MODERN PRACTICE IN MINING

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VOL. IV

THE VENTILATION OF MINES

BY

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WITH TWO FOLDING PLANS AND OTHER ILLUSTRATIONS

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PREFACE

So far as the author is aware there is no work devoted entirely to a comprehensive consideration of the subject of mine ventilation, though there are several excellent books which treat of certain sides of the subject. It is hoped, therefore, that this little volume may be of some service to the young mining student in arriving at an elementary knowledge of the principles and practice of mine ventilation.

For permission to use those of the figures which have not been specially drawn for illustrating the text, the author desires to tender his thanks to the various authors, manufacturers, and institutions from whom they have been borrowed.

Grateful acknowledgment is also made to Mr. G. H. H. Scott, who has devoted much time to correcting the proof sheets, and also to Mr. G. Poole for his kindly help in the same direction.

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CHAPTER I

INTRODUCTION: A SHORT HISTORY OF THE SUBJECT OF THE VENTILATION OF MINES

THE ventilation of mines is necessary for the fulfilment of two objects, viz. to render the hygienic conditions as good as possible by the dilution and removal of deleterious gases and so provide the men and animals with a sufficiency of respirable air; and to make the mine as safe as possible by preventing the accumulation of explosive gases.

When there are no currents of fresh air circulating through a mine, the stagnant atmosphere not only becomes partially depleted of its oxygen by the breathing of men and animals, and, to some extent, through absorption by the strata, especially in coal-mines—but it is also contaminated by the exhalations of men and animals, and by the gases generated in the strata, from decaying wood, &c. Accordingly, in the words of the official abstract of the Coal Mines Regulation Act (1887): "An adequate amount of ventilation must be constantly produced in every mine to dilute and render harmless noxious gases to such an extent that the working places of the shafts, levels, stables, and workings of the mine, and the travelling roads to and from those working places, shall be in a fit state for working and passing therein."

The necessity for artificial ventilation has proved more pressing in the case of coal-mines than in that of metalliferous mines, chiefly because of the more copious generation of noxious gases in the former as compared with the latter, but also to some extent on account of the more extensive area worked and the greater number of persons employed in a coal-mine. The ores worked in metalliferous mines, excluding stratified ironstones, are obtained almost entirely from highly inclined veins, impregnated beds, or intrusive dykes. Hence the area is dependent on depth and lateral extension, factors which are much more uncertain than length and breadth in stratified deposits such as coal or ironstone. It has been customary, therefore, in the case of metalliferous mines in the past to depend on the variable and uncertain "natural" ventilation, instead of causing large volumes of air to be forced or drawn down into the mines and properly directed through the workings by artificial means. Latterly, however, owing to the indifferent hygienic conditions existing in some metalliferous mines in some parts of the world, and the extensive depth and lateral extension of the same, artificial ventilation has been resorted to with most successful results (see p. 209), and might with advantage be extended to other mines of similar character.

The present improved state of ventilation in coal-mines has not been quickly attained, as is shown by the following historical retrospect.

In the early days of systematic coal-mining, about the fourteenth century, at which time coal was mined by shaft and adit, and narrow galleries were driven in the seam and small pillars left to support the roof; when difficulties arose from want of air, or from too much water, or from crush of the strata, the mine was abandoned and a new one opened out near by, each such mine being only a few acres in extent.[1] The fifteenth, sixteenth, and seventeenth century showed small extension or improvement in mining. It was in the latter century that we have the earliest account extant of a colliery explosion, contained in a paper contributed by one Roger Mostyn in 1677 to the Royal Society;[2] the explosion having occurred in February 1675 at the Mostyn Colliery in North Wales.

It was not until the invention of the atmospheric engine by Thomas Newcomen in 1710 that any real or marked developments took place in mining. At this time the deepest coal-mine shafts in Great Britain did not exceed 360 feet, their diameter 7 or 8 feet, and the radius of the area worked to one shaft was seldom more than 200 yards.[3] In 1750, Walker Colliery, on the banks of the Tyne, was sunk to a depth of 600 feet, which was considered a very remarkable attainment.

The explosion of the Mostyn Colliery has been mentioned, and much trouble seems to have been occasioned during the latter half of the seventeenth century by fire-damp, for in a paper on "Firy Damps in Mines" communicated to the Royal Society in 1681 by Mr. J. Beaumont,[4] he mentions that in the middle and more easterly coal works of the Mendip district there was

[1] Thus in 1356 Bishop Hatfield of Durham granted on lease five mines which the lessees were to work as far as they could be wrought by five barrowmen according to the view and oath of the Chief Forester and of the Viewers ("par Cynk Barrowmen par la vewe et serrement du chief fforester et des Veiours"), and they were limited to one keel a day (about 20 tons). *Archæologia Æliana*, Part 24, vol. viii., Society of Antiquaries of Newcastle-on-Tyne.

[2] *Phil. Trans.*, No. 136, p. 895.

[3] *Colliery Working and Management*, by H. F. F. Bulman and R. A. S. Redmayne, 2nd ed., p. 3.

[4] *Philosophical Collections*, No. 1, p. 6.

scarcely a pit in which firedamp was not present. He describes how that many persons had of "late years been there killed, many others maimed and burnt; some have been blown up at the pit's mouth." "Quickening" the air, and the use of very thin candles, were tried as a preventative of these results. At many collieries the firedamp which accumulated during the night was got rid of by burning or exploding it next morning, the practice being for one of the colliers to descend the mine in advance of the others, and, having saturated his clothing with water, to creep forward holding a long pole before him with a light attached to the end of it, and when the firedamp was ignited near the roof to lie flat on the floor till the flame passed him. The movement of the workmen in these mines of restricted area prevented the accumulation of the gas during the daytime.

No artificial means were taken to induce a draught in the mines, such ventilation as there was being entirely due to natural means, but the erection of stoppings in the excavated roads in order to conduct such currents as did exist right up to the coal face was practised as early as the fourteenth century, and was known as "face-airing." James Spedding, the son of Carlisle Spedding, the inventor of the steel mill (1740)[1] for lighting gassy mines (Fig. 1), introduced, about 1760,[2] the "coursing of the air" in place of "face-airing." By this means the air current was made to traverse the whole of the excavations

[1] A steel wheel was made, by means of gearing worked by hand, to rotate rapidly against a piece of flint, and it was considered that the sparks produced thereby afforded a safe light in dangerous parts of the mine. As many as one hundred of these instruments were in daily use in a fiery mine. *Report of the South Shields Committee*, 1848.

[2] Mr. Mathias Dunn regards this as the suggestion of the father, for he remarks, in *Working of Collieries*, 2nd ed., 1852, p. 151: "The ingenious Mr. Spedding, who is recorded as the inventor of the steel mill, suggested the coursing of the waste, which consisted of threading the air column up certain workings, and down others, until it ventilated the whole waste."

or nearly so, so that the back workings might not constitute a magazine for the accumulation of firedamp. Mr. Mathias Dunn says:[1] "It often happened that the state of the air in the fallen wastes was such that the naked candle could not be made use of, in which the only resource was the steel mill, the invention of Mr. Spedding before mentioned." He also states "that it was no uncommon occurrence for the air to travel thirty or forty

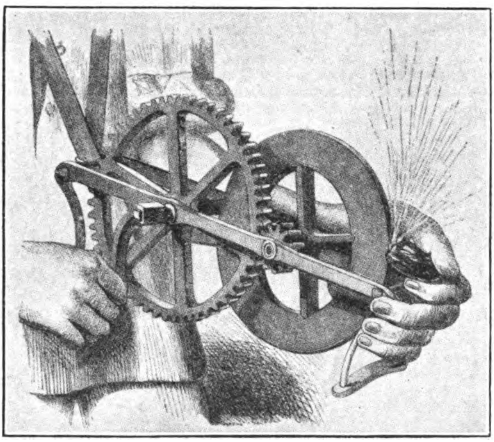

FIG. 1.—The Steel Mill.

miles from leaving the downcast pit to regaining the surface." The method of "coursing the air" has still to be partially resorted to under certain contingencies (see p. 206), but as a general system it was soon found to be inadequate for the proper ventilation of extensive workings. Figs. 2 and 3 represent diagrammatically ventilation by means of "face-airing" and "coursing the air."

Artificial or induced ventilation is first mentioned in the seventeenth century, when in 1606 Dr. Plot (Plot's *Staffordshire*, p. 138) says, speaking of damps in mines:

[1] *Working of Collieries*, 2nd ed., 1852, p. 154.

"Damps occasioned by smoke,[1] they expel either by water, where they have no air pits, and in winter time, but chiefly by fire which they let down in an iron cradle they call their lamp, which very way they use about

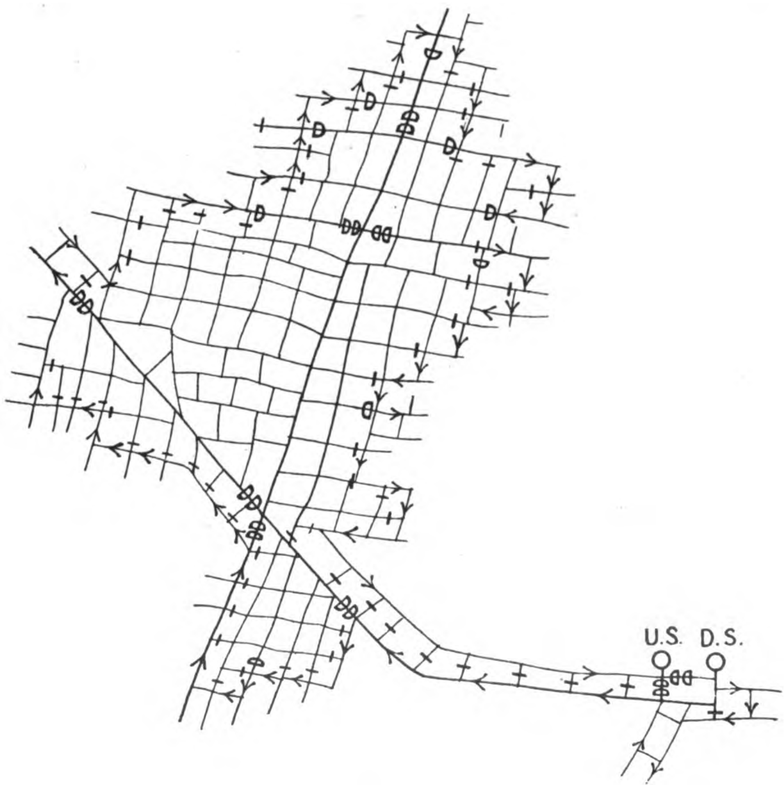

FIG. 2.—"Face-airing" of the Workings of a Coal-mine in the Eighteenth Century. The direction of the air-current is denoted by arrows; the position of doors are indicated by a D; and the stoppings are shown thus —.

Chedle." When furnaces came to be applied underground is uncertain, there being no definite information on the point. At the time the *Compleat Collier* was

[1] Page 152. The smoke being due to "fire-setting"—that is, to the fires which were lit to heat stone on which water was poured with a view to breaking it.

written (1708) it was regarded as too dangerous to introduce a furnace into the fiery mines of the North of England, but in 1785 we find that there was a ventilating furnace at the bottom of the B Pit of the Wallsend Colliery on the Tyne, and at which on one occasion the return air, overloaded with firedamp, ignited. The condition of an air current which had traversed thirty miles of roads in a fiery mine can be easily imagined, and it

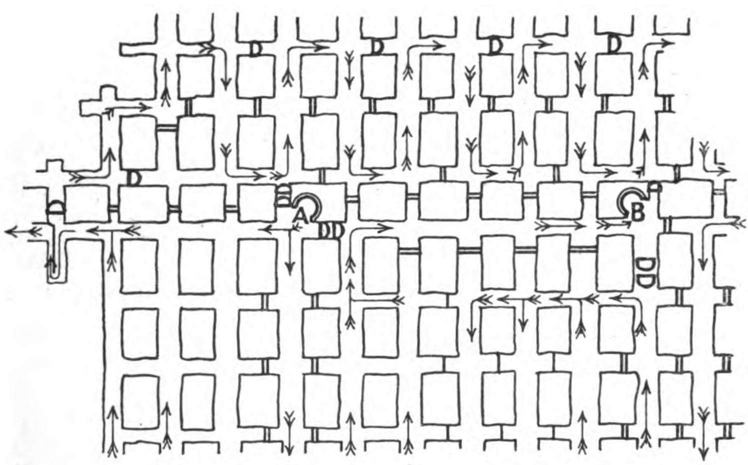

FIG. 3.—"Coursing the Air."

A. Downcast shaft. B. Upcast shaft.
D. Wooden separation doors. || Stone or brick stoppings.
| Canvas sheets, or stoppings with a scale.

is not to be wondered at that, on occasions, firedamp was seen to ignite on the current passing over the furnace.

In 1810 Mr. John Buddle, sometimes known as the "father of the coal trade" in the North of England, introduced the "panel" system of laying out a colliery, which consisted of working the mine by a number of districts separated from each other by ribs or barriers of coal, the idea being the localisation of the effects of "creep" and the limitation of the force of explosions should such occur. To all intents and purposes, therefore, each district constituted a miniature mine. This way of working led him to devise the further system of "compound" ventilation—that is, of dividing the air current into separate splits, so that each panel or district had its own separate supply of fresh air, having its own intake and return air current. This is now known as "splitting the air," and is the system of ventilation practised at all collieries (see Fig. 13, p. 81).

Repeated disasters at collieries led to the formation at Sunderland in the North of England, in 1813, of "A Society for the Prevention of Accidents in Coal Mines." This Society was instrumental in engaging Mr. (afterwards Sir) Humphry Davy, who in 1815 invented the safety lamp.

The story of the first experiment with this lamp in the North of England is interesting and somewhat dramatic. The experiment was carried out by the Rev. John Hodgson, Vicar of Jarrow-with-Heworth, near Newcastle-on-Tyne. Mr. Hodgson was a member of the Sunderland Committee.

The Vicar went down Hebburn Colliery with the new lamp in his hand. A correspondent, writing to the *Newcastle Daily Chronicle*,[1] thus describes the incident: "A solitary man was working by no other light than the sparks of a steel mill; no notice had been given him of what was about to take place. He was in an atmosphere of great danger, when he saw a light approaching—apparently a candle burning openly—the effect of which he knew would be instant destruction to him and its bearer. His command was instantly, 'Put out that candle!' (with a threat to send his pick through the body of the advancing intruder). The flame came nearer and nearer; no re-

[1] *Newcastle Daily Chronicle*, February 4, 1888.

gard was paid to his cries, which then became of the most terrific kind, mingled with imprecations against his comrade (for such he took Hodgson to be), who was tempting death in so rash and certain a way. Still not one word was said in reply. The light continued to approach until there stood before him, silently exulting in his heart, a grave and thoughtful man—a man whom he knew and respected, who four years before had buried in one common grave ninety-one of his fellow-workmen, holding up in his sight the triumph of science, the future safeguard of the pitmen."

Although there can be small doubt that the introduction of the safety lamp into "fiery" (gassy) mines was marked by a considerable falling off in the number of small colliery explosions, and so witnessed the saving of many lives sacrificed under the old conditions,[1] yet extensive and disastrous colliery explosions continued to occur, and the improvement in the ventilation of the coal-mines, which kept pace with the general expansion that characterised the coal trade at this period of its history, would seem to have had no mitigating effect. As the collieries became more extensive, the death-rate per explosion became greater. The era of big explosions may be said to have commenced with the nineteenth century, and to have continued to the present time. The reason for this is now well understood. A factor other than firedamp plays an important part in colliery explosions, namely, coal dust, but for many years its importance was not realised. Messrs. Lyell and Faraday, in 1845, when discussing the Haswell Colliery explosion in the pages of the *Philosophical Magazine*, and afterwards

[1] There are no official figures on which to base this assertion, but a study of records, letters, journals, and such other data as are available, are sufficient for the purpose, if the fact is not self-evident.

in their official report on that disaster, drew attention to some of the effects of coal dust as bearing upon the extent of the fire consequent upon an explosion of firedamp, and they said further: "There is every reason to believe that much coal gas was made from this dust in the very air itself of the mine by the flame of the firedamp, which raised and swept it along." But it was not until Mr. W. Galloway contributed a series of articles, entitled "Coal Dust Explosions," to a paper named *Iron*, in 1878, that the British mining public began to realise that coal dust itself might be the cause of colliery explosions. In these articles Mr. Galloway gave complete translations of some French papers dealing with the subject. In France, M. du Sonich, when reporting in 1855 and 1861 on an explosion at Firminy Colliery, alluded to the aggravating effects of coal dust in the inflammation of the gases. M. Verpilleux wrote much in the same way in 1864, and in 1872 M. Villiers asserted that coal dust had played a very important part in the explosion at the Jobin Colliery, and so did other French authors; but M. Seibel, Director of the Campagnas Mines, writing on December 5, 1874, expressed himself as convinced that an explosion at one of those mines, which happened at the commencement of the previous month, was due to coal dust alone; and in 1875[1] made experiments with coal dusts, arriving at the following conclusions, viz., that coal dust in well-ventilated workings may of itself give rise to disasters; in fiery workings it increases the chance of explosions, and that when accidents of this nature do occur it aggravates their consequences. And this may be said to be the present state of the case; but it is only of late years that the mining profession as a whole has come to concur in these results and accept them in their entirety.

[1] *Annales des Mines*, 1875.

In helping to a general recognition of the dangers of coal dust great credit is due to Mr. Galloway, who studied the question both from the theoretical or experimental standpoint and practically. Mr. H. Hall, H.M. Inspector of Mines, also carried out experiments with coal dust, and Messrs. J. B. and W. N. Atkinson contributed a valuable and illuminating work, in which they discussed the subject in relation to a number of colliery explosions, of which they had first-hand knowledge.

It is curious that so many experienced and able mining engineers should have opposed the coal-dust theory for so long, seeing that it was well known that explosions at flour mills were caused by the ignition of the flour dust. The present writer was himself unconvinced until, when a member of the Committee appointed by the North of England Institute of Mining and Mechanical Engineers in 1888, to make investigations and report upon the action in respect of inflammability of the so-called "flameless" explosives, he witnessed an ignition of coal dust which was accompanied by considerable violence, in a wrought-iron tube 3 feet in diameter and 101 feet in length.

Eventually the subject began to be regarded as one of such importance that in February 1891 a Royal Commission was appointed to inquire into coal dust and colliery explosions (reported in the same year), and a Committee of Coal Owners at the close of the year 1910 issued an elaborate report on a series of experiments on coal dust carried out on a large scale at Altofts in Yorkshire during the years 1908 and 1909.

This brief *résumé* of the history of mine ventilation and of colliery explosions will, it is hoped, put the reader in possession of the chief incidents in the development of this important subject, and serve to show how slow has been its growth. Perfection has by no means yet been attained, and mining engineers are still faced by problems as important as those surmounted by their forefathers, and requiring for their solution perseverance and ingenuity of a value equal to that hitherto exercised, and worthy of the earnest study of the best minds. Colliery explosions are still not infrequent, some of the disasters resulting in an appalling loss of life. All the care possible on the part of officials and workers, though it will materially reduce fatalities in mines, will not put a stop to *all* forms of accidents; but in respect of colliery explosions there is no absolute impossibility of prevention, seeing that neither firedamp nor coal dust will explode if not ignited. Force to "get" the coal, and light to enable the work to be performed, are necessities, and coal has to be obtained at a profit, otherwise it would not be worked; but much has been done in the past towards procuring safer means in these respects, and it is not too much to hope that before long absolute prevention of colliery explosions will be attained. Certainly greater strides in the way of precautionary measures have of late years been devised.

