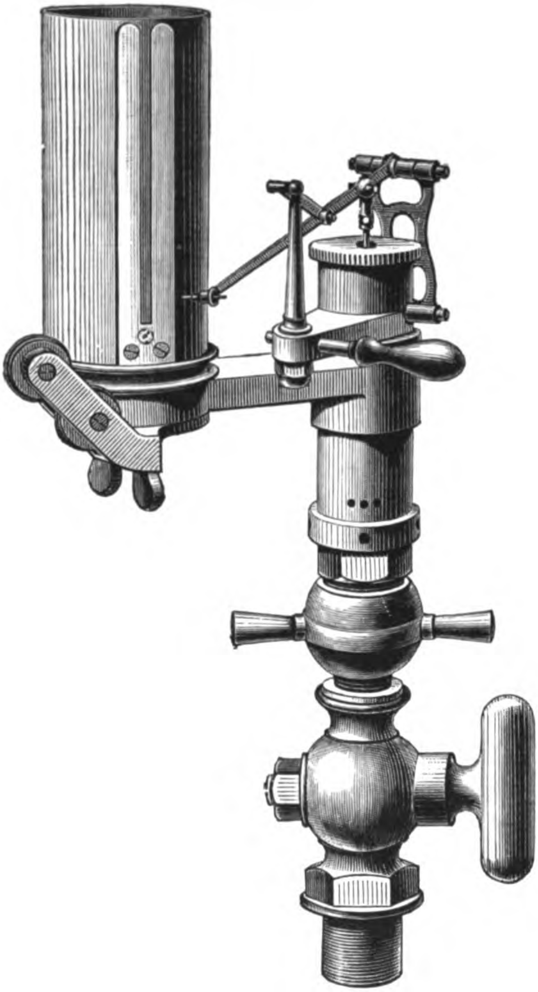


Fig. 671.—The Thompson Indicator.

Thompson may be used up to 600 revolutions per minute if a sufficiently strong spring is inserted.

If desired the apparatus can be supplied with a rising motion of the drum operated by hand for taking any number of diagrams successively on the same paper.

Messrs. Schäffer and Budenberg also make a *Double Indicator* which is a novelty in construction. In the ordinary indicator, the upper or steam-pressure curve drawn on the diagram does not in reality pertain to the back-pressure curve drawn beneath it, but belongs rather to that back-pressure curve which would result if the other side of the piston were indicated simultaneously; and the calculation from an ordinary diagram of the work done by the engine during a stroke is based upon the assumption that the action is identical during two successive strokes on both sides of the piston.

The object of the double indicator is to be enabled to place both sides of the piston in communication with the indicator at the same time, if desired, and in such a way as to record on the diagram, the actual resultant effective pressure on the piston throughout the stroke. This instrument consists practically of a combination of two indicators acting upon the pencil simultaneously, and causing the latter to record directly the resultant-pressure upon the engine piston. The double indicator has two pistons fixed upon the same spindle, and when both indicator cocks are open, the pistons are subject respectively to the pressures on both sides of the engine piston, the pressure being, of course, in opposite directions in the case of non-condensing engines, and in the same direction in condensing engines. There is only one spring, and this is employed in compression upwards or downwards, according to the direction of the resulting pressure on the engine piston, the arrangement of the spring being such that it can only be used in compression. The diagram traced by this indicator is therefore situated partly above and partly beneath the atmospheric line, and the distance of the diagram from the atmospheric line at every point represents the actual resultant-pressure on the piston. The areas of the diagram above and below the atmospheric line show respectively the work of the steam on each side of the piston, and the area of the whole diagram represents the actual work performed during one revolution. If only one indicator cock be opened at a time, separate diagrams from each cylinder end may be obtained, and by taking these on the same paper as the combined diagram, a complete record is obtained which shows all the details of the separate diagrams, and facilitates the calculation of the whole diagram.

A peculiarity of the double indicator is that its indications are independent of the barometric pressure, the piston rod being equilibrated for this pressure, whilst in all ordinary indicators the atmospheric pressure acts upon the upper side of the piston.

The instrument has not become very popular owing to the diagrams obtained from it differing to such extent from those of the ordinary indicator as to puzzle the occasional user.

PRESSURE AND VACUUM GAUGES.

The first form of gauge used to determine steam-pressure was the mercury column, and it is still used as the standard of comparison when graduating existing gauges. The indications of the mercury column are most accurate and reliable in careful hands, but it is inconvenient for general use, and impossible in cases where there is oscillation, as on locomotives. Fig. 672—the *Bourdon*—

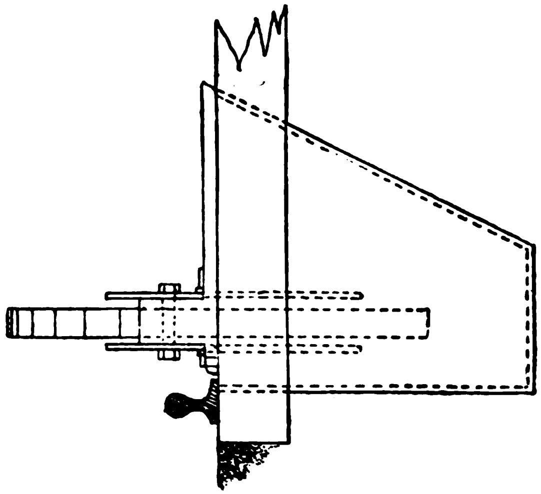


Fig. 678.—End View.