Another difficult problem is that in respect of the ventilation of very deep mines. When deposits of coal exist at great depth from the surface, and below strata the character of which necessitates the adoption of special methods of sinking, a huge capital is expended; and, in order that the investment may prove profitable, *i.e.* that a return of the capital may be secured within a measurable period together with a fair rate of interest, large outputs of coal are required. A considerable coal area is necessary, in order that the life of the mine may be such as to allow of the large outputs being continued over a long period, hence the distance of the workings from the shaft bottom will ultimately be very great—a feature which would also obtain in the case of working under-sea coal, even if the question of excessive depth were non-existent. Seeing that development in colliery enterprise is increasing in the direction of working coal at great depth, owing to the gradual exhaustion of the shallower portions of the coal-fields, two important ventilation problems will become more and more insistent, viz. the question of heat increment, and that of great frictional resistance to the circulating air currents. As the present writer has said elsewhere:[1] "The only points upon which we are certain in respect of heat gradients would appear to be that (1) the total length of the geothermic degree is quite undecided; (2) the length is different in different districts; (3) isogeotherms are not parallel down to a depth of 5000 feet; and (4) I do not think we need apprehend any difficulty due to heat increment in working down to a depth of 4000 feet that cannot be overcome by sinking large shafts and causing large volumes of air to circulate through the workings, and by, perhaps, some modifications in our present mode of ventilation. Undoubtedly some system of local or secondary ventilation will have to be resorted to, some arrangement whereby numerous splits of cool fresh air shall be taken off one or more intakes and circulated round only a circumscribed area of workings before being drawn to the surface. As emphasising this view, take for instance the case quoted by Mr. G. A. Mitcheson at Florence Colliery, in North Staffordshire. Though they have there a face temperature as high as 82·5° F., some places are as cool as 73° F., owing to

[1] Presidential Address by R. A. S. Redmayne before the South Staffordshire and East Worcestershire Institute of Mining Engineers, October 17, 1904. *Trans. Inst. M.E.*, vol. xxviii., pp. 37–46.

the fact that there is a greater current of air traversing a limited length of face, in the instance in mind about 3000 feet." Helping-up fans, driven by electricity, or where the conditions are not safe enough to warrant that mode of transmitting power, by compressed air, "might also with advantage be placed in the return air-ways near the face to give velocity to an otherwise sluggish current of air, or to draw artificially cooled air through pipes and distribute the same at the face. Some such devices as these may, I have no doubt will, be satisfactorily worked out and adapted to the needs of deep mining. It should not, at any rate, be difficult to increase greatly the velocity of the ventilating currents in the main intakes, this being chiefly a question of mechanical power."

CHAPTER II

THE ATMOSPHERE OF COAL AND OTHER MINES

Air—Composition and Physical Properties of Air—Moisture in the Air—Effects of Moisture in respect of Colliery Explosions—The Physiological Effects of Humidity—Temperature of Mines and Moisture in the Atmosphere—Observations in Belgian Mines—Temperature and Humidity of Metalliferous Mines—The Volume and Weight of Air under Variations of Temperature and Pressure—The Physiology and Chemistry of Breathing.

BEFORE passing to a consideration of the means to be taken for the production of the ventilating current, and the direction of the same into the proper channels underground, it is fitting that some account of the constitution—the chemical and physical properties—of the atmosphere should be given, namely, of the fresh air which it is sought to circulate through the mine; as well as of the gases produced in the mine, whether evolved from the coal or the strata enveloping the coal, produced by the respiration and excretions of men and animals, or by the combustion of explosives and burning of lamps and candles.

The Composition of Air.—When the present writer was a college student, one was taught that pure air was a mechanical mixture (as opposed to a chemical combination) of oxygen and nitrogen with a minute quantity of carbonic acid. It has since, however, been found by Rayleigh and Ramsay to contain another gas in minute quantities, viz. argon. Thus *pure* dry air, according to Haldane, consists by volume of—

	Per Cent.
Oxygen	20·93
Nitrogen	78·10
Argon	0·94
Carbonic acid	0·03
	100·00

But for all practical purposes, in so far as mine ventilation is concerned, we may continue to regard it as a mechanical mixture of oxygen and nitrogen, namely, 23 parts by weight of oxygen and 77 parts by weight of nitrogen, or in point of volume, 21 of oxygen and 79 of nitrogen. Analyses of air taken from different parts of the world have shown practically no difference to exist in respect of the relative proportions of oxygen and nitrogen. Of these two constituent gases, it is the first that is necessary to the existence of animal life. Without oxygen in the air all animal life on the surface of the earth would cease. In the process of respiration a man breathes in air, filling his lungs therewith, and the blood coursing through the system and passing into the lungs absorbs some of the oxygen in the air, the resulting oxidation producing the heat of the body which is necessary for the maintenance of life. The carbonic acid produced by this "combustion"—for the process is chemically similar to that of burning—being exhaled in the act of expiration or emptying of the lungs, the expired air containing from 3 to 6 per cent. of carbonic acid gas, the volume of air expired being from 350 to 700 cubic centimetres at each exhalation. Cessation of this process results in death. In order that life may be maintained, the air breathed must contain a certain minimum quantity of oxygen.

One sometimes hears it stated that men in mines have been poisoned by carbonic acid; but although symptoms similar to poisoning do appear when an atmosphere containing a very high percentage of carbonic acid is inhaled, in most cases death has resulted not from poisoning by carbonic acid, but from the effects produced by a shortage of free oxygen. This question is further dealt with when considering the properties of carbonic acid gas on p. 51. But the effects of a superabundance of free oxygen would be equally disastrous. A man could not live for long in an atmosphere of pure oxygen; the result would be a too rapid oxidation of the blood, too great animation, and exhaustion would rapidly ensue. An adulterant is therefore necessary, and this is the part played by the nitrogen.

The globe on which we live is enveloped by a mantle of air. We live, as it were, at the bottom of an ocean of air which presses upon each of us to the extent of several tons weight, but as it acts equally in every direction we do not feel it. The "thickness," if one may use the term, of this atmospheric envelope is about 45 miles, calculated from sea-level. The density of the air increases as the surface of the earth is approached, and so rapidly does it rarefy away from the surface, that life, so far as man is concerned, could not for long be supported at an altitude of five miles.

Moisture in Air.—Aqueous vapour is contained in air in varying quantities, the amount being governed by the seasons, the locality, and the temperature. Air of a given temperature cannot contain more than a certain quantity of moisture in solution. When it contains this quantity it is said to be *saturated*, and the greater the temperature the higher the saturation point, so that if saturated air is cooled the vapour falls as moisture. Thus if 1 cubic metre of air is saturated at 25° C. (68° F.) it will contain 22·5 grammes of moisture, but if the temperature be reduced to 0° C. (32° F.) it will lose 17·1 grammes, as it cannot, at the latter temperature, contain more than 5·4 grammes.

The percentage of moisture in the atmosphere of the coal-mine has an important bearing in two directions, viz. (1) as to the liability or non-liability of the mine to explosion; and (2) physiologically in respect to the limit of human and animal endurance.

The Effect of Humidity in respect of Liability to Colliery Explosions.—It is a well-established fact that, when conducting experiments on coal dust, if the air is moist, violent or extensive ignitions have seldom been secured with dusts which, when the atmosphere was fairly dry, were easily effected. During the carrying out of the experiments by the Prussian Commission in 1884, it was found that on foggy days no explosions could be obtained, as only a very slight dampness adversely affected the inflammability of the dust.

Colliery explosions are of more frequent occurrence in the winter than the summer months, owing to there being less moisture in the colder than the warmer air. The cold air entering the mine absorbs some of the moisture, and renders the coal dust drier. In view of this fact it would be interesting to collect statistics with the view of determining whether coal-mines situate in high latitudes are more subject to explosions than those lower down.

It is a fact also that vapour in the air adversely affects the explosibility of gases, the fine particles of water acting as heat absorbents, and it has been determined that a saturated air current of a temperature of 62° F. requires about 7 per cent. more heat units (B.T.U.) than dry air in order to attain a temperature of 1213° F., which is the ignition point of firedamp. Thus, petrol-driven automobiles and other internal combustion engines consume less fuel on a cold dry day than on one which is warm and moist.

The Physiological Effects of Humidity.—Experience and experiment have shown that a dry heat is more supportable than a moist heat, the effect of the moisture being to greatly increase the temperature of the body, and produce heat-stroke in what, were the atmosphere dry, would not be a formidable temperature. These conditions of great heat and moisture may exist in mines when swept by an explosive blast or when underground fires occur, but in some mines the normal temperature is so high as to have an important bearing from the physiological point of view.

The question of temperature and humidity are of importance not only in respect of their direct effect upon the human and animal body, but also in relation to the development of *ankylostoma.*

In moderate temperatures the humidity of the air has but small effect on the system or capacity for work, but when the wet-bulb thermometer attains the neighbourhood of 80° F. it is not possible to engage in continuous hard work without excessive fatigue and exhaustion. Dry temperatures, however, up to 110° F. are not necessarily injurious to health, but in the same temperature with the wet bulb registering 80° F., though showing only 16 per cent. of moisture in the atmosphere, a serious rise of body temperature would result from exertion.

The Temperature of Mines and Amount of Moisture in the Atmosphere.—The temperature of the air current, when some distance in from the shaft, varies but little summer or winter. In summer the hot air from the surface gives out some of its heat to the strata in passing through the mine. In winter the cold incoming current extracts heat from the surrounding strata. Other things being equal, the temperature of the mine varies directly with the depth, but, though this is so, no definite rule can be adduced (see p. 13). In Lancashire the ratio of increase of temperature varies from 1° F. in 60 feet to 1° F. in 80 feet of depth. Probably the highest normal temperature of any coal-mine in the United Kingdom is that at Pendleton Colliery, where the highest reading taken in the working faces was about 94° F., the depth from the surface at the shaft being 1545 feet, but owing to the dip of the seam about 1700 feet would have to be added to this in order to give the depth at the point at which the above reading was taken. Mr. Gerrard[1] has observed a temperature of 104° F. in a Belgian coal-mine at a depth of 3773 feet from the surface. It is worthy to note that both at this mine and at Pendleton Colliery the air was remarkably dry, the respective readings being:—

At Pendleton: Dry bulb, 93·3° F.; wet bulb, 79·7° F., or 49½ per cent. of humidity.

At the Belgian colliery: Dry bulb, 104° F.; wet bulb, 84° F., or 38 per cent. of humidity.

The present writer observed the temperature at the bottom of the shaft of a Belgian colliery, 3772 feet from the surface, to be 29° C. (84·2° F.), and no doubt the temperature was still higher in the workings.

A number of systematic observations made at the surface and in the mine of British collieries, at the instigation of Dr. Nasmyth,[2] are of interest in connection with the temperature of coal-mines and amount of moisture in the atmosphere. The wet and dry bulb thermometer (hygrometers) were placed in the intake air-way at a point 1000 yards from where the current commenced. The observations, which commenced in September 1887 and terminated in January 1888, were in every instance made at 9 A.M. The highest temperature recorded was on September 9, when 55° F. was

[1] Presidential Address by J. Gerrard, H.M. Inspector of Mines, to the Manchester Geological and Mining Society, *Trans. Inst. M.E.*, vol. xxviii. p. 364.

[2] *The Air of Coal Mines*, by T. G. Nasmyth, D.Sc., M.B., C.M., 1898.

indicated above-ground, the temperature in the mine being 55·5° F. The lowest temperature was on December 22, when the thermometer registered surface 25° F., and in the mine 53° F. The greatest differences below was 2·5° F. The relative humidity below-ground varied from 93° to 100°. Dr. Nasmyth says: "Practically the air is nearly always saturated. This excessive humidity is certainly not desirable from a sanitary point of view; but I do not know any bad consequences to the health of miners. The uniform temperature is certainly favourable." The temperatures in deep mines would, of course, be from 10° to 20° higher than those quoted above. The air current, even on dry and frosty days, will in the return air-ways of very many mines be saturated; the more or less dry air entering the mine taking up the moisture from the strata as it passes through the workings. The difference between the hygrometric conditions existent in the intake and return air-ways of a mine is frequently evidenced by the fact that fungoid growths, which though common on the timber in the return air-ways, seldom occur on that supporting the intake roads. Timber decays also more rapidly in the return air-ways as compared with that in the intakes. Hence also the dust in the mine is driest in the intakes, and will be drier in the winter than in the summer months.

Humidity and Temperature Observations in Belgian Coal-Mines.—The Belgian Coal Owners' Federation recently exhaustively investigated the subject of temperature and humidity in the mines under its control, carrying out in all about 24,000 determinations, with the following results:—

Temperatures exceeding 28° C. [82·4° F.] were observed,—
in 21 pits out of a total of 63 in the Mons district, reading obtained at an average depth of 940 yards;

in 7 pits out of a total of 96 in the Charleroi district, reading obtained at an average depth of 1240 yards;

in 3 pits out of a total of 60 in the Liége district, reading obtained at an average depth of 940 yards.

Central, none.

The highest temperatures in the districts being as follows:—

Mons . .	32° C.	(89·6° F.)	at a depth of		1150	yards.
Charleroi, Namur	28·5° C.	(83·3° F.)	,,	,,	1250	,,
Liége . .	29° C.	(84·2° F.)	,,	,,	1040	,,
Central . .	22° C.	(71·6° F.)	,,	,,	710	,,

The experiments further showed that the temperature at the lower part of the coal face exposed to the ventilating current may be below 28° C., whilst the upper part is higher; and, what is very remarkable, that parts of a road leading to cool workings may have a higher temperature than is registered in the workings.

The temperature increases fairly uniformly with the depth, but the degree of humidity decreases, one exception only being observed, viz. in the Charleroi district, where at a depth of 1200 yards a slight rise of humidity was observed.

It was contended by the Belgian Colliery Owners' Federation that in mines in which the temperature exceeds 28° C. (82·4° F.), the limitation of hours of work proposed by the Belgian Government should not apply, unless the humidity reached 85 per cent., which (at 28° C.) would represent a wet-bulb reading of about 26·4° C. (79·5° F.).

The Temperature and Humidity of the Atmosphere in Metalliferous Mines.—As has been previously remarked (p. 2), the subject of ventilation of metalliferous mines has not received anything approaching the attention that has been devoted to that of coal-mines. They have depended for their ventilation on the relatively higher temperatures of the mine air as compared with those of the surface atmosphere to ensure a ventilating current. Consequently, as might be expected, the atmosphere in some deep mines is very oppressive. The most remarkable case of high temperature in a metal mine is perhaps that of the famous Comstock Silver Mines in Nevada, a water temperature as high as 160° F. having been recorded on December 12, 1908, and the water from one of the shafts gave a temperature of 141·8° F. Even now, under a well-devised system of artificial ventilation, a range of very high temperatures and great humidity in the workings are experienced. The ventilation at this mine is secured by means of surface and underground fans.

Owing to excessive water and high temperature, deep mining at the Comstock Mine ceased after 1886, work being confined to the upper levels. Since 1898, however, the drainage of the mine has been in progress and mining is being carried on at 2500 feet from the surface. In one level of the mine (the Sutro-Tunnel level, 1750 feet from the surface), in June–July 1908, the air temperature taken in different parts ranged from 85° F. to 105° F., and the surface temperature from about 65° F. to 87° F. The average day humidity during the months July, August, and September (1907) was 21°. In some parts of the tunnel the air is nearly saturated, the atmosphere taking up moisture from the walls at the rate of 44 grains per square foot per hour.

In summer the air entering the mine during the day is of low humidity, and in winter of moderate humidity.

The Volume and Weight of Air under Variations of Temperature and Pressure.—The weight of air, and of gases generally, is for scientific purposes corrected as to temperature and pressure. This is necessary because the volume occupied by any gas is directly proportional to the temperature, and is inversely proportional to the pressure to which it is subjected.[1] The rate of expansion is practically the same for all gases.

Regnault found that the increase in volume of a given quantity of air at the temperature of melting ice (0° C. or 32° F.), when raised to the temperature of boiling water (100° C. or 212° F.), amounted to ·3665, which gives as the coefficient of expansion—

For the Centigrade scale, $\frac{\cdot 3665}{100} = 0{\cdot}003665$ or $\frac{1}{273}$ of the volume.

For the Fahrenheit scale, $\frac{\cdot 3665}{180} = 0{\cdot}0020361$ or $\frac{1}{491{\cdot}13}$ of the volume.

That is to say, 273 cubic feet of air at 0° C. become 274 at 1° C. and so on, or 491·13 cubic feet of air at 32° F. become 492·13 at 33° F. With the latter scale, however,

[1] (1) If the *volume* be kept constant, the *pressure* will vary directly as the temperature.

∴ If $p.t.$ represent the initial pressure and temperature, and $p_1.t_1.$ the same quantities changed in degree, then $\frac{p_1}{p} = \frac{t_1}{t}$.

(2) If the *pressure* be kept constant, the volume will vary directly as the temperature, that is, if $v.t.$ and $v_1.t_1.$ represent volumes and temperatures $\frac{v^1}{v} = \frac{t^1}{t}$.

(3) If the *temperature* be kept constant, the volume will vary inversely as the pressure, *i.e.* $\frac{v^1}{v} = \frac{p}{p_1}$.

Thus (Fig. 4) let the ordinates pp_1 represent pressures and the abscissæ vv_1 represent corresponding volumes, the temperature $t°$ being constant. Only one curve, the *rectangular hyperbola*, has ordinate X abscissa constant throughout, and that is the form of curve AB. Although always approaching the co-ordinates OC, OD, it only meets them in infinity. By reason of equality of temperature, AB is known as the *isothermal of a perfect gas*, *i.e.* of a gas following Boyle's laws perfectly.

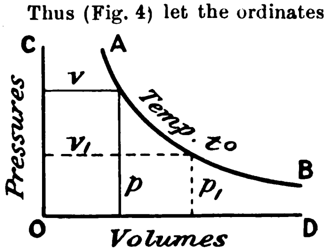

Fig. 4.—Diagram Illustrating the Isothermal of a Perfect Gas.

it is usual to reckon from the zero point (32 degrees lower down the scale than the temperature of melting ice), so that 491·13 cubic feet of air at 32° F. are reduced to 459·13 cubic feet at zero. Hence, starting at the zero points of the scales—although these points are representative of different temperatures—the coefficient of expansion of gases is $\frac{1}{273}$rd for Centigrade and $\frac{1}{459}$th for Fahrenheit scale.

Example.

If 100 cubic centimetres of air are measured at 12° C., what will be the volume at 120° C.?

$$\overset{v}{273} + \overset{t}{12} : \overset{v}{273} + \overset{t_1}{120} :: \overset{v}{100} : \overset{v_1}{x}.$$

$$x = \frac{393 \times 100}{285} = 137 \cdot 8 \text{ cubic centimetres.}$$

Another way, using the Fahrenheit scale, of exemplifying the calculation, would be as follows :—

Let v = volume of air at 32° F. At 212° F. the increase will be—

$$0 \cdot 3665 v \text{ or } \frac{v}{491 \cdot 13}$$

and if t be the temperature on the Fahrenheit scale, the number of degrees t is above the melting-point of ice is $t - 32°$ F.

Therefore for the temperature t the increase of volume

$$= \frac{v(t-32)}{491 \cdot 13}$$

and if V be the total volume—

$$V = v + \frac{v(t-32)}{491 \cdot 13}$$

$$\text{or } V = \frac{v(459 \cdot 13 + t)}{491 \cdot 13} \qquad (1)$$

and supposing V_1 to be the volume of the air at the temperature t_1—

$$\text{then } V_1 = \frac{v(459 \cdot 13 + t_1)}{491 \cdot 13} \qquad (2)$$

or by division (1) and (2)

(see arithmetical proportion)

$$V_1 = V \; \frac{459 \cdot 13 + t_1}{459 \cdot 13 + t}$$

or eliminating the decimals—

$$V_1 = V \frac{459 + t_1}{459 + t}$$

by which formula the volume of air (or other gases) can be determined under any given temperature, if the pressure to which it is subjected remains constant.

The Effect of Pressure on Air and other Gases.—A volume 1 of a gas under a pressure 1 becomes volume 2 under a pressure of $\frac{1}{2}$. Conversely, a volume 1 becomes volume $\frac{1}{2}$ under a pressure 2. It is usual, therefore, for scientific purposes, to reduce all gases to a standard temperature of 0° C. and pressure of 760 millimetres of mercury, or to the standard 32° F. and the pressure, at sea-level, of 29·922 inches of mercury, which is equivalent to 14·696 lbs., and in a water barometer would be represented by a column of 33·9 feet high.

To Calculate the Weight of a Cubic Foot of Dry Air at any Temperature or Pressure.—It has been ascertained by experiment that the weight of a cubic foot of dry air at 0° F. and 1 inch of mercury is ·0028885 lb.

Let W = the weight of a cubic foot of air at temperature t and pressure B.

Taking the volume at 0° F. to be 1, the volume at any temperature, the pressure remaining the same, would be—

$$1 + \frac{t}{459}$$

and the weight per cubic foot being inversely proportional to the change in volume—

$$1 : 1 + \frac{t}{459} :: W : 0{\cdot}0028885$$

$$1 : \frac{t + 459}{459} :: W : 0{\cdot}0028885$$

and multiplying the means and extremes—

$$W \times \frac{t+459}{459} = 1 \times 0 \cdot 0028885$$

$$\therefore W = \frac{459}{t+459} \times \cdot 0028885$$

$$= \frac{1 \cdot 3258}{459+t}$$

so that 1·3258 becomes a constant for all calculations.

$$\text{Hence } W = \frac{1 \cdot 3258 \times B}{459+t}$$

In order to calculate the weight of a cubic foot of any other gas, multiply the weight of the air equivalent by the specific gravity of the gas.

To Find the Weight of a Cubic Foot of Moist Air.—For this purpose a book of hygrometrical tables should be consulted. Those of James Glaisher will be found very reliable. From the tables find the amount of barometric pressure supported by the vapour of saturation—this is equal for all vapours, to the tension of the vapour at the existing temperatures—and subtract this quantity (b) from the barometric pressure B.

$$\text{Thus } B - b = B^1$$

Then find the weight of a cubic foot of dry air at temperature t and pressure $B^1 = W$, and next the weight of a cubic foot of dry air at temperature t and pressure b (*i.e.* pressure borne by the vapour), and multiply this by the specific gravity of vapour (0·623) which will give the weight of the vapour saturating one cubic foot of air $= w$, and

$$W + w = W^1$$

The Process of Breathing.—It is by virtue of a substance called *hæmoglobin*, which it contains, that animal blood acts as the oxygen carrier for the body, the oxygen from the air in the lungs combining—the combination is a somewhat unstable one—chemically with the hæmoglobin, and it is the hæmoglobin that gives to the blood corpuscles their red colour.

The blood of warm-blooded animals is of two states—red or *arterial* and purple or *venous*. The latter is the blood contained in the right side of the heart, which has circulated through the body and given up much of the loosely combined oxygen, and is charged with waste products. The arterial blood is that which, having passed through the lungs, has given up much of the carbonic acid gas which it had acquired during its circulation through the system, and has absorbed the necessary oxygen by means of the hæmoglobin. This will be given up in the journey of the blood through the body, by the process of oxidising various substances, resulting in the production of the carbonic acid gas. Man when at rest, and under normal conditions as to health and environment, makes about sixteen respirations per minute, each equivalent to about 34 cubic inches, so he requires under these conditions 544 cubic inches of air per minute. When at work he will breathe in about three times this amount of air, the result being—

	Per Cent.
Carbonic acid	7·0
Nitrogen	26·3
Unused air	65·7
	100·0

Horses require about thrice the quantity of air breathed by man.

CHAPTER III

THE ATMOSPHERE OF COAL AND OTHER MINES (*continued*): THE ABSORPTION OF OXYGEN IN THE MINE —GASES PRODUCED IN MINES—FIREDAMP

Absorption of Oxygen in the Mine.—Besides men and animals, there are other sources of consumption of the oxygen in the air current in mines, notably the coal itself. Coal occludes gases, and at the same time absorbs oxygen from the air, the rate of absorption being different in respect of different classes of coal. Thus a Northumbrian steam coal does not absorb oxygen to anything like the extent that a South Staffordshire or Warwickshire coal does, the difference in this respect being in all probability due to physical as well as to chemical characteristics. For example, a block of Northumbrian steam coal will withstand weathering influences to a far greater extent than a like sized piece of the South Staffordshire "thick" coal, the former being little changed physically after some years of exposure to atmospheric influences, to which the latter will soon succumb and undergo disintegration. It is also worthy of note that spontaneous combustion is all but unknown in the mines of Northumberland—the writer can only recall one instance, and that this was due to spontaneous ignition is not definitely proven—whereas in the mines working the "thick" coal of South Staffordshire and Warwickshire such fires are of common occurrence.

The old belief that the spontaneous ignition of coal was due *entirely* to the oxidation of the iron pyrites contained therein is a doubtful one. It is probable that the heat, in some measure at any rate, is due to the absorption of oxygen by the coal, the carbon and hydrogen of the latter being attacked by the oxygen of the air.[1]

That the heat increment in coal-mines is, to some extent at least, due to the absorption of oxygen by the coal has been admirably worked out by Dr. Haldane.[2] He found at Hamstead Colliery, in South Staffordshire, that on the average the temperature at the bottom of the shaft is about 11° F. higher than that at the surface, and that ordinarily the range of variation is very small, so that in the 3½ minutes occupied in the passage down the shaft of the air, the latter approximately reaches the temperature of the shaft walls. The difference of temperature as between the surface and bottom of the shaft might possibly be accounted for by the compression of the air, as the heating effect is 5½° F. for every 1000 feet depth (the bottom of the shaft is 1880 feet below the surface), and not be entirely due to the extraction of the heat from the strata.

A rise of 6° F. for every 3000 feet along the main intake was noted, and at about 6000 feet inbye from the shaft bottom the temperature was 71° F., and in the workings commonly about 80° F. to 85° F. It was further observed that in the main return air-ways there was "a slow though steady fall of temperature from the face to the upcast shaft; but the fall was not nearly so marked as the rise in the intake air-ways from the shaft to the face." Considering the various possible causes of heat in the mine, viz. burning of candles, presence of men

[1] Part at least of the oxidation is due to the presence of iron pyrites. Some varieties of FeS_2 are much more easily oxidised than others (*e.g.* that in the Staffordshire coal), so that the *amount* of iron pyrites in the coal is no criterion of its liability to spontaneous combustion.

[2] "Observations on the Relation of Underground Temperature and Spontaneous Fires in the Coal to Oxidation and to the Causes which favour it," by J. S. Haldane and F. G. Meachem, *Trans. Inst. M.E.*, vol. xvi. pp. 457–492.

and horses, heat due to friction by the settling of strata, from friction due to resistance encountered by the air currents, and estimating the amount of heat occasioned by each, Dr. Haldane arrives at the conclusion that the chief factor operating to raise the temperature of the air is the chemical action of the air on the exposed coal and other material. It was found that as the air traversed the mine it steadily lost in oxygen and gained in carbonic acid gas, but that the loss of oxygen was 3·13 times more than the gain in carbonic acid, whereas in most other collieries the ratio of diminution of oxygen to increase of carbonic acid is only as about 1·6. Further, that generally the temperature increment of the mine air rose with the diminution of oxygen. At Tylorstown Colliery in South Wales, by an application of the same reasoning, Dr. Haldane came to the conclusion that the heat production from oxidation is even greater than at Hamstead.

To quote Dr. Haldane[1] again: "It seems clear that as a general rule under present conditions the ventilation of coal-mines has a very considerable heating effect, except on the intake roads. On the working places themselves, the effect may be either cooling or warming, according to the amount of ventilation, as compared with the rate of oxidation. In some collieries the rate of oxidation on the intake roads themselves is so great, or the ventilation so small, that before the air reaches the working face it is heated much above the natural rock temperature.

"When air is passing, month after month, or year after year, down a shaft or along a roadway, the surrounding rock is either cooled or warmed to near the average temperature of the air. The rock round an up-

[1] "The Ventilation Problem in Deep Mines," by J. S. Haldane, M.D., F.R.S., a paper contributed to the *Mining and Engineering Journal of the University of Birmingham*, July 1909.

cast shaft is, for instance, gradually warmed, while that round the downcast is cooled. Any effect on the temperature of an alteration in the ventilation is thus very gradual, and may take months or years to develop fully."

From experiments made at Foxdale Mine (Lead) in the Isle of Man in June 1897, Dr. Haldane thinks[1] it probable that "the vitiation of the main air currents in metalliferous mines and of air in wells is due very much to the same cause as the vitiation (by oxidation) of the air of coal-mines."

Firedamp.—As has been already mentioned (p. 3), the part played by firedamp in colliery explosions was known as early as the year 1675, but we owe our first proper understanding of the nature and properties of marsh gas to Dalton about 1770–80.

All the earlier colliery explosions, viz. those occurring up to the nineteenth century, were probably due to the inflammation of "gas," as coal dust, as will be shown later, was not until later a factor to be reckoned with.

The chief constituent of firedamp, or marsh gas, or simply "gas," as it is sometimes called, is methane or light carburetted hydrogen gas (CH_4), but it is not entirely composed, as is so frequently supposed, of this explosive gas. "Pure" firedamp would be composed entirely of methane, but all analyses that have been carried out have so far shown it to contain small quantities of other gases. The purer the firedamp the more highly explosive will be the mixture which it forms with air.

Composition of Firedamp.—The pit gas or firedamp piped to the surface from a "blower" at Hebburn Colliery, on the banks of the Tyne (1893), had the following composition :—

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[1] "Observations on the Relation of Underground Temperature and Spontaneous Fires in the Coal to Oxidation, and to the Causes which favour it," by J. S. Haldane, M.D., F.R.S., and F. G. Meachem, M.Inst.M.E., *Trans. Inst. M.E.*, vol. xvi. pp. 457–492.

	Per Cent.
Methane (CH_4)	78·8
Nitrogen (N)	18·6
Oxygen (O)	1·7
Carbonic acid (CO_2)	0·9
	100·0

and analyses made at long intervals showed very little variation in respect of the amount of methane present.

As demonstrating the variation in the composition of firedamp, the average of analyses of the gas from six different sources in Austria may be quoted, which gave:[1]—

	Per Cent.
Methane (CH_4)	89·76
Ethane (C_2H_6)	...
Hydrogen (H)	0·23
Carbonic acid (CO_2)	1·98
Nitrogen (N)	7·14
Oxygen (O)	0·39
	99·50

The following analyses of "gas" found in different seams in the North of England were made by Sir H. de la Bêche and Mr. Lyon Playfair in 1846.[2]

TABLE I.—*Analyses of Firedamp in British Mines.*

	Wallsend Pipe.	Wallsend Bensham.	Jarrow Bensham.	Hebburn Bensham.	Jarrow Low Main.	Jarrow ¾ Seam.	24 feet below Bensham Hebburn Colliery.	Gateshead Oakwellgate.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Methane	92·8	77·5	83·1	86·0	93·4	79·7	92·7	98·2
Nitrogen	6·9	26·1	14·2	12·3	4·9	14·3	6·4	1·3
Oxygen	..	..	0·6	..	..	3·0	..	..
Carbonic acid	0·3	1·3	2·1	1·7	1·7	2·0	0·9	0·5
Hydrogen	..	...	..	..	..	3·0	..	..

Dr. Schondorff during the period 1882–84 made a

[1] Report of Austrian Firedamp Commission, *Trans. Inst. M.E.*, vol. iii. p. 531.

[2] *A Treatise on the Winning and Working of Collieries*, by M. Dunn, 1852, 2nd ed., p. 20.

series of analyses of firedamp for the Prussian Firedamp Commission, which gave the following results :—

TABLE II.—*Analyses of Firedamp in German Mines.*

	1.	2.	3.	4.	5.	6.
	Per Cent.	Per Cent.	Per Cent.	Per Cent	Per Cent.	Per Cent.
Methane (CH_4)	90·94	57·41	89·88	83·97	27·95	14·25
Ethane (C_2H_6)	...	...	...	...	...	...
Hydrogen (H)	1·40	5·68	5·84	2·15	1·35	0·90
Carbon dioxide (CO_2)	0·30	1·54	0·67	0·77	0·45	0·10
Carbon monoxide (CO)	...	...	...	...	...	...
Oxygen (O), Nitrogen (N)	7·36	35·37	3·61	13·11	70·25	84·75
	100·00	100·00	100·00	100·00	100·00	100·00

	7.	8.	9.	10.	11.	12.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Methane (CH_4)	7·00	9·58	4·749	84·89	60·46	57·33
Ethane (C_2H_6)	...	...	...	1·62	37·62	0·32
Hydrogen (H)	0·27	0·36	0·088	...	...	...
Carbon dioxide (CO_2)	0·41	0·98	0·134	0·65	2·56	0·12
Carbon monoxide (CO)	...	...	...	...	...	...
Oxygen (O), Nitrogen (N)	92·32	89·08	95·029	12·84	...	42·23
	100·00	100·00	100·00	100·00	100·64	100·00

The following are particulars in respect of the above samples :—

1. Blower in the Bonifacius Colliery at Essen. This gas is burnt at the mine.
2. Gas evolved from water collecting in the sump in the Kreuzgräben Colliery at Tulzbach (Saarbrück).
3. Blower in No. 1 Shaft, Consolidation Colliery, at Schalke. The gas is collected in tubes, and used for lighting the heapstead.
4. Blower in Shamrock Colliery at Herne. The gas is conveyed in pipes to the fire of the ventilating furnace.
5. Firedamp from the Lothringen Colliery at Castrop from a rise drift, in driving which, a few days before, a miner had been suffocated.
6. Firedamp from the Maria Colliery at Höngen, collecting in the roof.

7. Firedamp from the Zollern Colliery at Dortmund, collecting in a rise drift.
8. Firedamp from the Tremonia Colliery at Dortmund, collected in the roof.
9. Firedamp from the New Iserlohn Colliery, Shaft No. 2, at Langendreer, drawn from a rise drift in which there was a natural blower of gas.
10. Blower in the König Colliery at Neunkirchen (Saarbrück). The gas was collected in pipes and led to bank, where it was used by the Commission in the experiments for ignition of coal dust and firedamp.
11. Blower in the Schamburg Collieries at Obernkirchen.
12. Blower from the New Consolidated Friedenshoffnung Colliery at Waldenburg (Lower Silesia).

Dr. Poleck made some analyses at Breslau in 1881 for the same Commission, which, from the high percentage of carbonic acid and comparatively high percentages of ethane that they showed to be present, are of particular interest. The samples of mine atmosphere were taken from a dip drift in the Glückhilf Colliery at Waldenburg :—

	1.	2.
Marsh gas (CH_4)	34·93	32·65
Ethane (C_2H_6)	2·89	3·99
Hydrogen (H)	...	...
Carbon dioxide (CO_2)	41·49 [1]	41·49
Carbon monoxide (CO)	...	1·87 [1]
Oxygen (O), Nitrogen (N)	20·69	20·00
	100·00	100·00

Characteristics of Firedamp.—It is maintained by some that the presence of firedamp is indicated by a

[1] Poleck discovered, by the aid of the spectroscope, that the gases of the Glückhilf Colliery contain carbon monoxide. The considerable quantity of carbon dioxide associated with it points to the sample being the result of decomposition. A heavy combustible gas (specific gravity 1·0538 and 1·0650) occurs in the deepest part of the same district of the colliery, which does not, however, burn quietly, is irrespirable, and rapidly produces vomiting. Vide *Jahresbericht der Schles. Gesellsch. f. Vaterl. Cultur.*, 1882, p. 155, Appendix to Report of the Prussian Firedamp Commission.

faint sweet odour, and from personal observation the present writer inclines to the belief that such sometimes is the case, but methane itself is an inodorous as well as colourless and tasteless gas, which burns with a bluish-yellow non-luminous flame, and in so doing forms carbon dioxide (carbonic acid gas) and water.

Hydrogen is the lightest known gas, and as methane contains nearly 25 per cent. by weight of this gas, it is much lighter than air, for the specific gravity of air being unity, that of methane is 0·55314, so that firedamp in the mine rises upward to the roof and occurs in the higher parts of the workings, floating on the air until diffused in the atmosphere. When undiluted with air, methane extinguishes flame.

The effect of firedamp on the human system, when present in such quantity as to be inflammable, has been graphically described by Mr. P. S. Reid.[1] He mentions an instance at Pelton Colliery in the county of Durham (1847), where firedamp was issuing from a point in the mine in such quantity that "if you advanced beyond a certain distance into it, say 4 or 5 yards, your lamp went out, and you were immediately affected with a strange swimming in the head, and on attempting to speak you were conscious of your doing so, but could not hear your own voice if you still persevered, or remained standing. Your next sensation was a trembling of the knees, and if you did not attend to this, giddiness ensued, and you fell down insensible." He remarks that he has "frequently seen men brought out in this condition, who, on recovering their senses in the fresh air, were seized with vomiting and nausea which lasted several minutes."

In this connection the opinion of Dr. Haldane is

[1] *Trans. North of England Inst. Mining and Mechanical Engineers*, vol. iii. pp. 38–39.

valuable, according to whom the firedamp acts merely as a diluent to the oxygen in the air, and that an atmosphere containing from 50 to 60 per cent. of methane may be breathed for a time without harm. Pure methane is not therefore poisonous, and the injurious effects described by Mr. Reid were either those due to deficiency of oxygen, or to the presence in the firedamp of an impurity, very probably sulphuretted hydrogen, which is very poisonous. Paul Bert[1] has pointed out that the physiological effects of firedamp are very similar to those produced by air from which a proportion of its oxygen has been removed.

Inflammability of Firedamp.—When an atmosphere of air, under normal conditions as to humidity, contains 6·1 per cent. of methane, the mixture is just ignitible, so that 6·1 per cent. is spoken of as the limit of inflammability. But the most explosive mixture of atmospheric air and methane is that which is in the proportions of 9·55 volumes of dry air to 1 volume of firedamp, that is, when methane is present in the air to the extent of 9·47 per cent., or just sufficient to consume the whole of the oxygen in the air. Below and above that figure the explosive force diminishes. The scale of inflammability may be stated thus :—

Mixture. Proportionate Volume of Methane.		Air.		Inflammability.
1	in	30	3·4 per cent. methane.	Affects the lamp, but is not explosive.
1	in	15	6·8 ,, ,,	Mildly explosive.
1	in	13	7·7 ,, ,,	Mildly explosive; whole mixture converted into flame.
1	in	10	10·0 ,, ,,	Very explosive.
1	in	9	11·0 ,, ,,	Very explosive.

[1] French Firedamp Commission.

A further increase of the methane renders the mixture less and less explosive.

The high explosive range is with mixtures containing 8 to 14 per cent. of methane; when present to the extent of 22 per cent. the mixture is not explosive.

The complete ignition of methane may be chemically expressed by the equation—

$$CH_4 + 2O_2 + 16N = CO_2 + 2H_2O + 16N.$$

The temperature at which methane will ignite is 1202° F. or 650° C.,[1] and the calculated temperature of combustion is 3902° F. A mixture, therefore, of 9·55 volumes of dry air to 1 volume of methane will contain the exact proportion of oxygen necessary to effect complete combustion of the methane, hence the reason for its being the most explosive mixture, that is, the one in which the flame would be expected to travel with the greatest rapidity. But in the case of an underground explosion owing to the confining influence of the mine roadways and the expansion of that part of the mixture through which the flame has passed, the still unconsumed part is probably driven in front of the explosive wave and perfect combustion does not result.