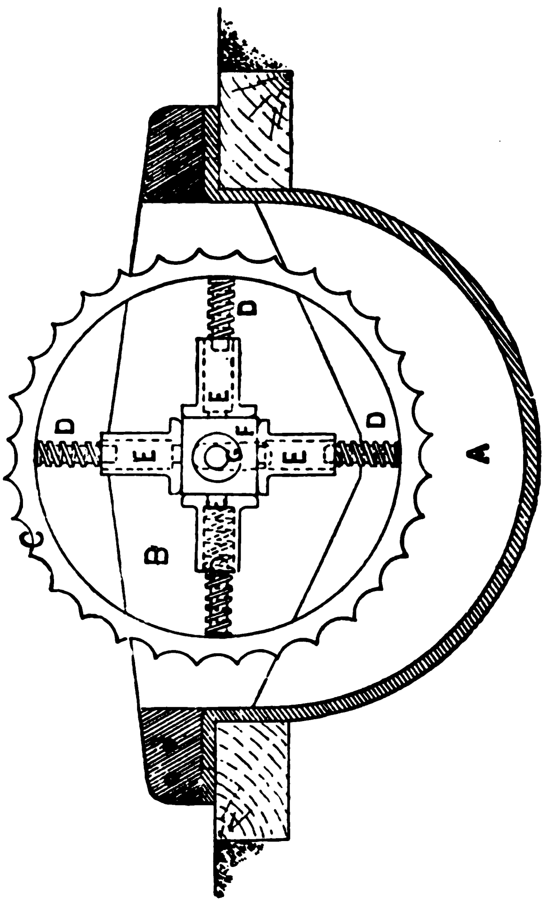


Fig. 677.—Section.

Dunford and Emens Patent Automatic Greasing Apparatus.

running tub comes into contact with one of the concave recesses in the rim, the journal receives a charge of lubricant, and, besides revolving the rim, pushes it in the direction in which it is running, the springs yielding, and the feet of the enclosing cases sliding on the flat faces of the boss, the rim being pushed temporarily into a position eccentric to the centre of its axle till the tub axle has passed over, when it returns to its normal position, to be acted upon in the same manner by the next following axle, which receives a fresh charge of lubricant. Thus the rim will suit itself to tubs having wheels of different diameters to the extent of several inches.

"The greasing wheel on one side of the line is entirely unconnected with the wheel on the other side; each acts freely to administer a charge of grease to the journal on its own side of the line of tubs.

"Where the tub wheels revolve on the axles this greaser is useless, but is very efficient for tubs to which the wheels are fixed to the axle and the whole revolve."

The Hardy Patent Pick Company, Limited, Sheffield, supply a *self-lubricating pedestal* for colliery tubs which is designed also to protect the axles of the tubs from dust. This is of much importance, especially in dusty mines, where ordinary bearings get full of coal dust which renders useless the newly applied lubricant. The construction of the bearing is very simple. The top portion bearing the load is similar to the old form of pedestal, but underneath a steel dish is fitted, which keeps the axle free from dirt and at the same time prevents it from leaving its bearing. The steel dish is stamped out of one sheet of metal, and is shaped to hold a quantity of felt or wool, which bears against the axle on the under side, and being soaked with the lubricant, keeps the journal well oiled. When first charged about 5 oz. of oil is required in the operation, but when necessary to replenish, a charge of 2 oz. is sufficient. It is said that the tubs will work two or three months without further attention, after once oiling, and the journals remain clean and well-oiled throughout that time. A tub with outside bearings may have the lubricant applied to the journals at any time as the tub rests on its wheels, but with inside bearings it is necessary to turn the tub by means of a lifter and then the oil may be poured through a hole which leads into a large hollow space in the top of the bearing plate. On returning the tub to its usual position on its wheels the oil runs down into the wool or felt.

Although the number of mines increased and the workings became more extensive as time went on, these two explosions were the most serious recorded until the present century. In March, 1767, 39 lives were lost at Fatfield and in October, 1799, a similar number were lost at Lumley.

An explosion occurred at Hebburn Colliery, in October, 1805, which destroyed 35 lives, and another in November, 1805, at Exclose, resulted in 38 deaths.

A much more disastrous explosion than any previously recorded, occurred at the Felling Pit, in Durham, whereby 92 lives were sacrificed in May, 1812. The magnitude of this disaster attracted considerable attention, which led to the formation of an association at Sunderland, the object of which was to guard the miners' interests. Nevertheless explosions continued with unabated force and frequency.

In 1829 the House of Lords began to investigate the cause of accidents in mines, and published their first report in 1833.

On June 18, 1835, a very disastrous explosion occurred at Wallsend Colliery, which caused the death of 102 persons.

Various efforts to secure information about these disasters were made by Select Committees and Royal Commissions, with a view to taking precautions against such catastrophes. Faraday and Lyell conducted an exhaustive inquiry into the Haswell Colliery disaster which occurred in September, 1844, and resulted in 95 deaths. These very careful investigators presented their report to the Home Secretary in 1845. In it they stated their opinion that the destructive force of the explosion was not due entirely to the ignition of a mixture of firedamp and air, but that it was greatly increased by the fine dry particles of coal-dust which had collected on the collars, props, sides and floor of the mine.

Following closely on this came the appointment of Government Inspectors towards the end of 1850, whose careful supervision has done much to improve collieries generally, but has not put an end to explosions.

The most notable explosions that have occurred since then, each involving the loss of 100 lives or more, are shown in the following table :—

Name of Colliery.	Date of the Explosion.	No. of Lives Lost.
Cymmer	March, 1856	114
Lund Hill	19th February, 1857	189
Risca	1st December, 1860	130
*The Oaks	12th December, 1866	334
Ferndale	November, 1867	178
Swaithe Main	6th December, 1875	143
Blantyre	22nd October, 1877	207
Haydock	7th June, 1878	189
Abercarne	11th September, 1878	268
Risca	15th July, 1880	120
Seaham	8th September, 1880	164
Penygraig	10th December, 1880	101
Pendlebury, near Manchester	18th June, 1885	178
Llanerch	6th February, 1890	176
Park Slip, Aberkenfig, near Bridgend	26th August, 1892	112
†Combs Pit, Thornhill Collieries, near Dewsbury	4th July, 1893	139
Albion Colliery, Cilfynydd, near Pontypridd	23rd June, 1894	290

* 334 were killed by the first explosion and 27 afterwards by closely following explosions, making a total of 361.

† A slight explosion of firedamp fired some shaft fittings: the smoke and afterdamp carried into the workings caused the 139 deaths.

the first explosion-wave being ignited a vacuum is formed in its rear towards which air rushes from all directions open to it; consequently fresh fire-damp is liberated from the face, the sides, roofs, and floors of the roadways and from the goaves remote from the seat of the explosion. The destruction of the vacuum may be followed immediately by a very violent inward pressure which creates a current sufficiently strong to force the flame of a safety-lamp through the gauze on to what has now become an explosive mixture and so ignite it. If instead of safety-lamps, naked lights are in use, no inward pressure would be necessary to produce one or more explosions at a distance. Again, there is the possibility of a blower being suddenly tapped by a fall far from any lights. The blower at first fouls the atmosphere in its immediate vicinity, and soon brings it up to the explosive point. As the issue continues the explosive mixture may gradually tail back through any channels open to it until it reaches lights, or shots at the moment of firing, in two or three places at the same time and so set up separate explosions. Or if the fire-damp from the blower does not penetrate more than one road in being given off, it may be ignited at its outward end by a defective safety-lamp or open light, and if the resulting explosion-wave then travel against the current impregnated with the gas, the expansion following the initial explosion may drive back the body of explosive gas upon an open light, defective safety-lamp, or flame from a shot, and so bring about a second explosion during the continuance of the first.

A blower of this magnitude would not be likely to exhaust itself before an examination of the position had been made, and would thus be detected, unless access to that part of the mine were prevented by standing fire, or heavy falls of roof. If on examination within a day or two, there is no abnormal issue of gas, this would be reasonable evidence against the theory that a blower had been tapped.

Under certain conditions of roof, and where the roof yields fire-damp freely, it may be that spaces exist, out of sight yet communicating with the air of the mine through cracks. These spaces being filled with an explosive mixture might form a train of fire, and thus ignite some accumulation of gas at a considerable distance. The condition most favourable for this, would be where the roof for a few feet immediately above the coal subsided gradually upon the gob as the face advanced, yet the next stratum above consisting of hard rock several feet in thickness only subsided at intervals of some weeks for a certain distance back from the face. Thus, a cavity for gas would be formed, which might extend for hundreds of yards following the line of faces. That this is so, is frequently shown by abnormal quantities of gas coming off when a sudden weighting of a strong upper roof occurs, while at other times no gas can be detected. For weeks the gas filling this hidden space may remain comparatively pure, but sooner or later some communications with the atmosphere of the mine will be effected, and will thus lead to the formation of a highly explosive mixture, mechanically, as well as by changes in pressure and by diffusion. A train may be thus prepared and lie ready in waiting throughout a network of crevices and fractures communicating with a distant place containing a body of gas which may have escaped detection. It is thus conceivable that if an explosion fires this train, the flame may be carried along until it reaches the distant gas when it would set up a second full explosion, which would then run its course independently but almost simultaneously with the first.

Another theory to account for these complex explosions which has found favour with some scientific authorities is that the pressure resulting from the first explosion is sufficient to cause an inflammable mixture in another portion of the mine to part with some latent heat which may of itself ignite an explosive mixture in that place. That it is possible by compression to ignite a certain percentage of fire-damp present in air may be proved on a small scale by means of Dr. Angus

Given, then, a coal-dust of sufficient fineness, quantity and dryness and an explosion-wave initiated by a strong ignition of fire-damp, and an indefinite extension of the explosion is inevitable.

Many other kinds of dust are known to be inflammable. Flour-dust explosions have occurred at mills in different parts of the country, which have been followed by patient and careful inquiry. It was formerly the custom to crush corn by a pair of mill-stones revolving rapidly in opposite directions. On the corn-feed being stopped, without the cessation of the machinery, sparks have been produced from the friction of the stones which have ignited the fine flour-dust in the immediate neighbourhood, and passing through a channel of agitated flour-dust have produced a violent explosion in a large room at a distance.

Similarly, explosions of malt-dust have occurred in kilns used for drying and grinding malt. Ground cork-dust, too, made in linoleum works, has ignited and caused explosions. Probably many other dusts of combustible substances, if sufficiently fine and diffused in proper proportions in the atmosphere, will explode or ignite on the application of a sufficient volume of flame.

On the one hand it may be affirmed that all the largest explosions have occurred in mines which are more or less fiery; and on the other hand, that no large explosion has ever occurred in a very wet mine.

Small explosions occur in all kinds of mines, wet and dry, and two have occurred in mines in which fire-damp was not known to exist, or had not been previously found, viz., one at the Camerton Colliery and another at Timsbury.