The pressure due to the combustion of the above mixture, when ignited in a closed space, has been shown to be 102·6 lbs. per square inch, which affords an explanation for the immense havoc which may be wrought by an explosion of firedamp in a mine even without the aid of coal dust.

Occurrence of Firedamp in Coal-Mines.—Though the occurrence of firedamp is not restricted to collieries, it is chiefly in this class of mine that the

[1] MM. Mallard and Le Chatelier, *Annales des Mines*, 1883.

quantity in which it exists renders its occurrence a matter of importance. It exudes from the pores and cavities of the coal, and to some extent from the surrounding strata, where it is pent up in a state of greater or less pressure. Firedamp is in all probability given off in every coal-mine, the reason that some collieries are described as exempt in this respect being that it is generated in such small quantities as never to have been detected on the flame of candle or lamp. The rate at which firedamp is given off, however, varies very greatly as between districts, collieries, and seams of coal. The physical or chemical character of the coal has so far proved of little or no guide in this respect; for instance, the anthracites of South Wales are not remarkable for the quantity of gas given off in the mines, and the steam coals of Northumberland are comparatively free from firedamp, yet the anthracite region of the United States of America is noted for its explosions, and the steam coals of South Wales are very "gassy." The "thick" coal of South Staffordshire does not usually give off much firedamp, but collieries working the "household" coal seams near the sea-coast of Durham are all "safety-lamp" pits. The existence of "gas" is usually, though this cannot be accepted as a rule, a question of depth from the surface. Speaking generally, the deeper the seams are from the surface, the greater the quantity of firedamp present in the mine. Doubtless, also, the nature of the over-burden has an influence in this respect. Thus it is found in those Natal collieries, the surface of which is covered by thick sheets of diorite, that the danger from firedamp is more insistent than in those in which the over-burden is entirely composed of the more permeable sandstones and shales.

Firedamp may be evolved from the strata (not necessarily coal alone) in three ways:—

1. Normally, as fresh faces are exposed the gas exuding from the pores of the coal or other strata in which it exists at varying pressure.
2. As sudden outbursts.
3. As blowers.

1. *Pressure of Firedamp in Coal Seams.*—Sometimes, especially if the coal face be wet, the firedamp can be heard issuing from the coal, but as often as not this is not the case. Nor is the occasional hissing

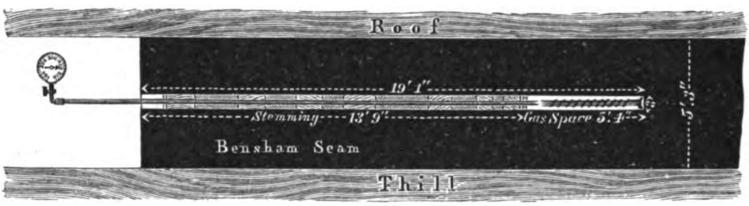


FIG. 5.—Showing Method of Determining the Pressure of Firedamp in Coal Seams.

(From *Trans. N. of Eng. Inst. Mining Engineers*, vol. xxx. p. 196.)

of the gas at the face necessarily indicative of pressure; frequently the gas when enclosed in the seam at greatest pressure is liberated with least sound.

With a view to determining at what pressure firedamp exists in the coal, Mr. (now Sir) Lindsay Wood carried out a series of experiments in 1880[1] for the Royal Commission on Accidents in Mines (appointed 1879). With this end in view, holes were bored into the coal at the face at several collieries in the county of Durham, and a pressure gauge inserted. This being carefully sealed up, the pressure, which gradually rose, was noted (see Fig. 5). The depths of the seams experimented on from the surface

[1] "Experiments showing the Pressure of Gas in the Solid Coal," by Lindsay Wood, *Trans. North of England Inst. Mining and Mechanical Engineers*, vol. xxx. p. 163.

were from 750 to 1268 feet, and the longest bore-hole in the coal was 47 feet, the shortest being 3½ feet. Pressures varying from 200 to 461 lbs. per square inch were obtained, the highest pressure being obtained on the third day of an experiment at Boldon Colliery. The greatest volume of gas given off from the holes was equivalent to 5927 cubic feet per hour per square foot of surface, and was obtained at Eppleton Colliery. Though it must be borne in mind that these results were obtained at "gassy" collieries, at the same time they show how small an effect normal variations in barometric pressure can have on the issue of gas from the strata. The firedamp which shows in a mine during periods of low barometer is that which exists in the waste, and in the cracks and crevices of the strata, not that, or only to a very small extent, contained in the strata itself. The former may, however, prove a more serious feature in point of safety conditions than the latter.

A peculiar feature was many years ago observed by the present writer at a colliery in the North of England at which he was stationed, viz. that the firedamp from a certain waste used to "show" for some distance back from the goaf in an old road connected therewith some time before a fall in the barometer indicated a decrease in the atmospheric pressure. The late Mr. John Buddle was aware of the same phenomenon as long ago as the year 1830, for he writes: "We observe . . . that the discharge of the gas generally precedes the fall of the barometer by a brief interval, probably owing to the more delicate nature of the balance."[1]

2. *Sudden Outbursts of Gas.*—Sudden outbursts of gas are not of frequent occurrence in the coalfields of

[1] "An Account of the Explosion which took place in Jarrow Colliery," by John Buddle, *Trans. Nat. Hist. Soc. of Northumberland and Durham*, vol. i. p. 187.

Great Britain, and, as in the case of blowers, the possibility of their occurrence is seldom heralded by indicative circumstances. They are of great danger, as before precautions can be taken a great portion of the mine atmosphere may be converted into an explosive mixture. In Belgium sudden outbursts of gas under the appellation *degagements instantanés* are a well-recognised phenomenon. Miners in Great Britain term these sudden outbursts "bags of gas," or "bags of sudden foulness." Thus, writing in the year 1830, that great miner, the late Mr. John Buddle, remarks:[1] "The explosion of the No. 1 Pit, Washington Colliery, on the 20th of November 1828, was occasioned by 'a bag of foulness' breaking down from the roof of the air-course board, behind the workmen, which, at the same time, stopped the ventilation by the fall of roof it occasioned. So that a double cause operated in this instance to occasion the explosion, the stoppage of the ventilation, and an enormous discharge of inflammable air occurring at the same instant." The Jarrow Colliery explosion (31st August 1830) was also due to a sudden outburst of firedamp from a cavity in the coal. This instance has points of peculiar interest, the particulars of which have been admirably recorded by Mr. Buddle.[2] He says that the drift, at the face of which the outburst occurred, was 9 feet wide and 5 feet high, and the whole block of coal across the face seemed to have been forced forward with great violence, there being a jagged aperture 9 to 12 inches in width along the roof and down the left side. The block was 4 feet thick, "and a space of 7 feet wide behind it extended to a downcast hitch of 3½ feet. This space

[1] Mr. John Buddle in *Trans. Nat. Hist. Soc. of Northumberland and Durham*, vol. i. p. 187.

[2] *Ibid.*, p. 197.

was filled with (*danty*) disintegrated coal of a sooty appearance" (see Fig. 6). Mr. Buddle adds: "When I first examined the place no inflammable air was discernible, nor has there been the least appearance of any since. From this it would appear that there was an isolated bag of gas, without communication with any blower-threads whatever, and that it quite exhausted itself by a single eruption."

Very shortly afterwards two similar occurrences took

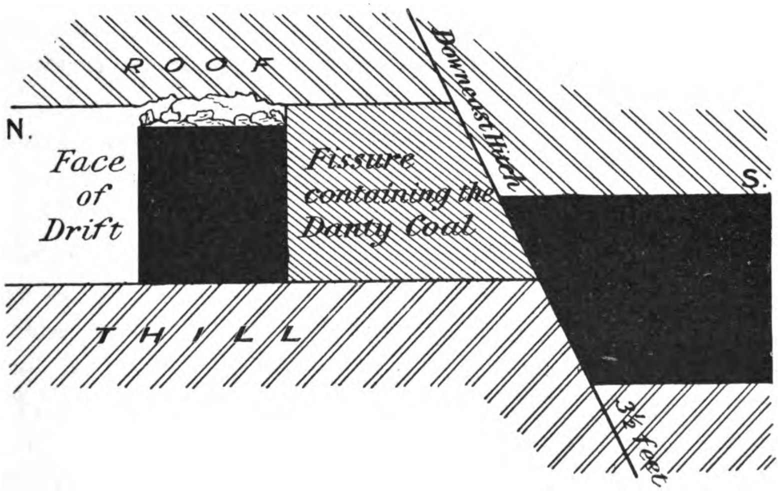


FIG. 6.—Section at Point of Occurrence of a Sudden Outburst of Firedamp at Jarrow Colliery in 1830.

place at the colliery, except that they were of less magnitude, and nothing serious resulted therefrom. Both the succeeding outbursts occurred at small "hitches" in the seam, accompanied by linings of "danty" (disintegrated coal), which led Mr. Buddle to remark: "From which it is to be presumed that the law of this part of the mine is to discharge its gas from those reservoirs or fissures of disintegrated coal, as very little gas is discharged from the pores of the good coal. It would seem as if those fissures of disintegrated coal formed runners, channels, or outlets for the gas which has been evolved from the adjoining coal." An explanation of great practical importance, and one which goes far, in the opinion of the present writer, to explain the *raison d'être* of sudden outbursts of coal in other parts of the world, notably those which have recently occurred on such a grand scale in Austria-Hungary.

Blowers of Gas.—Blowers of gas, or *soufflards*, as they are termed in France, differ from "bags of gas," in that the stream of gas continues to issue at a constant pressure for a long period of time, in some cases for many years, and the projection of a mass of coal is not necessarily an accompaniment of their occurrence. Instead, therefore, of their being due, as in the case of "bags of gas," to the storage of firedamp in a cavity or in a mass of spongy disintegrated coal, and bursting forth on the reduction in the strength of the retaining envelope of coal or other strata, the gas issues from a well-defined channel, which is constantly fed with gas at a more or less constant pressure. This feature, and the fact that blowers are frequently accompanied by the discharge of water, would seem to point to hydraulic pressure being a possible explanation for their occurrence and constancy, but for the fact that a water discharge does not always accompany a blower of gas, nor does the disposition of the strata always lend itself to the theory.[1] Probably the explanation is to be found in several directions; the

[1] Messrs. Pernolet and Aguillon in their Report upon the Working and Regulation of Fiery Mines in England, p. 51, state that "it is frequently found that when a blower comes from a well-defined crack or fissure, large quantities of water are also met with," their view of the blower being that "the fracture through which it is discharged, whether a simple fissure or a fault, pre-existed, and this gas is discharged when the drifts reach it; whilst in the sudden outbursts there is no pre-existing fracture, but one which takes place suddenly, and produces the discharge."

hydraulic theory accounting for the existence of some blowers, and the settlement of strata for others, and the fact of a number of ramifications—"threads" to use Mr. Buddle's expression—feeding into one common main channel, though in the latter case one would expect to find a constantly decreasing pressure in the issuing gas.

In an interesting description of a spasmodic "blower" which occurred at Pelton Colliery, in the county of Durham, in the year 1847, which gave off bags of gas at intervals, Mr. Reid[1] put forward the hydraulic theory as a possible explanation. His reasoning is interesting. At this colliery there was a fault of a throw of 60 feet dislocating the strata, and consequently there must have been many crevices and fissures branching off from the main line of weakness, all of which would act as service pipes "from some head of water, perhaps at a very great distance"; and he considered that "until the moment of cutting the feeder . . . this column of water held back the gas in a highly compressed state, and that as soon as it was relieved of the pressure it issued into the mine at the same or even greater pressure than that of the water itself." He illustrates his theory by means of a diagram, a reproduction of which is given in Fig. 7. On one occasion there was an eruption of gas from this blower in the form of a "bag" of gas amounting to 47,044 cubic feet in about one minute. An interesting feature of the Pelton blower was the coldness of the issuing gas and water. Although the temperature was not definitely recorded, the gas felt, at the point of issue, says Mr. Reid, "as if a blast of excessively cold wind was blowing against you." The sudden expansion of gas pent up at high pressure

[1] *Trans. of the North of England Inst. of Mining and Mechanical Engineers*, vol. iii. pp. 30–34.

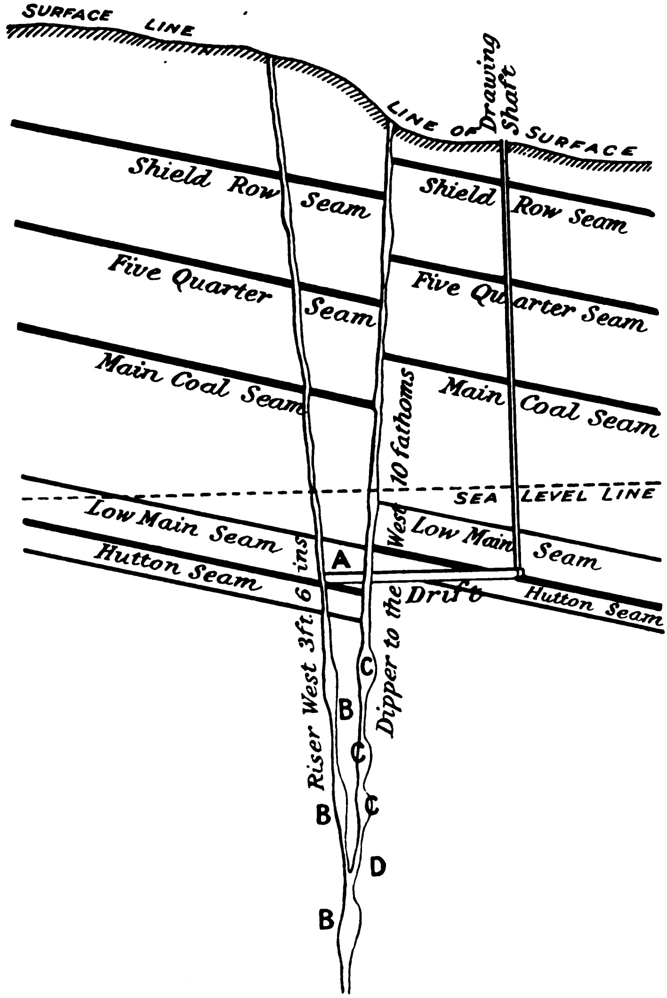


FIG. 7.—Sketch to Illustrate the supposed relative Positions of Water and Gas at a "Blower" in a Drift at Pelton Colliery in 1847.

Reference.—A, position in which the blower was met with in the drift, April 1847. B, B, B, fissures supposed to be filled with water communicating with a much superior height by means of disruption of strata caused by a small fault communicating with the main fault at D, and by its pressure holding back gas at great tension in the cavities or fissures of the main fault at C, C, C, and which flowed out with great violence at the blower A when cut in the stone drift so soon as the water abated.

would, of course, produce great cold. It is worthy of notice that very shortly after the final diminution of water the eruptions of gas ceased. It is not clear to the present writer whether this so-called "blower" should not rather be regarded as a series of separate eruptions of gas in the nature of "bags of gas."

Occurrence of Firedamp in Metalliferous Mines.—In the year 1886 two explosions of firedamp occurred in the Mill Close Head Mine at Darley Dale in Derbyshire, in one of which five men were killed. At this mine the galena occurs in the Carboniferous Limestone series, the Yoredale shales of which occasionally give off a little firedamp,[1] and records show that men have been killed by firedamp explosions as far back as 1732 at the lead mines in the district.

A blower of firedamp has occurred from time to time in the famous Van Lead Mine near Llanidloes in Montgomeryshire, the vein of which traverses rocks of Lower Silurian age. Water generally accompanies the emission of the gas. It is difficult to account for the presence of firedamp at this mine, as there are no carboniferous rocks in the neighbourhood, and decaying timber could not account for the existence of a blower of gas. Possibly it owes its origin to the decay of plant or animal life of the Silurian period, and has been pent up through the succeeding ages, or it may be due to the chemical action of acidulated waters on mineral substances. It is a peculiar feature of the emissions at this mine that they are accompanied by sulphuretted hydrogen.

The occurrence of firedamp in the hæmatite iron mines of Cumberland and North Lancashire is by no means rare, and is undoubtedly due to the decay of the timber, little

[1] There is a band of coal one inch in thickness between the shales and limestone at this mine. Safety lamps are still in use at the mine.

of which is recoverable, and so decays in the wastes. The conditions are favourable to its generation, the encasing débris being soft and wet. It has frequently been observed, too, that in those parts of the mine where gas is detected the temperature is greater than in those where it is free from firedamp, and such temperature is much above that which can be accounted for by depth from the surface, it being undoubtedly due to fermentation in the timber. Gas has been detected issuing from the end of a decaying prop in quantity sufficient to allow of ignition.

Firedamp is well known also in the clay ironstone mines of Cleveland, Yorkshire (Lias), though it seldom exists in dangerous quantities; and it has been found in the diamond mines of South Africa.

The occurrence of gas in metalliferous mines forms the subject of an interesting paper which the late Mr. Bennett H. Brough contributed to the *Transactions of the North of England Institute of Mining Engineers*, which may be consulted for further instances of the occurrence of firedamp in metalliferous mines in Europe.[1]

[1] *Trans. of the North of England Inst. of Mining Engineers*, vol. xxxviii. pp. 59–70.

CHAPTER IV

GASES PRODUCED IN MINES (*continued*): CARBON DIOXIDE — CARBON MONOXIDE — AFTER-DAMP — SULPHURETTED HYDROGEN—NITROUS OXIDE

Carbonic Acid Gas or Carbon Dioxide: Occurrence in the Mine (CO_2).—It is doubtful whether absolutely pure carbonic acid gas is generated or exists in mines, though it is a constituent of "black-damp" (*stythe* or *choke-damp*), "firedamp" (to a small extent), and of "after-damp."

"Black-damp," according to Dr. Haldane,[1] consists of about 87 per cent. of nitrogen and 13 per cent. of carbon dioxide, but probably the relative proportions of the two gases constituting the mixture are variable. Carbonic acid gas is found also, usually in small quantities only, as a component of firedamp (see p. 33), and experienced miners can determine its presence in the firedamp by the colour of the "cap" on the lamp; thus pure methane gives a faint bluish-yellow cap, whereas if carbonic acid gas be present in the firedamp the cap has a brownish tinge. Instances have occurred when the presence of carbonic acid gas has been present in such quantity in the firedamp as to render the latter inexplosive. The writer himself on one occasion noted this phenomenon. Carbonic acid gas or carbon dioxide occurs in after-damp as

[1] "The Causes of Death in Colliery Explosions," by John Haldane, M.D. Report to Secretary of State, 1896, p. 16.

one of the products of the inflammation of firedamp or coal dust, the amount being governed by the completeness of the combustion of these inflammable substances, the more perfect the combustion the higher the percentage of CO_2 in the resultant after-damp—carbonic oxide (carbon monoxide) being the result of incomplete combustion—or in other words carbon dioxide is always found when carbon is burnt in excess of air.

The following analysis of after-damp resulting from the explosion of firedamp in air may be given :—

	Per Cent.
Nitrogen	71·2
Carbonic acid gas	9·6
Steam	19·2
	100·0

Chemical and Physical Properties of Carbon Dioxide.—Carbonic acid is one of the two compounds which carbon forms with oxygen, which are carbon monoxide, or CO, and carbon dioxide, or CO_2.