In those cases where under apparently normal conditions a serious explosion has occurred, traversing extensive workings all in full operation,—thus precluding the idea of fire-damp having previously been present in large quantities, or long stretches of explosive currents, in accessible parts of the mine, no other theory appears tenable than that of a sudden and immense outburst of fire-damp, or of an explosion intensified and magnified a thousandfold by coal-dust. If a sudden outburst, it would seem to require an appreciable time to have elapsed between the discharge of gas and its ignition, so as to form a large explosive body pervading the whole of an extensive mine. On the other hand, if coal-dust has been chiefly instrumental in an extensive explosion, it is not necessary to assume any large emission of gas. The latter alternative, therefore, seems by far the more probable.

Explosions have in many cases been confined to dusty roads, and have stopped where the dust has ended, although the course pursued was not that in which the most fire-damp might have been expected. In a case where a double intake, made up of duplicate airways, extends from the downcast shaft, and then merges into a single one, the explosion-wave has been observed, after reaching the merging point in its outward course, to select the dusty road without penetrating far along the other, which was free from dust. Where the choice of routes exists between a dusty and a clean road, the explosion-wave generally apparently follows the dusty one, even if the clean one presents a straight and more direct passage to the shaft. Of course the choice would be the same if fire-damp were in one of two roads, but of all places in the mine the least likely for gas to be found are the main intakes. In the case of two roads, free from dust, one containing a current up to the explosive point, and the other comparatively pure air, an explosion would certainly follow the former, while it would only expand into the latter and not travel far in it.

Then again, as explosions almost invariably traverse the intakes outwardly, the natural point at which to expect an explosion to exhaust itself is at the downcast shaft, or before reaching it. Where the upcast is near the downcast, this shaft also is frequently affected, not so much by the passage of the blast through the returns, as by its passage through openings near the downcast, the separation doors in which do not form an invulnerable barrier. The noise of an explosion heard on the surface is usually followed closely by ejections of smoke and dust likely to be found, there are instances where, probably, an equal amount is deposited in the return air-ways. These are cases where the seams of coal yield an exceptional amount of fine dust in the actual process of working. From the faces it is carried by the air-current into the return air-ways, and settles there. The more active the ventilation, the more coal-dust will find its way into the returns.

The comparatively high temperature which prevails in most underground workings, doubtless, assists the fine division of dust-particles. Dust remaining a long time subjected to this heat becomes drier, and it may be that it absorbs oxygen or undergoes some chemical change which makes it more dangerous. On the other hand, long exposure to the atmosphere probably causes it to part with some of its volatile gases. It is known that the capacity for absorbing oxygen is greater in freshly worked coal than in that which has lain some time, but it appears that the newly-made coal-dust is not so inflammable as old dust. The chemical changes which take place would therefore appear to increase the dangerous character of coal-dust, but the precise nature of these changes is still unknown and requires investigation.

Since the date of Mr. Galloway's and Sir Frederick Abel's experiments already referred to, many other investigations have been made with a view to satisfy the doubts prevailing, and to ascertain the precise circumstances, if any, under which coal-dust and air may be ignited. It is obvious that such experiments must be very costly. They cannot be made in a mine for fear of serious damage. They have, therefore, for the most part, been carried on in specially constructed boxes on the surface. Results, both of a negative and positive kind, have been obtained. In nearly all the experiments, the firing of a cannon has been taken to represent the concussion and flame from a shot, and has been discharged into a gallery containing (*a*) coal-dust and air alone; and (*b*) coal-dust, air and fire-damp. Although in the case of (*a*) it was said that ignition resulted in some of these experiments, the conditions were such as would not be found in a mine. The Camerton explosion and Mr. Hall's experiments conclusively prove that coal-dust and air may be ignited.

Slight ignitions of coal-dust with air only are stated to have occurred about the colliery screens, from which clouds of coal-dust have been raised and carried by the air to some gas-light or other large flame. Some of these ignitions have merely been a momentary "flare-up" of the flame of a gas-jet, perhaps owing to a paucity of coal-dust particles. A more serious ignition of coal-dust only, resulting in loss of life, is reported to have occurred in a hopper on the surface of Brancepeth. In this case a large quantity of dust was thrown from a height, raising a dense cloud, which ignited at an oil-lamp when the hopper was being cleaned out—a case which could hardly occur in the limited height of underground roadways. These ignitions showed no signs of great explosive force, but this may have been due to their occurring where there was ample space for free expansion from the heat.

The Altofts colliery explosion was officially reported to be due entirely to the firing of coal-dust. It has indeed been suggested that a very slight percentage of fire-damp may have been present to aid the initiation of that explosion. It is urged, in support of this theory, that probably no serious colliery explosion has ever occurred, except in a fiery mine. Wet mines are sometimes fiery, and explosions have occurred in them. These would be due to fire-damp alone. Some dusty mines are free from fire-damp, and have certainly had no coal-dust explosion during the last 50 years and probably never at all. Seams of this sort are being worked in the Radstock portion of the Somerset coalfield, and in the Forest of Dean. Explosions of coal-dust have now occurred in two of these mines, which are free from fire-damp, and leave no room for question as to the inflammability of coal-dust. Still there are doubts in the minds of some as to whether fire-damp, although

Refractory coal-seams and very hard rippings, however, require blasting; to dispense with explosives, would mean an additional cost which the colliery could not bear.