Carbon dioxide is a colourless gas, inodorous, but possessing a slightly acid taste. It has a specific gravity of 1·529, air being taken as unity. Hence in the mine black-damp or stythe is first discovered near the floor of the mine, and only rises to the roof when present in such quantities as to fill the roadways. It is soluble in water, but if the water is boiled the gas is expelled. As to the extent of its solubility 1 volume of water at 0° C. will dissolve 1·797 volumes of gas; and the higher the temperature of the water the less the quantity of gas that will be dissolved, but the greater the pressure the greater the quantity of gas absorbed. Carbon dioxide does not support the combustion of most substances, and is injurious to life both as a diluent of the oxygen in the air and as a poison, but it rarely exists in the mine in such quantities as to act in the latter capacity. According to Professor Clowes,[1] it requires the presence of 15 per cent. of carbonic acid gas in the atmosphere to extinguish lights. Dr. Haldane[2] states that when present in great excess it is a distinctly poisonous gas, and that it "does not act, as sometimes stated, by merely diluting the oxygen of the air. Air mixed with 50 per cent. of carbon dioxide causes rapid death, while air mixed with an equal proportion of nitrogen has very little effect." He finds that the first distinctive "effects on the human system are felt with about 3 or 4 per cent.; the respirations increase with the percentage, both in frequency and depth, until with about 6 per cent. there is distinct panting." At 7 or 8 per cent. the panting is very distressing; at 10 or 11 per cent. "the respiratory distress is extreme. With a somewhat higher percentage an anæsthetic action seems to occur, so that consciousness is benumbed or entirely lost, although life may not be endangered at least for many hours, judging from experiments on animals."

The quantity of carbon dioxide present in the atmosphere at the surface amounts, as has been shown (p. 15), to about 3 volumes in 10,000 of air, which quantity is nearly constant.

The Sources from which Carbon Dioxide is Derived.—This gas is produced by the burning of lamps in the mine, the breathing of men and animals, from the firing of explosives, the decay of timber, and is yielded naturally from the coal and the strata enveloping the coal.

It has been frequently observed that those mines,

[1] *Trans. Inst. M.E.*, vol. vii. p. 420.

[2] "The Causes of Death in Colliery Explosions." Report to Secretary of State by John Haldane, M.D., p. 15.

the coal or enveloping strata of which generate carbonic acid gas, seldom give off firedamp—that is, to any extent—a circumstance which is not easy of explanation. Before, however, any deductions could be drawn from this statement, it would be necessary to more firmly establish the fact by carefully prepared statistics.

Sudden Outbursts of Carbon Dioxide in the Central Plateau of France.—Outbursts of carbonic acid gas, or at any rate of gases in which carbon dioxide is the predominant gas, are not unknown, those which occurred at Rochebelle Collieries[1] in France in 1879 being very remarkable. The mine workings comprised three groups, viz. Rochebelle, Cendras, and Fontanes, in all of which carbon dioxide is present. It is supposed that the gas did not originate *in situ*, but to be the result of volcanic action. At Fontanes there is a schistose formation, which is less fissured than the compact sandstones at Rochebelle; hence, whereas at Rochebelle the gas is contained in fissures, at Fontanes it is enclosed in the coal, at Rochebelle also it gradually diminished being replaced by firedamp, which was not observed until 1886. The outbursts at Fontanes were of so sudden a character as to have the force of explosions, throwing down vast quantities of coal, sometimes amounting to hundreds of tons, and occasioning in some instances a serious loss of life. At Rochebelle the gas issued from the fissures, which were full of water and carbonic acid, and the outbursts were delivered with much less force than those at Fontanes. The following symptoms were noticed in advance of every outburst in all the cases:—

[1] "Les Dégagements instantanés d'Acide Carbonique aux Mines de Rochebelle," by G. Hanarte, *Annuaire de l'Association des Ingénieurs sortis de l'École de Liège*, vol. vi. 1887, pp. 1–14; "Les Dégagements d'Acide Carbonique aux Mines de Rochebelle," by C. Lange, *Bulletin de la Société de l'Industrie Minérale*, 1892 series, vol. vi.

heaviness of the atmosphere, dull explosions, pressure in bore-holes, detachment of plates of coal, and decrepitation and change in the physical character of the coal. The force of the outbursts increased with the depth. It would almost appear as if these eruptions of carbon dioxide were in a large measure due to hydraulic pressure. It is of interest to note, in connection with what has been said on p. 42 respecting the presence of disintegrated coal in the case of sudden outbursts of firedamp, that the same characteristic existed in the case of these sudden outbursts of carbonic acid gas; the two phenomena are very similar, and it may be supposed that the gas is absorbed and condensed by the powdered coal, and not under pressure, when this coal is under static equilibrium; the pressure becoming apparent after only a slight shock, causing the gas to leave the coal. According to Professor Graham, who made many experiments on the special attraction of gases for solid bodies, charcoal is able to absorb about ninety times its own volume of carbonic acid gas. Blowers of carbonic acid gas and of firedamp are frequently found in the neighbourhood of faults where the coal has been finely crushed, and where the gas is unable to escape, and the resulting absorption of the gas by the coal dust may, in some cases, explain the enormous volumes of gas given off, notwithstanding its feeble pressure in the coal. It is a notable fact that bore-holes were useless for the purpose of draining off the gas from the coal at Rochebelle.

The "Du Grosmenil" Colliery in the Brassac basin (Departments of Puy-de-Dôme and Haute-Loire) is subject to outbursts of carbon dioxide of great violence, equal in intensity to those of Fontanes. The coal deposits, also, that traverse the central plateau from north to south between Decize and Champagnac are also characterised by similar eructations, as also are the collieries in the Department of Gard (Rochebelle and Nord d'Alais and other collieries). On July 6, 1907, an outburst occurred in the Nord d'Alais district of a peculiarly violent character, the entire workings being filled for several hours with carbon dioxide—cases of poisoning, resulting in the death of three persons, occurring *at the surface*, within a radius of several hundred yards from the pit. Another notable outburst occurred at the Singles Colliery (Puy-de-Dôme) on July 26, 1909, causing the death of five out of the ten persons below-ground at the time.[1]

The black-damp, or stythe, which makes its appearance in the workings of the mine on the occasion of a "low glass," is that which is pent up in the wastes or goaves of the mine. Old miners speak of the "damp" as appearing when the wind is from the south, but the direction of the wind has nothing to do with its presence. When the wind blows from the south it comes from a rainy quarter, and heralds a fall in the barometer and a moist atmosphere, and moist air is lighter than dry air; hence the appearance of the stythe.

In those mines which are subject to spontaneous combustion, due to the absorption of oxygen by the coal, carbon dioxide is extensively produced as well as carbon monoxide.

Carbon Monoxide (CO).—Carbon monoxide, carbonic oxide gas, or white-damp, when it exists under normal mining conditions, is present only in minute quantities, and is due to the exploding of gunpowder or

[1] "Les Dégagements instantanés d'Acide Carbonique dans les Mines du Plateau Central Français," and "Du Rôle joué par une Chambre de Sauvetage dans un Accident survenu à la Mine de Singles le 26 Juillet 1909." Papers read by J. Loiret (Ingénieur au Corps des Mines à Clermont-Ferrand) at the International Mining Congress held at Düsseldorf, 1910.

other explosives. It occurs also in the after-damp resulting from the incomplete combustion of firedamp or coal dust, and is also present in the smoke and "gob-stink" which are produced by the spontaneous combustion of coal.

Chemical and Physical Properties.—As the result of the incomplete combustion of carbon, carbonic oxide in its pure state may be prepared by passing a slow current of carbon dioxide over pieces of charcoal heated to redness in a tube, thus—

$$CO_2 + C = 2CO$$

It is a colourless and tasteless gas, but it has a peculiar odour. It burns with a lambent blue flame to form carbonic acid gas. It is an unstable gas, as it soon passes into carbonic acid gas. Its specific gravity is 0·969, so that it is a little lighter than air. It is a strong poison, and as its effect is cumulative, an atmosphere containing only a very small portion, if inhaled for a sufficient period, results in death. The destructive effect of carbon monoxide is due to its action on the hæmoglobin of the blood, with which it chemically combines, destroying life by robbing the body of the oxygen which is normally carried by the hæmoglobin; for hæmoglobin, when saturated with carbon monoxide, cannot take up oxygen, so that death results. Dr. Haldane[1] has clearly shown that carbon monoxide has no other effects than those caused by interference with the oxygen supply to the tissues, and he puts the affinity of carbon monoxide for hæmoglobin at about 250 times that of oxygen. So that if an atmosphere containing only 0·1 per cent. of carbon monoxide is inhaled for a sufficient length of time, the hæmoglobin of the blood will become about equally saturated with carbon monoxide and oxygen, and complete

[1] Papers by J. S. Haldane, F.R.S., in the *Journal of Physiology*, vol. xviii. (1895), pp. 200, 430, and 463.

helplessness will not ensue; but with 0·2 per cent. the blood becomes about 67 per cent. saturated, "and," says Dr. Haldane, "complete helplessness, with loss of consciousness, would doubtless occur;" a fact of the utmost importance, in view of the precautions necessary in the rescue operations after a colliery explosion.

Indications of the Presence of Carbon Monoxide.—It is easy to determine whether persons have been killed by carbon monoxide poisoning from the pink colour of the tissues and organs, which is due to the fact that the action of carbon monoxide on the blood is to render it more highly coloured: blood saturated with carbon monoxide being bright scarlet, like arterial blood. Dr. Haldane has made use of this colouring effect on the blood to devise, by means of a colourmetric test, an easy, rapid, and fairly accurate mode of determining the percentage of saturation, which has only one disadvantage—that it cannot be carried out in artificial light. A mouse or some other small animal is exposed in the deleterious atmosphere for, say, ten minutes, and then drowned, if not already killed by the after-damp, and a drop of its blood having been diluted with 100 times its volume of water, is compared with a drop of normal blood similarly diluted. Part of the latter solution is then saturated with coal gas (coal gas contains about 5 per cent. of CO), and the three solutions poured into narrow test tubes of equal diameter for purposes of comparison. Care must be taken to preserve the same depth of colour of the three solutions by the addition of water or blood. When this is obtained the *tints* of the three solutions can be compared, "when, according to the percentage saturation of the sample of blood under examination, the tint of the first solution will approach to that of the normal blood, or the blood saturated with coal gas (*i.e.* with carbon monoxide), and a rough estimate may be made of the percentage saturations."[1] The diluted normal blood will have a pink-orange tinge, the diluted blood from the body will be pink, and the normal blood saturated with carbon monoxide will be coloured carmine. By the colour test as small a quantity in the air as 0·01 per cent. of carbon monoxide can, according to Dr. Haldane, be detected and roughly estimated.

Carbon monoxide, if present in the air to an extent greater than about 1 per cent., can be detected by the cap it shows on the ordinary flame of a safety lamp, but neither men nor animals could long exist in such an atmosphere. So, as a practical indicator, the lamp test may be disregarded. The present writer has seen men so badly overcome by the effects of carbon monoxide as to remain ill for some days, when the lamp gave no indication of its presence in the atmosphere in which they had been, either by a "cap" or other effect on the flame (see also p. 59). And Mr. John Buddle noted in his description of the Wallsend Colliery explosion (1835) that "although the Davys burnt very well, the after-damp was so strong that it overpowered them, and but for the reserve . . . some of them would have been lost."[2]

The best indicator of the presence of carbon monoxide in the atmosphere in dangerous quantity is a small animal,

[1] "The Causes of Death in Colliery Explosions and Underground Fires," a Report to Secretary of State, by John Haldane, M.D., 1896, pp. 19, 20. This excellent Report should be closely studied by all mine managers, as it constitutes a compendium of chemical and medical information on the subject treated.

The subject of carbonic oxide poisoning in collieries occupied the attention of medical men in the North of England as early as 1866. See *Proceedings of the Northumberland and Durham Medical Society*, session 1866-67, pp. 28 and 35, when Dr. William Murray correctly diagnosed the effects of CO poisoning, and a few years later carried out, in conjunction with Professor Freire-Marreco, experiments on small animals, subjecting them to the action of carbonic acid and carbon monoxide.

[2] *Trans. Nat. Hist. Soc. of Northumberland and Durham*, vol. ii. p. 359.

such as a mouse or small bird, which, carried in a cage, will give ample warning to those engaged in rescue work after a colliery explosion by reason of its condition, as it takes about twenty times as long for a man to be distinctly affected by the gas as a small animal. Of the two animals, however, a small bird the present writer has found to be the most suitable, as when it falls from its perch it is time for the person carrying the cage and those with him to return into fresh air.

After-damp.—The nature of after-damp is the subject of much notice in the Report of the Committee of the South Shields Society for the Prevention of Accidents in Mines, which was appointed in 1839. "Very different indeed was the system pursued by the Committee to that followed by certain theorists who have from time to time given to the public an accumulation of unphilosophic and impracticable suggestions," wrote Mr. Mathias Dunn.[1] Condensing the views of the Committee on the subject of after-damp, he continued: "The explosion leaves, in its results, the after-damp, which consists of more than two-thirds of nitrogen, and under no circumstances more than one-tenth of carbonic acid gas. The after-damp may then be said to be formed of "8 of nitrogen, 2 of aqueous vapour, and 1 of carbonic acid gas." The Committee seem to have been quite unaware of the presence of carbon monoxide in the after-damp. Though its effects are described, they are attributed to the existence of nitrogen, thus: "Where carbonic acid gas prevails, the lamps are instantaneously extinguished, which of itself is abundant warning for the miner to retire, as the air must be in an

[1] *The Winning and Working of Collieries*, by M. Dunn, 2nd ed., pp. 227–229. Mr. Dunn writes as though he were quoting in full from the Report of the Committee, but a reference to that Report shows that he has very accurately condensed their results, *vide* pp. 67, 68 of the Report published by Longmans, Green & Co., 1868.

unfit state to support animal life. But where there is a large proportion of nitrogen, the lamp continues to burn as in sulphuretted hydrogen, even when the miner is struck down, life being in the latter case extinguished before flame.

"The above opinions of the Committee were corroborated by Dr. Pereira, Professor Graham, and Professors Hope and Christison of Edinburgh, all of whom concurred in the opinion that the deleterious nature of after-damp does not arise from the presence of carbonic acid gas alone."

More recent investigation has shown that the effects attributed to nitrogen are due to carbon monoxide.

After-damp is composed of carbonic acid, carbon monoxide, nitrogen, unconsumed oxygen, sometimes unconsumed firedamp, and vapour, but in what proportions it is difficult to say, as of course the procuring of samples of undiluted after-damp is difficult to effect. In all probability, also, the proportions vary in different instances, being governed by three factors: (1) The extent to which the contributing inflammatory substances were present at the time of the explosion or fire; (2) their composition, firedamp and coal dust varying largely in their composition; and (3) the intensity of the inflammation.

Dr. Bedson, from an examination of a sample of gas taken by Mr. J. B. Atkinson (H.M. Inspector of Mines) at Usworth Colliery, after the explosion in 1885, found it to be composed of—

	Per Cent.
Carbon dioxide (CO_2)	4·54
Carbon monoxide (CO)	2·48
Methane (CH_4)	8·68
Oxygen (O)	7·23
Nitrogen (N)	76·80
	99·73

It must be mentioned, however, in respect of this analysis, that the sample was taken from behind a stopping which had been erected to cut off the air from a fire caused by the explosion. The atmosphere might not, therefore, be exactly representative of the after-damp existing in a mine immediately after an explosion, the sample being taken four days after the last stopping had been built.

From analyses of samples of after-damp taken by Mr. W. J. Orsman after ignitions of the coal dusts in the experiments carried out by Mr. Hall (H.M. Inspector of Mines) in 1892, it was shown that these were composed of—

	Per Cent.
Oxygen	3·9
Nitrogen	75·9
Carbon dioxide	12·1
Carbon monoxide	8·1

Dr. Haldane thinks it probable that undiluted after-damp contains on the average about 3 per cent. of carbon monoxide; adding, however, that undoubtedly "there may be either a higher or a lower percentage, according as the conditions vary in different parts of the explosion"; and this would seem to be the truth of the matter, so far as one can judge from actual experience.

The active poisoning agent in after-damp is, then, carbon monoxide. It is doubtful whether carbonic acid gas is often present in sufficient quantity to induce any anæsthetic effect, the latter acting rather as a diluent of the oxygen than as a poison. The nitrogen (and argon) has no specific action on men or animals; its presence in excess in the after-damp is due to the deficiency of oxygen. When air which contains less than 2 per cent. of oxygen is breathed, loss of consciousness ensues in about fifty seconds. When there is only about 17 per cent. of oxygen in the air, candles and safety lamps are extinguished. When absence of oxygen from the air is the cause of death, the indicating appearances are blueness of the face, lips, tongue, &c., and distension of the veins of the neck and chest.

Dr. Haldane found that sulphuretted hydrogen was present in the after-damp produced in the experimental coal-dust explosions at Altofts Colliery, and thinks it probable that the smarting of the eyes and irritation of the air passages so often produced by after-damp are due to this gas.

Sulphuretted Hydrogen (H_2S): Chemical and Physical Properties.—This is a very poisonous gas, somewhat heavier than air, having a specific gravity of 1·175. It is easily ignitible, as an iron wire at dull red heat is sufficient to inflame it, when it burns with a bluish flame, forming water and sulphur dioxide (SO_2). It dissolves in water to a considerable extent, and is frequently so found in mineral springs (Harrogate waters for instance). The presence of the gas in the atmosphere or dissolved in water is readily detected by its noisome odour, which is like to that of rotten eggs. When inhaled, even if diluted with large quantities of air, it is an active poison.

Occurrence.—Sulphuretted hydrogen, when found in coal-mines, is due to the decomposition of iron pyrites. When gunpowder is exploded it is also generated to a very small extent. It is also sometimes found in metalliferous mines, being caused by the chemical action of acidulated waters on the sulphides of various metals in the ores, or occurring in springs in the mine, and although deaths to persons in mines have sometimes been attributed to poisoning by this gas, there are few, if any, well-established cases. Thus in the two incidences recounted by Mr. Nicholas Wood before the Committee of the House of Commons on Accidents in Mines, in 1853, it is quite possible that the poisoning mentioned was due to carbon monoxide rather than to sulphuretted hydrogen gas. Use can be made of the blackening effect that the gas has on white lead (compound of lead carbonate and lead hydroxide, $2PbCO_3 + PbH_2O_2$), which it converts into sulphide of lead (PbS) to determine its presence, the best way being to paint a sheet of paper with white lead and suspend the same in the place where the presence of the gas is suspected.

Nitrous Fumes and Gases Derived from Explosives.—These are produced by the partial burning of explosives. It is very doubtful whether any nitric peroxide whatever is produced by the complete explosion of nitro-glycerine compounds, but it has been stated that "one tiny $\frac{7}{8}$ in. cartridge of blasting gelatine can produce enough nitric oxide to kill many people."[1] In the complete explosion, however, of nitro-glycerine, large amounts of carbonic oxide and carbonic dioxide are produced.

The Transvaal Mining Commission state[2] in their Report "that it is only within recent years that the relative frequency of this form of gassing in the mines of the Rand [*i.e.* poisoning by nitrous fumes] has been at all recognised, and there is little doubt that many of such cases, in which no history of exposure to fumes has been elicited, have been looked upon as acute pneumonias. From the statistics before us, such poisoning appears to occur most commonly as the result of even *very brief* exposure, *immediately or shortly after blasting*, to fumes caused by the accidental *burning*

[1] Final Report of the Transvaal Mining Regulations Commission, 1910, p. 117.
[2] *Ibid.*, p. 119.

of comparatively small quantities of explosives in *drives or levels without subsequent explosion.*" They add that in moist atmospheres the danger from NO "seems to disappear within an hour or so after blasting, owing no doubt to the high solubility of the gas."