In order, as far as possible, to prevent explosions, and limit their fatal effects when they occur, it is absolutely necessary to—

(1) render harmless all issues of fire-damp, by constant and sufficient ventilation;
(2) prevent, as far as practicable, coal-dust from being blown off the trams or carried down from the surface screens by the downcast current, and thus reduce the deposition of coal-dust on roadways;
(3) thoroughly damp all coal-dust which is unavoidably deposited in roadways;
(4) exclude all naked lights;
(5) prohibit all explosives and fuzes for blasting which are liable to produce flame outside the shot-holes;
(6) fire shots only during the intervals between shifts, and fire them electrically;
(7) employ the most competent men at all work involving risk and responsibility;
(8) enforce rigid discipline and strict compliance with all rules and regulations.

Then the diameter of the drum at the first and succeeding revolutions will be—

$x + t = $ 1st diameter.
$x + 3\,t = $ 2nd do.
$x + 5\,t = $ 3rd do.

and so on, and at the nth revolution the diameter will be represented by $x + t\,(2\,n - 1)$. The circumference will therefore be—

$\pi(x + t) = $ 1st revolution.
$\pi(x + 3t) = $ 2nd „
$\pi(x + 5t) = $ 3rd „
And $\pi\{x + t\,(2n - 1)\} = n$th „

Since this series of terms are in arithmetical progression, their sum may be obtained in the usual way, and it will be, $n\pi(x + nt)$ which is equal to the length of rope wound on the drum, and therefore the depth of the pit. Therefore, $d = n\pi\,(x + nt)$. Or by denoting the depth of the pit in yards by D, then

$$D = n\,(x + nt) \times \cdot 087266,$$

from which the following formulæ are deduced :—

$$D = n\,(x + nt) \times \cdot 0872 \qquad (1)$$

$$x = \frac{11\cdot 45915\,D}{n} - nt \qquad (2)$$

$$t = \frac{11\cdot 45915\,D}{n^2} - \frac{x}{n} \qquad (3)$$

$$n = \frac{\sqrt{11\cdot 45915 Dt + \left(\frac{x}{2}\right)^2} - \frac{x}{2}}{t} \qquad (4)$$

Thus $D = (nx + n^2t)\,\cdot 0872$

$$D = \cdot 0872\,nx + \cdot 0872\,n^2t$$

$$\cdot 0872\,nx = D - \cdot 0872\,n^2t$$

$$x = \frac{D}{\cdot 0872\,n} - nt$$

$$x = \frac{11\cdot 45915\,D}{n} - nt \qquad (2)$$

$$nt = \frac{11\cdot 45915\,D}{n} - x$$

$$t = \frac{11\cdot 45915\,D}{n^2} - \frac{x}{n} \qquad (3)$$

$$n = \frac{11\cdot 45915\,D}{nt} - \frac{x}{t}$$

$$n^2 = \frac{11\cdot 45915\,D}{t} - \frac{nx}{t}$$

$$n^2 + \frac{x}{t}\,n = \frac{11\cdot 45915\,D}{t}$$

$$n^2 + \frac{x}{t}\,n + \left(\frac{x}{2t}\right)^2 = \frac{11\cdot 45915\,D}{t} + \left(\frac{x}{2t}\right)$$

Number of revolutions of drum = 18·872.
Diameter of drum in feet at last revolution = 16·359.

$\frac{16·359 + 14}{2} = 15·1795$, mean diameter of drum.

$15·1795 \times 3·14159 \times 18·872 = 900.$

$$\left.\begin{array}{r}\text{Depth from surface to}\\ \text{meeting of cages}\end{array}\right\} = \frac{16·359 + 15·1795}{2} \times 3·14159 \times \frac{18·872}{2}.$$
$= 467·464$ feet.
$= 77$ fathoms, 5 feet, 5·568 inches.
say 77 fathoms, 5 feet, 6 inches.

$$\left.\begin{array}{r}\text{Distance of meetings}\\ \text{from the bottom of pit}\end{array}\right\} = \frac{15·1795 + 14}{2} \times 3·14159 \times \frac{18·872}{2}$$
$= 432·5$ feet.
$= 72$ fathoms, 0 feet, 6 inches.

And the accuracy of the result is shown by adding together

	fathoms.	ft.	inches.
	77	5	6
And	72	0	6
Depth of pit	150	0	0

If the formula already given be applied to the working of this question, precisely the same result will be obtained.

Question 155.—A shaft 34 fathoms deep is worked by a drum 3 feet in diameter. It is required to find the diameter of another drum which must be keyed on the same axis to wind from another shaft 21 fathoms deep, flat ropes, 1 inch thick, being used to coil one lap upon the other?

In considering this question it is obvious that there must first be ascertained the number of revolutions of the 3-foot diameter drum in drawing the cage up the 34-fathom shaft, and then as the same number of revolutions must draw the cage up the 21-fathom shaft, proceed to find the diameter of drum for the shallower shaft, knowing the depth, number of revolutions, and thickness of rope.

Adopting formula (4) given in answer 153, and using the same values for the symbols there given, the number of revolutions in the 34-fathom shaft $n =$
$$\frac{\sqrt{11·45915 \times 68 \times 1 + \left(\frac{36}{2}\right)^2} - \frac{36}{2}}{1} = 15·21$$ revolutions. Now use formula (2) given in Answer 153, to find the diameter of drum for the 21-fathom shaft, thus
$$x = \frac{11·45915 \times 42}{15·21} - 15·21 \times 1 = 16·432 \text{ inches}$$
as the diameter of drum required for the 21-fathom shaft.