Seven ounces of gunpowder, when exploded, produce 1·92 cubic feet of gases, which have the following composition :[1]—

CO_2 . . .	49·9 per cent. or	0·95808 cubic feet.
CO . . .	14·1 ,,	0·27072 ,,
N . . .	33·3 ,,	0·63936 ,,
H_2S . . .	2·7 ,,	0·05184 ,,

"To become harmless, the carbon dioxide must be reduced to 1 per cent., the carbon monoxide to 0·02 per cent., the nitrogen increased to 77·95 per cent., and the sulphuretted hydrogen reduced to 0·02 per cent."[2]

[1] *Trans. Inst. M.E.*, vol. ii. p. 378.
[2] *Ibid.*, vol. xiii. p. 401.

CHAPTER V

THEORETICAL CONSIDERATIONS RELATING TO THE CIRCULATION OF AIR CURRENTS IN MINES—THE FRICTIONAL RESISTANCE OF THE AIR-WAYS

General Consideration of the Subject.—The factors that have to be considered in relation to the flow of air through mines—other than those which have already been mentioned and described in the preceding chapters, such as humidity, generation of noxious gases, and the requirements of persons and animals—are those which, for want of a better word, may be included under the expression the Mechanics of Ventilation, as opposed to the Chemistry and Physics of Ventilation.

These factors are largely interdependent on one another, and can conveniently be considered under three heads, viz. :—

1. The retarding force of friction.
2. The velocity of the air currents.
3. The power required for the production of the air current.

Two theorems of mine ventilation may be laid down, the truth of which will presently be demonstrated :—

(1) The sectional area of the roadway and the velocity of the air current remaining constant, the power necessary to overcome friction varies directly as the rubbing surface of the roadway.

(2) The rubbing surface of the roadway and the velocity of the air current remaining constant, the power per square foot necessary to overcome friction varies inversely as the sectional area of the roadway. The greater the area the less the power per square foot required.

The Retarding Force of Friction.—The extent to which this operates is determined by the extent and nature of the rubbing surface of the air-ways of the mine, and the extent of the rubbing surface is governed by—

(*a*) The length of the air-way.
(*b*) The ratio of its perimeter to its sectional area.

(*a*) *The Length of the Air-way.*—Other things being equal, the greater the length of the air-way the greater will be the total amount of friction offered to the flowing air, so that in a very long air-way the difference between the pressure at the points of inlet and outlet may be

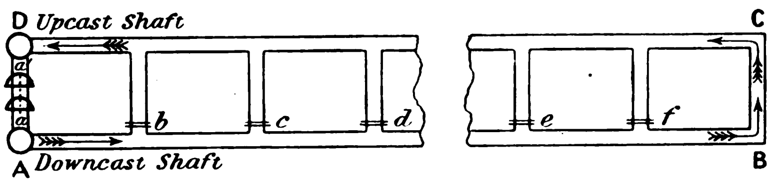


FIG. 8.—Illustrating the Loss of Pressure in a Ventilating Current.

quite considerable. Thus, supposing AB and CD (see Fig. 8) to represent two parallel roads, of which AB is the intake and CD the return air course, if the loss of pressure be measured, either by the water-gauge or the barometer, at the points *a*, *b*, *c*, *d*, *e*, and *f*, the readings will be found to show a gradual decrease, being greatest at *a* and least at *f*; that is to say, the density of the air has been reduced, so that if barometric readings be taken at, say, the points *a* and a^1, and these points are at the same level, that taken at a^1 will indicate a fall in the mercurial column, whereas a reading taken at B will differ but little from one taken at C. The density of the air decreasing as it proceeds on its journey, *a* and a^1 are the points of greatest difference. That is to say, there is a gradual reduction in the pressure of the air in its course through the mine, which is due to the absorption of pressure in overcoming the resistance of friction, the reduction of pressure being an exact measure of the pressure so expended.

(*b*) *The Perimeter of the Air-way.*—The length of any two or more roadways being the same, the amount of the rubbing surface will vary directly in accordance with the perimeter, the area of the rubbing surface being the multiplicand of the length by the perimeter.

The form of roadway, therefore, which will offer least rubbing surface to the passing air is the circular, the circumference of a circle, in proportion to its area, having less extreme perimeter than that of any other figure. Thus, if the area of a circle is unity, the length of its circumference is 3·545,[1] whereas the length of the perimeter of a square of like area is 4.

Another aspect of the question may be mentioned, though it is perhaps sufficiently obvious. One large air-way offers less rubbing surface than several air-ways, the sum of whose areas are equal to the area of the single road. Thus in the case of an air-way 12 by 10. The area $= 12 \times 10 = 120$ square feet, and the perimeter $= 12 + 12 + 10 + 10 = 44$ lineal feet; whereas four roads each having an area of $5 \times 6 = 30$, and $30 \times 4 = 120$ square feet, yet the perimeter of each

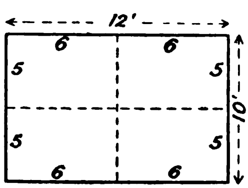


FIG. 9.—One Large Rectangular Air-way compared with Four Small Rectangular Air-ways.

[1] Calculated in the following manner:—

If the area of the circle $= a = 1$ and the diameter $=$ D.

The relation of the diameter to the perimeter $= \pi = 3{\cdot}1416$.

Then $D^2 \times {\cdot}7854 = a$.

$$\text{or } D = \sqrt{\frac{a}{0{\cdot}7854}}$$

and the perimeter $= \pi \times D = 3{\cdot}545$.

$= 5 + 6 + 5 + 6 = 22$ and $22 \times 4 = 88$, or twice as much as that of the single large road (see Fig. 9).

Of all rectangular figures of like area the square has

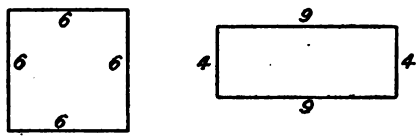


FIG. 10.—The Square as compared with other Forms of Rectangular Air-ways.

the least perimeter. Thus taking a square, the sides of which are each 6 feet (see Fig. 10),

$$\text{area} = 6 \times 6 = 36 \text{ square feet,}$$

$$\text{perimeter} = 6 \times 4 = 24 \text{ feet,}$$

and a rectangular figure 9 by 4,

$$\text{area} = 9 \times 4 = 36 \text{ feet,}$$

$$\text{perimeter} = 9 + 9 + 4 + 4 = 26 \text{ feet,}$$

or 2 feet more than in the case of the square.

The question of area in connection with the second theorem may now be considered.

Suppose the case to be that of two air-ways (see Fig. 11), the one 9 feet by 8 feet, and the other 13 feet by 4

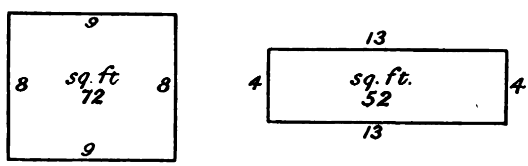


FIG. 11.—Rectangular Roadways of Like Area, but Unequal Perimeter.

feet. The perimeter of both will be the same, viz. 34 feet; but the area of the one will be 72 square feet and the other only 52 square feet—that is to say, $\frac{52}{72}$, or $\frac{13}{18}$ths of the other.

The rubbing surface and the velocity of the air being the same in each case, the pressure required per square foot of sectional area would, in the larger road, be but $\frac{13}{18}$ths of that in the smaller road, that is $\frac{13}{18}$ths of the water-gauge, and the quantity of air passing along the air-way of less area would also be $\frac{13}{18}$ths of that passing along the road of greatest area for $Q = va$, where Q = the volume of air, v the velocity of the current, and a the sectional area of the roadway.

The Power required to Overcome Friction.—Were friction a negligible quantity, the problem of the circulation of air through mines would be a simple one, for it would then resolve itself into a consideration of velocity only, as in the case of wind.

If, for instance, there was a difference of pressure at any two points of 1 lb. per square foot, there would be induced a velocity of air current, or wind, of 28 feet per second. Thus supposing 1 lb. to be the weight of a column of air, at normal temperature and pressure, 1 square foot sectional area and 13 feet high.

Then if v = the velocity of a falling body in feet per second.
h = the height in feet through which the body falls, viz. 13 feet.
g = the force of gravity (32·2).

Then $v^2 = 2gh$,
or $v = 28{\cdot}9$ feet.

But the problem is not such a simple one, for, as has been indicated, a mine for purposes of theoretical consideration in connection with the flow of air through it must be regarded as a very long conduit or passage, through which the air is forced or drawn, so that in considering the work done in drawing an air current through a mine not only does velocity form a factor, but, also, the pressure necessary to overcome friction. Thus if

u = work performed in foot pounds.
p = the unit of pressure, *i.e.* pressure per sq. ft.
a = sectional area of the roadway of the mine.
v = the velocity of the air current in feet per second.

Then $u = pav$.

It is generally accepted that in the same air-way the pressure required to overcome the frictional resistance of an air current varies directly as the square of the velocity of the current. That is to say, that if the velocity of an air current forced or drawn through a mine is doubled, the quantity is doubled, and double the quantity of air meeting the resistance offered by the sides of the roadway at double the velocity, frictional resistance is increased fourfold, and the original pressure will have been increased four times to secure the result. By the same reasoning, if the velocity be reduced to half, the pressure required to overcome the friction, due to half the quantity of air travelling at half the velocity, will have been reduced to a quarter.[1]

If it is assumed that—

(1) The measurement of the perimeter is uniform throughout the length of an air-way.

(2) That the density of the air is constant.

Then P or $pa = ksv^2$

where P = the total pressure,

[1] The present writer has, however, reason to doubt the accuracy of this statement. Judging from actual experiment with air currents, he is inclined to believe that the viscosity of the air is an important factor in the case, and one which has been hitherto disregarded in calculations respecting mine air currents, and that the variation in the pressure required to overcome the friction opposing the air current is somewhat less than the square of the velocity. For the determination of the constant, however, further and more detailed experiments are necessary, but the writer is in agreement with M. Rateau, who has described an experiment carried out at the Montrambert Collieries on the whole mine, which showed the ventilating pressure to increase as the 1·75th power of the volume. (*Bulletin de la Société de l'Industrie Minérale*, series 3, vol. vi. p. 133.)

and p = the pressure, per square foot of sectional area of the air-way, required to overcome friction.
a = the sectional area of the air-way in square feet.
k = a coefficient of friction taken in the same terms as the unit p.
s = the extent of the rubbing surface in square feet.
v = the velocity in thousands of feet per minute.

Of course the greater or less the density, *i.e.* the weight, of the air, the greater or less the amount of the friction—other things being equal; but the variation in respect of density is so small as to be a negligible quantity.

The Coefficient of Friction.—The determination of the coefficient of friction, or in other words the pressure measured in inches or decimals of an inch of water-gauge, or in feet of air-column required per square foot of rubbing surface to move air through the mine at a velocity of 1000 feet per minute, is a matter of the highest importance, and one which has occupied the attention of several experimentalists.

MM. d'Aubuisson and Péclet determined the amount of frictional resistance offered to the passage of air through iron pipes, and the late Mr. J. J. Atkinson, in his classic work "On the Theory of the Ventilation of Collieries" (1854),[1] commenting on their experiments, arrived at the conclusion that the coefficient of friction for the galleries of mines is on the average equal to 0·26881. That is to say, that a column of air of the same density as the flowing air 0·26881 feet high would be required to overcome the frictional resistance due to a current of air moving at a velocity of 1000 feet per minute through a passage 1 foot sectional area, and presenting 1 square foot

[1] "On the Theory of the Ventilation of Collieries," by J. J. Atkinson, *Trans. N. of E. Inst. M.E.*, vol. iii. (1854–5) pp. 73–222.

of rubbing surface to the moving air. This is equivalent, at a temperature of 32° F., to 0·0217 lb. pressure.

In 1866 M. Raux carried out a series of experiments at the Crachet-Picquey, Forchies, and Grand-Buisson Collieries,[1] in Belgium, and arrived at an average coefficient of 0·00166 inch of water-gauge, which M. Devillez increased to 0·00180 in order to account for the natural ventilation.

M. Daniel Murgue[2] a short while ago carried out a series of elaborate experiments, with a view to determining, as accurately as possible, the coefficient of friction for various types of galleries, the results of which may be summarised as follows:—

Nature of Gallery.	Average Coefficient of Friction in Decimals of an Inch of Water-gauge.
Unlined galleries . .	From 0·00087 to 0·00123
Arched „ . .	„ 0·00030 „ 0·00062
Timbered „ . .	„ 0·00147 „ 0·00241

From which it will be seen, as might be expected, that the friction is greatest in the case of timbered roadways, less in unlined galleries, and least in those which are arched.

The experiments showed that the coefficient increased with the curvature of the roadways, and that there was a sensible increase in the coefficient—especially in timbered galleries—in the case of galleries of small area.

Epitomising the results arrived at by various authorities, and stating them in terms of the water-gauge, we have as coefficients—

Atkinson (1854), 0·00417 inch.
Devillez (1866), 0·00186 „
Murgue (1893), 0·00030 to 0·00241 inch.

[1] *Ventilation des Mines*, by M. Devillez, pp. 32–34.

[2] "The Friction of, or Resistance to, Air Currents in Mines," by Daniel Murgue, *Trans. Inst. M.E.*, vol. vi. (1893–4), pp. 135–78.

It will be seen that the later experiments point to a diminution in the coefficient; but in respect to the use to which it may be put, the advice of Mr. A. L. Steavenson that he "would not recommend any one who had to erect a fan, or who had to go seriously into the question of the friction of the mine, to take any value very much less than that recommended by Mr. Atkinson," should be borne in mind. The employment of a wide margin is most necessary in all such calculations.

Examples in the Use of the Preceding Formulæ.—The practice to which the preceding formulæ can be put may be demonstrated by the following examples, which will also perhaps assist the student more clearly to apprehend their meaning:—

Example 1.—What pressure will be necessary to overcome the frictional resistance offered to an air current of 25,000 cubic feet per minute passing along three-quarters of a mile of air-ways of an average cross-section of 9 × 5 feet?

$9 \times 5 = 45$ square feet sectional area, and $25{,}000 \div 45 = 555$ (about) = velocity of air current in feet per minute; $\frac{3}{4}$ mile = 3960 feet, and the perimeter measures 28 feet; $\therefore$ $3960 \times 28 = 110{,}880$ square feet of rubbing surface.

$$\text{Then } p = \frac{ksv^2}{a} = \frac{\cdot 26881 \times 110{,}880 \times 0\cdot 3}{45}$$
$$= 204 \text{ pressure in feet of air column}$$

and as the weight of a cubic foot of air at standard temperature and pressure is 0·080728,

$$204 \times 0\cdot 080{,}728 = 16\cdot 46 \text{ lbs. per square foot}$$

$$\text{or } \frac{16\cdot 46}{5\cdot 2} = 3\cdot 16 \text{ inches of water-gauge (about)}$$

and $\frac{25000 \times 16\cdot 46}{33{,}000} = 12\cdot 5$ horse-power involved in overcoming the total frictional resistance offered to the air-current.

In the calculation it is presupposed that one air course only is under consideration, but in determining by calculation the probable power necessary to overcome the ventilation friction in an actual mine all the separate splits would have to be considered. Often in mines there are stages of development in the mechanical ventilation commensurate with the growth of the requirements. But it is common practice to estimate the fan requirements from actual practice at another working mine where the conditions in respect to depth, size of roads, &c., are similar.

Example 2.—The water-gauge shows a depression of 2·25 inches as between the bottom of the downcast and upcast shafts. A reading of the anemometer near the bottom of the downcast shaft, where the roadway is 12 feet wide by 6 feet high, shows the velocity per minute of the current to be 1041·66 feet. What is the horse-power of the ventilating current?

$1041{\cdot}66 \times (12 \times 6) = 75{,}000$ cubic feet of air per minute. The weight of a cubic foot of water may be taken as 62·5 lbs., so that $62{\cdot}5 \div 12 = 5{\cdot}2$ lbs., being the weight of a layer of water 1 inch deep and 1 square foot in area, or the difference of pressure per square foot represented by one inch of water-gauge.

$$\text{Therefore} \quad \frac{75{,}000 \times (2{\cdot}25 \times 5{\cdot}2)}{33{,}000} = 26{\cdot}6 \text{ H.P.}$$

The question of splits, or separate currents, does not enter into consideration in this case, as the total work being performed is measured by the total amount of air entering the mine, and the difference in its density as between the point of inlet and outlet of the same.

Example 3.—Given a current of 96,000 cubic feet of air circulating in a mine, the water-gauge as between points of inlet and outlet measuring 1·2 inches, what will be the water-gauge if the current is increased to 126,000 cubic feet per minute?

The velocity is increased $\frac{126,000}{96,000}$, or 1·31 times, and as the friction varies directly as the square of the velocity, therefore—

$$x : 1{\cdot}2 :: 126{,}000^2 : 96{,}000^2$$
$$\text{or } 1 : 1{\cdot}31^2 :: 1{\cdot}25 : x$$
$$\therefore\ x = 2{\cdot}067 \text{ inches.}$$

Example 4.—If with an air-current of 96,000 cubic feet per minute there is a water-gauge of 1·2 inches, what will be the amount of the current, other things being equal, if the velocity of the fan is increased so as to show a water-gauge of 2·067 inches?

If the frictional resistance varies directly as the square of the velocity, the velocity of the air (the area remaining the same), and hence the quantity of the air, is directly proportional to the square root of the measure of the friction, *i.e.* of the water-gauge.

$$\text{Hence} \quad \sqrt{1{\cdot}2} : \sqrt{2{\cdot}067} :: 96{,}000 : x$$
$$x = 126{,}000 \text{ cubic feet per minute.}$$

Or thus: The water-gauge is increased $\frac{2{\cdot}067}{1{\cdot}2} = 1{\cdot}7225$ times,

$$\text{and} \quad \sqrt{1{\cdot}7225} = 1{\cdot}31,$$
$$\text{and } 96{,}000 \times 1{\cdot}31 = 126{,}000 \text{ cubic feet per minute.}$$

$$\text{Or thus:} \quad x^2 : 96^2 :: 2{\cdot}067 : 1{\cdot}2$$
$$x = 126{,}000 \text{ cubic feet per minute.}$$

Example 5.—Supposing there are two air-ways of the same rectangular section varying only in respect of length, the one being 14,000 feet, the other 18,000 feet long. If the quantity of air descending the mine is 23,000 cubic feet per minute of this current, how much will each air-way take?

By the formula $ksv^2 = pa$

and dividing by ks, $v^2 = \frac{pa}{ks}$

or $v = \sqrt{\frac{pa}{ks}}$

In these two air-ways pa and k have the same value. So that v varies only in respect of the rubbing surface, which, as the perimeters of the two roads are the same, varies only in respect of l the length of the road.

$$\text{Therefore } V \propto \sqrt{\frac{1}{l}}$$

the cross sectional areas of the two roads are the same, and as Q the quantity $= Va$

$$Q \propto \sqrt{\frac{1}{l}}$$

The proportion will therefore be as

$$\sqrt{\frac{1}{14{,}000}} : \sqrt{\frac{1}{18{,}000}}$$

or as 0·0084 : 0·0074, or multiplying these figures to eliminate the decimal

$$\begin{aligned} 0{\cdot}00084 \times 50{,}000 &= 42 \\ 0{\cdot}00074 \times 50{,}000 &= 37 \\ & \overline{79} \end{aligned}$$

and $\frac{42}{79}$ths of 23,000 = 12,228 cubic feet.
$\frac{37}{79}$ths of 23,000 = 10,772 „

23,000 „

It will be seen that the formula $Q \propto \sqrt{\frac{1}{l}}$ is that which would be employed to determine the effect of lengthening an air course, the sectional area and perimeter remaining constant.