Question 156.—A shaft 100 fathoms deep is worked by a drum 12 feet in diameter. What must the diameter of another drum be which is keyed to the same axis to wind from another shaft 120 fathoms deep, flat ropes being used, $\frac{3}{4}$ of an inch thick, to coil one lap upon the other?

This may be worked in a similar manner to the previous question, but proceed to work it algebraically in order to show another method of obtaining the same and brittle, breaking with a smooth conchoidal or shell-like fracture. It is very rich in gas, ignites readily, and burns with a bright flame like that of a candle. It scarcely soils the fingers when touched, and gives off very little smoke in burning. It frequently contains the teeth and scales of fishes. The cannel varieties of coal are generally considered to represent an intermediate stage of the change in chemical composition, which has resulted in the conversion of lignite into bituminous coal and are hence usually found to occupy a corresponding position in the coal strata. Varieties occur chiefly in Yorkshire, Lancashire, North Wales, and in the Lanarkshire, Mid Lothian, Fife, and Ayrshire coal-fields. Some of these varieties decrepitate and crack loudly on the fire. It is largely used for gas-making.

Lignite or brown coal is formed of a mass of vegetable matter, some varieties presenting the appearance of undecomposed wood. Its colour is from brown to pitch black, its lustre sometimes resinous, sometimes dull. It burns easily and gives a smoky flame and unpleasant odour. It has a much larger proportion of oxygen than the bituminous coals, and a large amount of water is generally present.

Brown coal is not found in the strata of the true carboniferous system, but in the Tertiary, Cretaceous, and Oolitic rocks. It is in fact the name given to all coals which occur in formations more recent than the Carboniferous period.

> *Question* 164.—What is a dynamometer? Sketch and describe it, and explain the method of its application for ascertaining the power given out by a prime mover. The power of a portable engine is tested by passing a strap or belt over the fly-wheel, which is 6 feet in diameter; one end of the belt is secured to a spring balance, and a weight of 200 lbs. hangs on the other end. What is the horse-power of the engine when the balance registers a tension of 100 lbs., and the fly-wheel makes 220 revolutions per minute?

A dynamometer is a measurer of power.

It is often of great importance to know the amount of work done by a prime mover over and above that which is required to overcome the friction of its parts. The dynamometer is employed for this purpose.

In the case of a steam-engine, by means of the indicator, the total amount of work performed is ascertained, and if that shown by the dynamometer be deducted from the total indicated work, the balance represents the amount of work absorbed by friction among the moving parts of the engine itself.

Fig. 682 shows a common form of dynamometer as applied to the steam-engine.

It consists of a friction-brake, A, loaded at one end by weights, B, to a known amount.

At the other end of the lever a steel band, C, is fixed. This band may be tightened or slackened by means of the nut D working on the screw attached to the end of the band. The appliance is placed on some attachment to the main shaft, such as the fly-wheel of the engine to be tested, or a pulley. This is done by placing blocks of wood, E E, between the lever and the engine-pulley, F, and also between the steel band and engine-pulley. After the engine is started, the nut D is turned until the blocks E E bear upon the engine-pulley F with sufficient force to cause such friction between the blocks and the periphery of the wheel that the weighted lever will be lifted and float between the stops G G. The purpose of the stops, which are securely fixed, is to prevent the weighted lever from being carried round with the wheel, or dropped down in the other direction. In adjusting the weights B, they must be varied, until the flotation of the lever is accurately effected. When the blocks, lever, and weights are thus

Question 168.—With what other minerals is coal usually associated?

Coal is usually associated with beds and nodules of ironstone, sandstone, gannister or fire-clay, iron pyrites, &c.

Question 169.—In what rock formations are the following minerals found in the British Isles?
(1) Coal beds; (2) Salt; (3) Lead; (4) Iron.

The principal formations are (1) Carboniferous beds; (2) Trias beds; (3) Carboniferous limestone and silurian schists; (4) Lias, carboniferous and silurian beds. Inferior coals are found in several formations, but these coals are not of a true order of beds, as the carbon period.

Question 170.—What are beds and veins of minerals, and how do they differ?

Veins are fissures in the earth's crust carrying metallic ores; beds are layers of mineral matter lying conformable to the stratification.

Question 171.—Explain the difference of a vein, a "true fissure" vein, and a lode.

An ordinary vein is a fracture or fissure containing metallic ores; a true fissure vein is a vertical mass of metallic ores, of indefinite depth and longitudinal extension, but of a definite thickness; and a lode is a wall of mixed minerals enclosed in a fissure dipping at angles between the vertical and 70°, intersecting the successive rock masses with which it comes in contact.

any one of the "high" or violent explosives is employed, in one or other of the modes described, in substitution for powder.

But the methods of operation which furnish effective safeguards when applied in conjunction with the "high" explosives, fail to furnish such safeguard when applied in the same way, together with *powder*.

Unless, therefore, effective measures be adopted for the removal of dust, as completely as practicable, in the vicinity of the place where the shot is to be fired, such removal being followed by copious watering, the employment of powder, or of any explosive preparation of a similar nature to powder, should be prohibited in dry coal-mines where fire-damp may pervade the air, and where at the same time coal-dust accumulations are unavoidable.

Precautions to be Observed with Blasting.

With the view of promoting security from accidents under circumstances where blasting may be practised in coal-mines, we would recommend that the following instructions be observed:—

1. That all work involving blasting in mines should be entrusted only to experienced workmen.
2. That in order to lessen the risk from blown-out shots, particular care should be taken that each shot should be assisted by under-cutting and nicking or shearing whenever it is practicable.
3. That the tamping, stemming, or ramming should consist of very damp or non-inflammable material.
4. That where strong tamping is needed, the compression of air at the bottom of the hole should be avoided by pushing in the first part of the tamping in small portions.
5. That where safety lamps are used, and powder is employed, the shots should be fired only by specially appointed shot-men, who before firing the shots shall satisfy themselves that the foregoing instructions are observed, and shall also satisfy themselves, by carefully examining all accessible contiguous places within a radius of 20 yards of the shots to be fired, that fire-damp does not exist to a dangerous extent.

Precautions in Firing Shots.

The employment of the ordinary miner's fuze, which, when burning, is liable to allow fire to escape from its extremity or laterally into the atmosphere, should not be permitted in any mine-workings where the exigencies of safety dictate the exclusion of powder and the substitution, for it, of one or other of the "high" explosives in conjunction with water.

Similarly, no description of mining fuze, however safe in itself, should be allowed to be ignited in such localities, by means either of a lamp-flame, or of a wire which has been made red-hot by inserting it into the gauze of a safety-lamp, or by means of any other source of fire, which when applied to the lighting of the fuze, must come into contact with the atmosphere of the mine.

Use of Electricity for Firing.

Electrical exploding appliances present very important advantages from the point of view of safety, over any kind of fuze which has to be ignited by the application of flame to its exposed extremity, as the firing of shots by their means is not only accomplished out of contact with air, but is also under most

The power and uniformity of illumination given by a lamp can be notably improved by using, as the illuminant, vegetable or animal oil mixed with about one-half of its volume of a petroleum oil of safe flashing point.

Danger of Volatile Illuminants.

The use of petroleum spirit or benzene as the illuminant in safety-lamps, instead of vegetable or animal oil, is attended with some advantages, but it is also liable to introduce new sources of danger. Special care is needed in the filling and trimming of lamps, and in the arrangement of lamp-rooms, to avoid the ignition of the highly explosive mixture formed by air with the vapour arising from this spirit.

The selling of petroleum spirit, or of spirit of similar character as to volatility, under designations which are calculated to mislead in regard to the nature of the illuminant, is a proceeding fraught with danger, unless all vessels containing such illuminants bear a prominent label, indicating the dangerous nature of their contents.

Stringent regulations as to the conditions under which illuminants of this class are to be used and stored are absolutely necessary.

Electric Lighting.

The advantages in point of convenience and efficiency which attend the employment of electric glow-lamps for illuminating the pit's bottom, and roadways immediately adjacent to it, have already been demonstrated at several collieries, where this utilisation of the electric light has been combined with illumination at the surface by arc-lights.

In applying electric glow-lamps to underground illumination, to the extent indicated, through the medium of conducting cables leading from the generators to the pit bottom, it is essential to safety, as well as to the permanent efficiency of the installation, that the cables should be placed in positions where they are thoroughly protected against possible accidental injury. It is also essential, in all mines where fire-damp has been known to occur, that the glow-lamps should be excluded from direct contact with the air of the mine, in one or other of the ways indicated in this Report.

Portable, self-contained electric lamps have been devised, which will furnish, for several successive hours, a light considerably superior to that of the best safety-lamps, and which at the expiration of eight hours and upwards will still give a light fully equal to that of a freshly lighted Davy lamp. These lamps are perfectly safe, but, as they do not afford any indication of the condition of the atmosphere in a mine, their employment, even if special fire-damp detectors are used, cannot in any case entirely dispense with the necessity for the use of some safety-lamps.

For exploring purposes after accidents, or in foul places, these lamps must prove very valuable even in the present condition of their development, and as auxiliary lights they cannot fail to prove very useful. The great progress which has recently been made in the construction of portable electric lamps affords promise of a speedy utilisation of such lamps to an important extent in coal mines.

Over-winding.

Whilst we think that the safety-hooks at present available may have contributed to prevent fatalities from over-winding, we believe that the best appliance for the purpose is an automatic steam brake attached to the winding-gear, and we think it desirable that such brake should be introduced where practicable.

APPENDIX II.

THE EXPLOSIVES IN COAL MINES ORDER, 1897.

DATED DECEMBER 20, 1897.

WHEREAS by section 6 of the Coal Mines Regulation Act, 1896, it is enacted that a Secretary of State, on being satisfied that any explosive is, or is likely to become, dangerous, may by order prohibit the use thereof in any mine or in any class of mines, either absolutely or subject to conditions:

I hereby, in pursuance of the power conferred on me by the aforesaid section, make the following Order:—

Absolute Prohibition of Certain Explosives in Unsafe Mines.

1.—(1) In all coal mines in which inflammable gas has been found within the previous three months in such quantity as to be indicative of danger, the use of any explosive, other than a permitted explosive, as hereinafter defined, is absolutely prohibited in the seam or seams in which the gas has been found.

(2) In all coal mines which are not naturally wet throughout, the use of any explosive, other than a permitted explosive, as hereinafter defined, is absolutely prohibited in all roads, and in every dry and dusty part of the mine.

Conditional Prohibition of other Explosives in Unsafe Mines.