Example 6.—The ventilation of a mine is by two drifts (see Fig. 12); at the point A the main current is split into three districts, X, Y, and Z, the splits uniting again at the point B. It is required to find the quan-

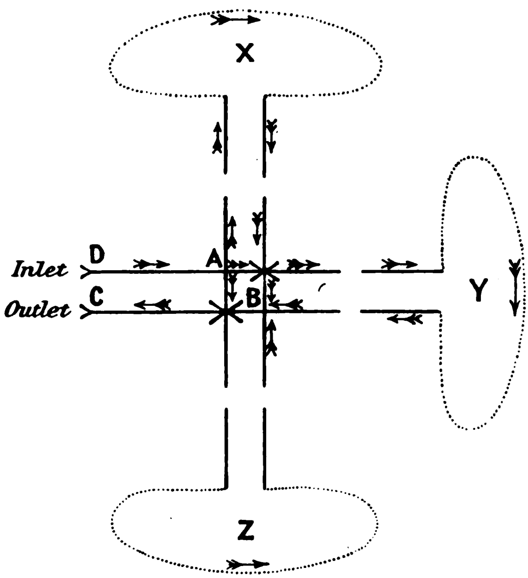


FIG. 12.—Showing three Air-current "Splits" of Unequal Length.

tity of air in each split, and the water-gauge as between D and C.

The distances and sections of the roads are as follows :—

D to A is	100 feet long,	and	$10' \times 10'$	section.
Split X	1000	,,	$8' \times 5'$	,,
,, Y	2000	,,	$6' \times 4'$	,,
,, Z	3000	,,	$10' \times 5'$	,,

B to C is 200 feet long and $10' \times 10'$ section, and the total quantity of air passing into the mine is 100,000 cubic feet per minute.

This example is somewhat similar to the preceding one, but it will be noticed that the cross-sectional areas and the perimeter vary in respect of each of the splits.

The water-gauge also is required.

For comparing the quantities which travel in the various splits it has been shown that

$$\frac{ksv^2}{a}=p$$

and dividing by $\frac{ks}{a}$, $v^2=\frac{pa}{ks}$

$$\text{or } v=\sqrt{\frac{pa}{ks}}$$

and multiplying by a, $va=Q=a\sqrt{\frac{pa}{ks}}$

but as p and K are the same for all the splits,

$$Q \propto a\sqrt{\frac{pa}{ks}}$$

Again, as a certain pressure is maintained to induce the air to flow through the mine, it is obvious that the water-gauge taken across the ends of the splits is the same in each case, as the air flows through as many splits as are provided proportionately to the resistance, the greater the resistance the less the volume of air; and the water-gauge of any of the splits—if all the splits commence from the bottom of the downcast and unite at the bottom of the upcast shaft—will be the water-gauge for the whole pit.

The water-gauges across the ends of the splits X, Y, Z, are the same, and since in the formula $p=\frac{ksv^2}{a}$ for each split p, the pressure per square foot is equal, and k is a constant. Then, denoting the splits by the indicating letters x, y, z—

(1) $$\frac{s_x v_x^2}{a_x}=\frac{s_y v_y^2}{a_y}=\frac{s_z v_z^2}{a_z}$$

and as the square of the velocity varies, other things being equal, directly as the area and indirectly as the rubbing surface,

(2) $$v_x^2 : \frac{a_x}{s_x} : : v_y^2 : \frac{a_y}{s_y} : : v_z^2 : \frac{a_z}{s_z}.$$

(3) That is, the velocity varies as $\sqrt{\frac{a}{s}}$.

$$\text{but since} \quad v = \frac{Q}{a} \text{ (for } va = Q) \quad v^2 = \frac{Q^2}{a^2}$$

transposing $\frac{Q}{a}$ for v in the expression $v \propto \sqrt{\frac{a}{s}}$

$$\frac{Q}{a} \propto \sqrt{\frac{a}{s}}$$

$$\text{or} \quad Q \propto a\sqrt{\frac{a}{s}}$$

Returning to the figures in question—

For the road DA

$$v \begin{pmatrix}\text{in thousands of} \\ \text{feet per minute}\end{pmatrix} = \frac{Q}{a \times 1000} = \frac{100{,}000}{100{,}000} \text{ or } 1 \begin{pmatrix}\text{thousand feet} \\ \text{per minute}\end{pmatrix}$$

$$p = \frac{ksv^2}{a} = \frac{\cdot 02^* \times (40 \times 100) \times 1^2}{100} = \cdot 8 \text{ lb.}$$

For the quantity in the splits—

$$\text{In split X, } Q = 40\sqrt{\frac{40}{26 \times 1000}} = 1{\cdot}569 \times 29985 = 47{,}046 \text{ cub. ft.}$$

$$\text{,, Y, } Q = 24\sqrt{\frac{24}{20 \times 2000}} = 0{\cdot}588 \times 29985 = 17{,}632 \text{ cub. ft.}$$

$$\text{,, Z, } Q = 50\sqrt{\frac{50}{30 \times 3000}} = 1{\cdot}178 \times 29985 = 35{,}322 \text{ cub. ft.}$$

$$3{\cdot}335)100{,}000(29{,}985$$

N.B.—The multiplier 29985 is arrived at by dividing 3·335 into Q, which is 100,000.

To arrive at the water-gauge or pressure required to overcome the frictional resistance.

$$\text{V in thousands of feet per minute} = \frac{Q}{1000 \times a} = \frac{47046}{1000 \times 40}$$

$$\text{so that in } p = \frac{ksv^2}{a}$$

$$p = \frac{\cdot 02 \times (26 \times 1000)}{40} \times \frac{47046 \times 47046}{(1000 \times 40) \times (1000 \times 40)} = 17{\cdot}98 \text{ lbs.}$$

---

* The value of Atkinson's coefficient, see p. 71.

and the same pressure per square foot of sectional area will be found in the case of the splits Y and Z also.

To this figure, however, will have to be added the pressure for the inlet drift already determined at ·8 lb., and that for BC, viz.—

$$\text{For BC} \quad p = \frac{ksv^2}{a} = \frac{\cdot 02 \times 40 \times 200 \times 1^2}{100} = 1\cdot 6 \text{ lbs.}$$

		lbs.
Pressure per square foot for inlet drift	=	0·80
" " " the splits (one or all) .	=	17·98
" " " fan drift (*i.e.* resistance due to distance, BU)	=	1·60
Total . . .		20·38

$$\frac{20\cdot 38}{5\cdot 2} = 3\cdot 92 \text{ inches.}$$

Example 7 : *Power necessary to Produce an Air-Current.*—To determine the amount of work done in producing ventilating currents of air for the mine.

The work done in producing the air current is the product of the total pressure by the velocity. So that, if U = the work done per minute [1]—

$$\begin{aligned} \text{As} \quad pa \text{ or } P &= ksv^2 \\ Pv \text{ or } u &= (ksv^2)v \\ u &= ksv^3 \end{aligned}$$

That is to say, the power necessary to produce an air-current in a mine varies as the cube of the velocity, and this is the formula that would have to be used for calculating the horse-power of the contrivance for creating the current.

[1] It has been previously stated that u = Quantity × Pressure per square foot. Both formulæ are correct, for

$$Q = va \therefore Qp = u \text{ and } vP = u \text{ where } P = pa$$
$$\text{Since } vP = vap = Qp.$$

Example 8 : *Splitting of Air-Currents.*—If the roads of a mine are level, no matter how long or short they may be, each roadway will get a fixed share of the whole of the air descending the downcast shaft. The advantage in ventilating the mine by a number of separate currents instead of one continuous current may be stated thus :—

1. The total friction offered to the circulating air is lessened, for though the rubbing surface is not lessened, the area is greatly increased. Hence—

2. For a given pressure the quantity is increased.

3. The individual currents do not become so fouled with gas, or suffer loss of oxygen to the same extent as one continuous current.

It stands to reason that the more equally the splits are divided the better, so that it should be possible, under ideal conditions, to dispense with the use of regulators (sliding doors) or fictitious obstructions necessary to check the quantity of air passing into a particular district, in order that another district may secure a supply of air greater than would otherwise be the case. The effect of obstructions in the air-ways is to diminish the total quantity of air ventilating the mine.

Splitting the air should not be carried out to such an extent as by diminishing the velocity of the currents will render them unable to sweep away the noxious gases generated in the mine.

It may be taken as a good rule that the velocity of the air current for coal-mines should be—

In the main intakes about 10 to 18 feet per second.
At the face about 3 to 4 feet per second.

The advantage of splitting the air can be mathematically demonstrated thus.

Considering, in the first instance, the case of three separate splits. Let Fig. 13 represent the plan of a mine which is divided into three districts separately ventilated, the air splitting at the bottom of the down-

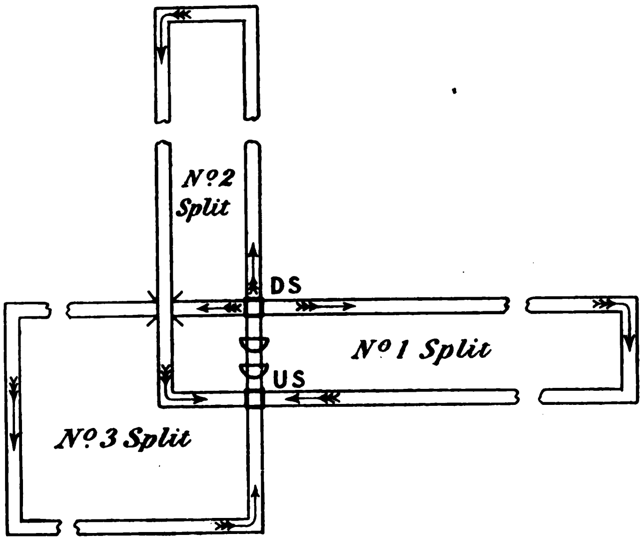


FIG. 13.—Three Air-current "Splits" of Equal Length, the Roadways being of the same Cross-section.

cast shaft and reuniting at the bottom of the upcast shaft.

The distance traversed by each split is the same, and it is assumed—

1. That the cross section of the shafts and levels is the same, viz. $15 \times 6 = 90$ square feet.

2. That the water-gauge as between the downcast and upcast shafts is 1 inch $= 5{\cdot}2$ lbs. per square foot[1] $= 64$ feet in height of motive column.

3. That the splits are 3000 feet long $= 126{,}000$ square feet of rubbing surface.

[1] $5{\cdot}2 \div {\cdot}0807 = 64$, see p. 101.

$$\text{Then } p = 64$$
$$s = 126{,}000$$
$$a = 90$$
$$k = 0{\cdot}26881$$
$$v^2 = \frac{64 \times 90}{0{\cdot}26881 \times 126{,}000}$$
$$v = {\cdot}491$$

Velocity in feet per minute $= 0{\cdot}412 \times 1000 = 412$
Cubic feet per minute in each split $= 412 \times 90 = 37{,}080$
Or a total quantity in the mine of $37{,}080 \times 3 = 111{,}240$ cubic feet.

Now supposing that the mine is ventilated by one

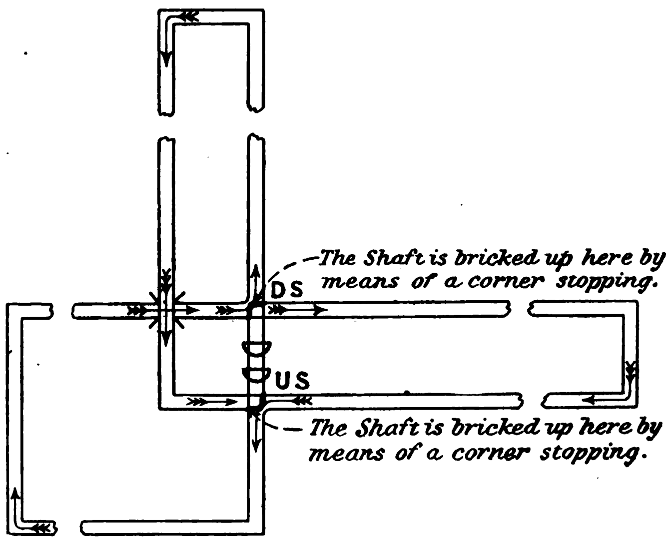


FIG. 14.—Mine Ventilated by a Single Air-current.

continuous current, this can be done by building stoppings at DS and US in the manner shown in Fig. 14.

$$\text{Then } p = 64$$
$$s = 378{,}000$$
$$a = 90$$
$$k = 0{\cdot}26881$$
$$v^2 = \frac{64 \times 90}{{\cdot}26881 \times 378{,}000}$$

$$v^2 = 0{\cdot}0567$$
$$v = 0{\cdot}238$$

Velocity in feet per minute $= 0{\cdot}238 \times 1000 = 238$
Quantity in cubic feet per minute $= 238 \times 90 = 21{,}420$

The second calculation would have been shortened by simply dividing the velocity calculated in the first case by $\sqrt{3}$ to arrive at the velocity in the second case, thus—v = velocity in first example, v_1 = velocity in the second example. Thus—

$$v_1^2 = \frac{pa}{k3s}$$

Multiplying both sides of the equation by 3, $3v_1^2 = \dfrac{pa}{ks}$

$$\text{But} \quad v^2 = \frac{pa}{ks}$$

$$\therefore\ 3v_1^2 = v^2$$

$$\text{or} \quad v_1 = \frac{v}{\sqrt{3}}$$

and so for any number of splits $v_1 = \dfrac{v}{\sqrt{\text{number of splits}}}$

The student, by an application of the reasoning demonstrated in the foregoing examples, will be able to solve many kindred problems dealing with the theory of ventilating currents.

CHAPTER VI

INSTRUMENTS USED FOR AIR MEASUREMENTS IN THE VENTILATION OF MINES: THE BAROMETER, THERMOMETER, HYGROMETER, ANEMOMETER, PITOT TUBE, AND WATER-GAUGE

The Barometer.—The principle on which the barometer acts, and its construction, are too well known to need description here. The value of the barometer as an indicator of the atmospheric pressure is considerable, but it is very doubtful whether it is a sufficiently delicate instrument to indicate the atmospheric changes as soon as gaseous substances do. Water barometers, as being more marked indicators in point of height of fall or rise of pressure, have sometimes been used—as the density of water being 13·6 times less than that of mercury, the height of water column is 13·6 times greater than the corresponding mercurial column—but such barometers are, of course, subject to the inconvenience of evaporation and unwieldy height. It would be an advantage could some reliable form of gas barometer be devised, as being more susceptible to variations of pressure, but an apparently insurmountable difficulty exists in the great expansion and contraction effects due to variations in temperature.

A thermometer is usually attached to the case containing the mercurial barometer, as it is necessary to know the degree of temperature prevalent at the time of making a barometric observation in order to reduce each reading to its equivalent at 32° F. or 0° C. As mercury expands ·00014 of its volume for each degree Fahrenheit (or ·00025 for each degree Centigrade), it is necessary to subtract $\frac{1}{10000}$ of the observed height of the column of mercury for each degree above 32° F., or if the temperature is below 32° F., to add $\frac{1}{10000}$ for each degree.

The height of the mercurial column may be said to vary from 28 to 31 inches at sea-level, hence 30 inches has been selected in Britain as the standard height. And as 1 cubic inch of mercury weighs 0·49 lb., a column 30 inches high (or more accurately 29·922 inches) will represent an atmospheric pressure per square inch of surface of $0·49 \times 30 = 14·7$ lbs. (or more exactly 14·696 lbs.).

Corrections of Barometer for Difference in Level and Variation in Temperature. — Barometric depressions for meteorological and other scientific purposes are reduced to sea-level readings. Thus if a reading of 28 inches be observed at some spot at an altitude of 1800 feet above sea-level, the depression reduced to sea-level would be about 30 inches.[1]

FIG. 15.—Form of Barometer in Common Use at Collieries. (Messrs. Davis & Son.)

The manner of correcting barometric readings for difference of level can be best explained by means of an example :—

[1] The following formula can be used for the reduction of barometric readings to standard level :—

Let X = the height in inches of the mercurial column at the lower station (*e.g.* sea-level).

Supposing the barometer is situated at 250 feet above sea-level, and the reading is 29·5 inches.

Let A = the altitude above sea-level at which the barometer is placed.
h = the height of the mercurial column in inches.
H = the height of a column of air capable of balancing a column of mercury 1 inch in height.

The value of H is determined by comparing the densities of mercury and air, the former being 10,466 times greater than the latter. Hence a column of mercury 1 inch in height will require to balance it 10,466 inches or 872·16 feet of air column, which height is the value of H.

$$\text{Correction } C = \frac{A \times h}{30 \times H} = \frac{A \times h}{30 \times 872{\cdot}16} = \frac{A \times h}{26164{\cdot}8}$$

Or in the example quoted—

$$\frac{250 \times 29{\cdot}5}{26164{\cdot}8} = 0{\cdot}28 \text{ inch}^{1}$$

H = the height in inches of the mercurial column at the higher station.
T = the temperature in degrees (Fahrenheit) at the lower station.
t = the temperature in degrees (Fahrenheit) at the higher station.
D = the difference of level in feet between the two stations.

Or simplified—

$$(1)\quad D = 49{,}000\,\frac{(X - H)}{(X + H)}\left(1 + \frac{T + t}{900}\right)$$

$$(2)\ \therefore\ X = H\,\frac{49{,}000\,(900 + T + t) + 900\,D}{49{,}900\,(900 + T + t) - 900\,D}$$

It will be seen that the first of these two formulæ can be applied to working out the differences in heights of various stations from barometric readings—in other words, to levelling by means of barometers. For rough and ready purposes it is usual to allow 900 feet ascent or descent for every inch of variation in the heights of the mercurial column.

Consult also *Ganot's Physics*, trans. by E. Atkinson, Ph.D., F.C.S., 7th ed., pp. 124–27.

[1] 30 inches is taken instead of 29·922 for simplicity in calculation. Another and simple method is to add 0·1 in. for each—

85 feet up to 510 feet that the point is above sea-level.
90 feet from 510 up to 1140 feet.
95 feet from 1140 up to 1900 feet.
100 feet when above 1900 feet.

That is, the corrected reading is 29·78, or a difference in pressure of 0·28 inch.

Correction for Temperature.—As mercury expands or contracts with the application and withdrawal of heat, this correction is made by reducing the reading to zero of the Centigrade thermometer, to which the 32nd degree of Fahrenheit corresponds, that is, the height of the column is that which would appear if the temperature were at the freezing-point of water.

H = height of barometer.
t = temperature in degrees.
h = its height at zero.
d = density of mercury at zero.
d^1 = density of mercury at t degrees.