2. In all such coal mines or parts thereof as aforesaid, the use of permitted explosives is prohibited unless the following conditions are observed:—

(*a*) Every charge of the explosive shall be placed in a properly drilled shot hole and shall have sufficient stemming:

(*b*) Every charge shall be fired by an efficient electrical apparatus, or by some other means equally secure against the ignition of inflammable gas or coal dust:

(*c*) Every charge shall be fired by a competent person appointed in writing for this duty by the owner, agent, or manager of the mine, and not being a person whose wages depend on the amount of mineral to be gotten:

(*d*) Each explosive shall be used in the manner and subject to the conditions prescribed in the schedule hereto:

Provided that nothing in this Order shall prohibit the use of a safety fuze in any mine in which inflammable gas has not been found within the previous three months in such quantity as to be indicative of danger.

Conditional Prohibition of all Explosives in Main Roads.

3. In every coal-mine the use of any explosive is prohibited in the main haulage roads and in the intakes unless all workmen have been removed from the

composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium);

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "as defined in the list of permitted explosives"; and, further, that each inner package shall be clearly marked with the words "permitted explosive, to be used only with not less than No. 6 detonator," and marked with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients;

(4) That the explosive, if in a frozen condition, shall be thoroughly thawed before use in a safe and suitable manner.

Electronite, No. 2, consisting in every 100 parts by weight of the finished explosive of not more than 96 parts and not less than 94 parts of nitrate of ammonium, with not more than six parts and not less than four parts of wood-meal and starch, and with no other ingredient. Provided—

(1) That the explosive shall be used only when contained in a waterproof metal case made of an alloy of lead and tin;

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as No. 6 (*i.e.*, the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 15 grains of a composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium); and

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "as defined in the list of permitted explosives"; and, further, that each inner package shall be clearly marked with the words "permitted explosive, to be used only with not less than No. 6 detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients.

Faversham Powder, consisting in every 100 parts by weight of the finished explosive of not more than 87 parts and not less than 83 parts of nitrate of ammonium with not more than 14 parts and not less than nine parts of thoroughly purified di-nitro-benzol, with not more than two parts and not less than one part of chloride of ammonium, and not more than three parts and not less than two parts of chloride of sodium, and with no other ingredient; the whole being uniformly incorporated. Provided—

(1) That the explosive shall be used only when contained in a case of paper thoroughly waterproofed with paraffin wax, and with or without a lead nozzle;

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as 6½ (*i.e.*, the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 19 grains of a composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium);

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "as defined in the list of permitted explosives;" and, further, that each inner package shall be clearly marked with the words, "permitted

(4) That the explosive, if in a frozen condition, shall be thoroughly thawed before use in a safe and suitable manner.

Nobel Carbonite, consisting in every 100 parts by weight of the finished explosive of not more than 27 parts and not less than 25 parts of thoroughly purified nitro-glycerine, and not more than 36 parts and not less than 30 parts of nitrate of potassium, and with not more than 43 parts and not less than 40 parts of wood-meal, with or without not more than half a part of sulphuretted-benzol, and not more than half a part of carbonate of sodium and carbonate of calcium or either of them, and with no other ingredient; the whole being uniformly incorporated and of such character and consistency as not to be liable to exudation. Provided—

(1) That the explosive shall be used only when contained in a non-water-proofed wrapper of parchment paper;

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as No. 6 (*i.e.*, the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 15 grains of a composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium);

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "as defined in the list of permitted explosives"; and, further, that each inner package shall be clearly marked with the words "permitted explosive, to be used only with not less than No. 6 detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients; and

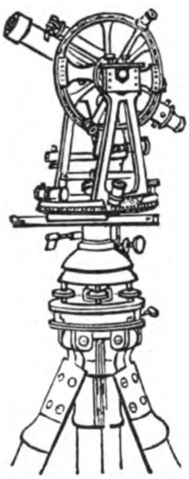
(4) That the explosive, if in a frozen condition, shall be thoroughly thawed before use in a safe and suitable manner.

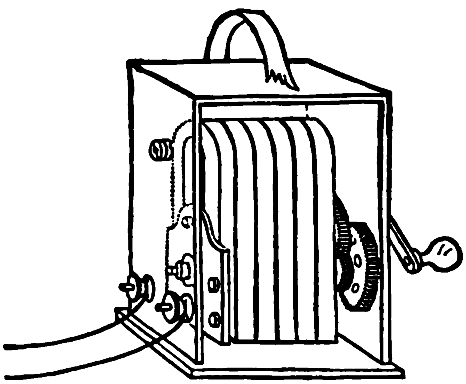
Nobel Gelignite, consisting in every 100 parts by weight of the finished explosive of not more than 63 parts and not less than 54 parts of thoroughly purified nitro-glycerine, with not more than 5 parts and not less than 3 parts of nitro-cotton, carefully washed and purified, not more than 34 parts and not less than 26 parts of nitrate of potassium, and not more than nine parts and not less than six parts of wood-meal and with or without not more than half a part of chalk and with no other ingredient; the whole being uniformly incorporated, and of such character and consistency as not to be liable to exudation. Provided—

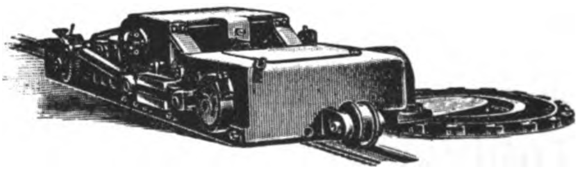
(1) That the explosive shall be used only when contained in a non-water-proofed wrapper of parchment paper;

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as No. 6 (*i.e.*, the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 15 grains of a composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium);

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "as defined in the list of permitted explosives"; and, further, that each inner package shall be clearly marked with the words "permitted explosive, to be used only with not less than No. 6 detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients; and





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