The heights H and h are in the inverse ratio of the densities d and d^1, that is—

$$\frac{h}{H} = \frac{d^1}{d}$$

If the volume of mercury is represented at zero by 1, and $1 + Dt$ is equal to the inverse ratio of the densities, that is—

$$\frac{d^1}{d} = \frac{1}{1 + Dt}$$

From these equables we have—

$$\frac{h}{H} = \frac{1}{1 + Dt} \text{ whence } h = \frac{H}{1 + Dt}$$

Substituting for D its value $\frac{1}{5550}$ we get—

$$h = \frac{H}{1 + \frac{t}{5550}} = \frac{H \times 5550}{5550 + t}$$

Example.—Find the true height of barometer, which marks 30 inches when the temperature is 25° C.?

$$h = \frac{30 \times 5550}{5550 + 25} = \frac{166500}{5575} = 29{\cdot}86 \text{ inches.}$$

Great Barometric Falls.—The present writer has seen the mercurial column stand, at sea-level, below 28 inches. On that occasion, December 8–9, 1886, it fell to 27·57 inches at eleven o'clock on the night of the

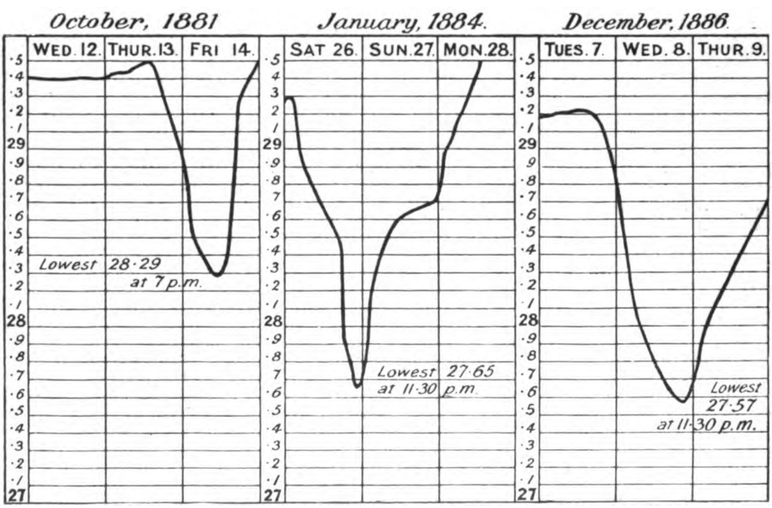


FIG. 16.—Barometric Chart showing three notable Atmospheric Depressions.

8th. The period from the 6th to 12th December of that year will be long remembered in the annals of meteorology as being marked by the most abnormal atmospheric conditions, which are almost, if not totally, unprecedented. The exceptionally low readings of the barometer over the whole of the British Islands, especially over the North of Ireland and the North of England, were altogether extraordinary.

The diagram on p. 88 represents graphically this and two other remarkable depressions (see Fig. 16).

Variations in the Height of the Barometer.—Not only does the height of the barometer vary from day to day, but also during the same day. The *mean daily height* is the division by 24 of the sum of the hourly readings taken over one day, and in our latitudes this average corresponds to the height at noon—that is, the barometer falls from noon to four o'clock, then rises and reaches its maximum at about 10 P.M., after which it again sinks, reaching a second minimum towards 4 A.M. and a second maximum at 10 A.M., and these hours of maxima and minima are the same for all climates. The *mean monthly height* is the sum of the mean daily heights for a month divided by 30, and the *mean yearly height* is obtained in a similar manner. The *general mean height* at sea-level is 29·96 inches, or 761 mm. The mean monthly height of the barometer is greater in winter than in summer months, but the greatest variations occur in winter.

Thermometric Scales.—Three scales are in use for reading temperatures: The Fahrenheit, which is chiefly in use in the United Kingdom, the Colonies, and the United States of America; the Centigrade (Celsius), which is employed in all scientific work, is the scale in common use in France, Germany, Austria, and Italy; and Réaumur, which is the scale of Russia, Sweden, and Norway.

These scales may be graphically represented thus—

	Fahrenheit.	Centigrade.	Réaumur.
Temperature of boiling water	212°	100°	80°
Temperature of freezing water	32°	0°	0°
	O	O	O

The relation between the scales, it will be observed, is 9, 5, 4. But in converting Fahrenheit into the other two the number 32 must always be first subtracted, and in converting Centigrade and Réaumur into Fahrenheit readings the number 32 must be added after the multiplication and division have been carried out.

Thus to convert 70° Fahrenheit into Centigrade and Réaumur readings—

$$\frac{5}{9}(\text{F.}^\circ - 32) = \text{C.}^\circ \ \textit{e.g.}\ \frac{5}{9}(70 - 32) = 21{\cdot}11^\circ \text{ C.}$$

$$\frac{4}{9}(\text{F.}^\circ - 32) = \text{R.}^\circ \ \textit{e.g.}\ \frac{4}{9}(70 - 32) = 16{\cdot}89^\circ \text{ R.}$$

or Centigrade and Réaumur into Fahrenheit—

$$\frac{9}{5}\text{C.}^\circ + 32 = \text{F.}^\circ$$

$$\frac{9}{4}\text{R.}^\circ + 32 = \text{F.}^\circ$$

or Centigrade into Réaumur or Réaumur into Centigrade—

$$\frac{4}{5}\ \text{C.}^\circ = \text{R.}^\circ$$

$$\frac{5}{4}\ \text{R.}^\circ = \text{C.}^\circ$$

It is perhaps easier to remember the fact that there are 180° between 32 and 212 on the Fahrenheit scale, and only 100° between 0 to 100 on the Centigrade scale, so when converting Fahrenheit into Centigrade deduct 32, and calculate by proportion thus—

$$70 - 32 = 38$$

$$\therefore\ 180 : 100 :: 38 : 21{\cdot}11$$

or C.° into F.°—

$$100 : 180 :: 21{\cdot}11 : 38^\circ$$

$$38 + 32 = 70^\circ.$$

Proceed in the same manner in respect to the Réaumur scale, remembering that as between F.° and R.° the proportion is as 180 : 80, and as between C.° and R.° it is as 100 : 80.

The Hygrometer.—For the purpose of measuring the extent of the humidity of the air, a wet and a dry bulb thermometer are used, these being placed side by side and constituting the instrument known as the hygrometer. The dry bulb, of course, registers the temperature of the surrounding atmosphere, whilst the bulb of the other thermometer, being kept damp by means of a covering of thin cotton, cambric, muslin, or woollen substance, shows a lower temperature, due to the evaporation of the moisture producing cold, unless the air be saturated, when there will be no difference—the difference between the indications of the two thermometers being greater in proportion as the air can take up more moisture. The covering of the "wet bulb" is kept moist by being connected with a small reservoir of water.

If air containing aqueous vapour be chilled, a point is arrived at, at which condensation takes place, and the vapour is deposited as dew. This is known as the dew-point—that is to say, the air and vapour both being gases, contract with the descent of temperature, and when a temperature is arrived at, at which the atmosphere is saturated, the dew will fall. The degree of humidity of a given volume of atmosphere is the ratio that the quantity of vapour present bears to complete saturation of the same volume—in other words, the ratio of the elastic force of vapour at the temperature of the dew-point to the elastic force of vapour at the temperature of the air, and the degree of humidity is expressed as so much per cent.

Figs. 17 and 18 show two good forms of hygrometers

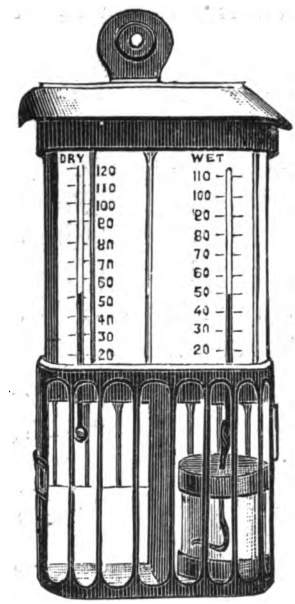


FIG. 17.—Stationary Hygrometer. (Messrs. J. Davis & Son.)

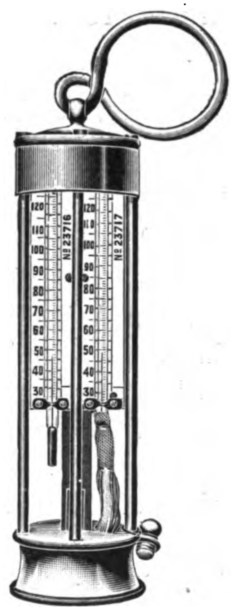


FIG. 18.—Portable Hygrometer. (Messrs. J. Davis & Son.)

for use in mines; the one is for stationary purposes, the other is portable.

If e = the tension of the aqueous vapour,
e' = the maximum tension corresponding to the wet bulb,
h = the height of the barometer,
t = the temperature as registered by the dry bulb,
t' = the temperature as registered by the wet bulb.
Then $e = e' - 0{\cdot}00077(t - t')h$,
0·00077 being constant, but a constant which depends on the position of the instrument. The above figure is such as would obtain for the conditions in a mine; in a small closed room it would be 0·00128.

The values of e' for various wet bulb temperatures have been given by Regnault as follows :—

TABLE III.—*Tension in Millimetres of Aqueous Vapour at Various Temperatures.*

Temperature in ° C.	Tensions in Millimetres.	Temperature in ° C.	Tensions in Millimetres.
– 10	2·078	+ 28	28·101
8	2·456	29	29·782
6	2·890	30	31·548
4	3·387	31	33·405
2	3·955	32	35·359
0	4·600	33	37·410
+ 1	4·940	34	39·565
2	5·302	35	41·827
3	5·687	40	54·906
4	6·097	45	71·391
5	6·534	50	91·982
6	6·998	55	117·478
7	7·492	60	148·791
8	8·017	65	186·945
9	8·574	70	233·093
10	9·165	75	288·517
11	9·792	80	354·643
12	10·457	85	433·410
13	11·062	90	525·450
14	11·906	91	545·780
15	12·699	92	566·760
16	13·635	93	588·410
17	14·421	94	610·740
18	15·357	95	633·780
19	16·346	96	657·540
20	17·391	97	682·030
21	18·495	98	707·260
22	19·659	98·5	726·150
23	20·888	99	733·210
24	22·184	99·5	746·500
25	23·550	100	760·000
26	24·998	100·5	773·710
27	26·505	101	787·630

Glaisher's method of obtaining the dew-point was by multiplying the difference of wet and dry bulb readings by a constant, depending on the temperature of the air at the time of observation, and subtracting the product from the reading of the dry bulb temperature, Glaisher's factors or multipliers being as shown in Table IV. below.

TABLE IV.—*Glaisher's Multipliers.*

Dry Bulb Temperature in ° F.	Factor.	Dry Bulb Temperature in ° F.	Factor.
Below 24	8·5	35–40	2·5
24–25	6·9	40–45	2·2
25–26	6·5	45–50	2·1
26–27	6·1	50–55	2·0
27–28	5·6	55–60	1·9
28–29	5·1	60–65	1·8
29–30	4·6	65–70	1·8
30–31	4·1	70–75	1·7
31–32	3·7	75–80	1·7
32–33	3·3	80–85	1·6
33–34	3·0		

This table was framed from an extensive series of comparisons executed at the Royal Observatory, Greenwich, by Mr. Glaisher.

If the temperature of the air be below 32° F. the wet bulb may register a higher temperature than the dry bulb; such a reading must not, however, be recorded, but the wet bulb must be moistened, when a coating of ice will form on it, from which evaporation will take place, though an hour or more may elapse before the temperature of the wet bulb has fallen below that of the dry one. If the temperature of the air should rise above 32° F., the wet bulb should be carefully immersed in warm water, to melt away any ice which may remain on it.

Example.—Let the dry bulb thermometer read

58° F. and the wet bulb 51° F. The difference = 7° F. And 7 × 1·9 (Glaisher's multiplier) = 13·3. The dew-point will therefore be 58 − 13·3 = 44·7° F. That is to say, the atmosphere would not be saturated until its temperature was lowered 1·9° F. below 58° F. And as the amount of moisture at saturation at 44·7° F. is 3·43 grains per cubic foot, and at 58° F. is 5·48 grains (about); then, as 5·48 : 3·43 : : 100 : 58·39 ; so 62·6 (about) is the percentage of saturation of the atmosphere of the mine air giving the above dry and wet bulb readings.

Dr. Apjohn's formula is—

$$F = f - \frac{d}{88} + \frac{h}{30}$$

Where

F = elastic force of the vapour at the dew-point.
f = tension of the vapour for the temperature of the wet bulb.
d = difference of wet and dry bulb thermometers in Fahrenheit degrees.
h = height of barometer in inches.
88 = constant coefficient for the specific heats of air and aqueous vapour when wet bulb is above 32° F.
96 = constant coefficient for the specific heats of air and aqueous vapour where the wet bulb is below 32° F.

Having arrived at the value of F., the dew-point may be found from the tables.[1]

THE MEASUREMENT OF THE AIR-CURRENT

The Anemometer.—The volume of air in cubic feet per minute passing in any part of a mine is found by multiplying the velocity of the air in feet per minute by the cross-sectional area in square feet of the part of the roadway where the velocity has been measured. An accurate method of finding the velocity is to measure

[1] The tables in common use are those of James Glaisher, F.R.S.

a stated distance of roadway, and having, by means of a stop-watch, determined the time taken for a puff of smoke, say of gunpowder, to traverse this distance, to calculate therefrom the velocity per minute. But there are two very obvious drawbacks to this method. It

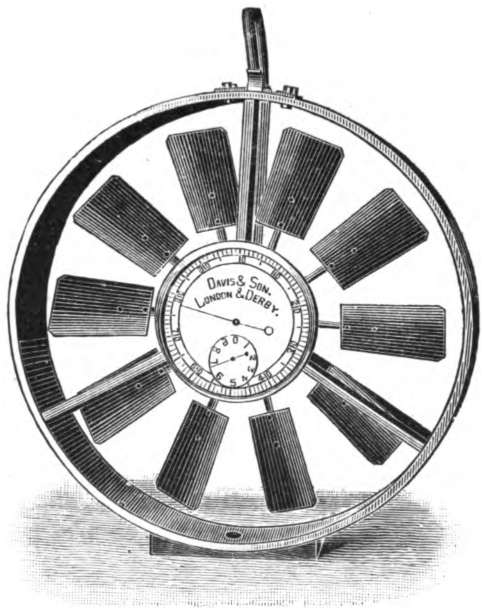


FIG. 19.—The Biram Anemometer.

requires for its fulfilment two persons, one to ignite the powder at one end of the measured length, and one at the other end to note the time, and it introduces an open light which, in many coal-mines, would be prohibitive on account of danger.

Air velocities are therefore usually measured by means of anemometers, of which there are a variety, all depending, however, on the same principle, viz. the rotation by the air current of a series of vanes supported from a spindle, which works an endless screw engaging in a series of toothed wheels ; the revolutions of these wheels are indicated by dial pointers. Fig. 19 illustrates a common form of anemometer used in coal-mines which is known as Biram's anemometer. The instrument is held up with its back to the current of air to be measured, the revolution of the light vanes being recorded upon a dial in the centre of the instrument. A convenient form of anemometer is that which was originally designed by L. Casella for Dr. Parkes of the Royal Victoria Hospital, Netley, for measuring the state of ventilation in that large military establishment, since when it has been used largely in mines. On the large dial the low velocity of 50 feet per minute may be measured, and by the smaller ones continuous registration is extended up to 10,000,000 feet (Fig. 20).

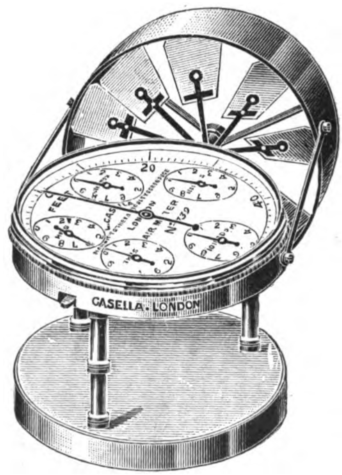


FIG. 20.—The Casella Anemometer.

Anemometers are usually fitted with a small catch for putting the instrument in and out of gear, and each instrument is furnished with a correction factor which has to be added to or subtracted from the mean velocity per minute recorded by the instrument.

Anemometers should be frequently tested in order to check the correction for friction. The method at one time used by the writer was to have constructed a wooden box 1 foot square and 60 feet in length, one end of which passed through a brick stopping into the main return air-way of the mine, near to the upcast shaft, the other end opening into the main intake air-way, so that a strong current of air passed into the return. A little gunpowder was fired at the intake end of the box, and the passage of the smoke observed by means of glass windows in the box. The anemometer was then placed inside and half-way along the box, and the vanes revolved for one minute, and the correction determined, *e.g.* the time taken by powder smoke to pass over a given distance, say 60 feet, is 23 seconds; therefore 60 cubic feet of air passes per 23 seconds, and 23 : 60 : : 60 : 156 cubic feet per minute. Number of revolutions recorded by the anemometer in one minute = say 105, and $156 - 105 = 51$, which is the addition which must be made for friction.

By means of a sliding door or regulator at the "return" end of the box, a very considerable variation in velocities, and a consequent series of corrections, can be secured.

When using the anemometer for measuring velocities of air-currents in mines, care should be taken to hold it at arm's length and at right angles to the direction of the current. As the sides of a roadway or shaft offer considerable resistance to the flow of air, the maximum velocity of current will be in the centre of the road. In order, therefore, to correctly determine the volume of air passing along the road, it will be necessary to take an average of the velocities, which can be done either by moving the instrument about in a zigzag direction over the sectional area, or, if great accuracy is desirable, the cross section of the roadway should be divided up into equal squares by means of fine wires, and the operator, seated in a recess in the side of the roadway, should, with the anemometer held at the end of a long stick, take a separate reading in the middle of each square, and strike the average of the same.

The Pitot Tube.—The Pitot tube is sometimes used for measuring or calculating the pressures in the case of air-currents instead of the water-gauge, and hence can be applied for the determination of velocities also. Its use is a simple matter if the velocity is as much as 4 or 5 feet per second, but below this, manometers are rather delicate.

The Pitot gauge was first described by its inventor, Pitot, in *Mémoires (Histoire) de l'Académie*, 1732, pp. 103 and 363. By observing the rise of the water-level in an open L-shaped tube, the opening in the lower end of which was opposed to the current, Pitot measured the velocity of the river Seine between the Pont Neuf and the Pont Royal. The water in the tube rises to such a height as would be necessary to give a velocity of efflux equal to the velocity causing the rise. In the case of a frictionless fluid, this is given by the Torricellian theorem, $v^2 = 2gh$ (1). Hence if h represents the height to which the water rises, and p the pressure corresponding to the height h,[1] as $h = \frac{v^2}{2g}$ (2) $h = \frac{p}{\rho g}$ (3) when ρ is the density of the water or other fluid.

Substituting for h in (1) $v = \sqrt{\frac{2p}{\rho}}$.

For purposes of air-current measurements in mines the simplest apparatus is the Darcy arrangement, or combination of Pitot tube A with straight tube B by means of a U tube ab (Figs. 21 and 22).

Although many engineers regard the Pitot tube as indicating with accuracy a certain height of dynamic pressure h estimated in terms of the column of fluid experimented upon, others do not accept it as accurate

[1] That is h represents the difference between the height of the total pressure measured in the Pitot tube and the height of the static pressure measured in the straight tube.

without multiplying by a factor a, the value of which is approximately 1·15, so that $h = a\frac{v^2}{2g}$. Perhaps, as Rateau[1] points out, "both may be right in particular cases, but not generally, the coefficient a being variable, and even widely variable, according to circumstances."

The Water-gauge.—The water-gauge is the instrument commonly used in mines for determining the amount of, and variations in, the difference of ventilating

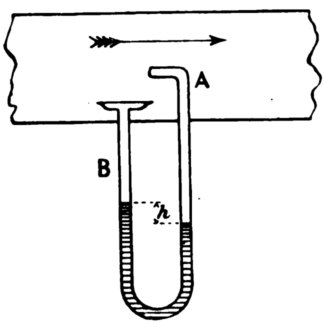


FIG. 21.—Pitot Tube (A) combined with U Tube for Measuring Velocities of Gas, Air, or Steam in Pipes.

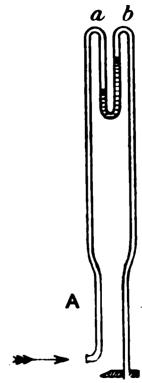


FIG. 22.—Darcy Arrangement. Pitot Tube (A) combined with Straight Tube (B) through the medium of a U Tube (*ab*) for Measuring Velocities of Air-currents in Mines.

pressure as between any two air-ways (intake and return air-ways). Thus, if there is a reading of, say, 3 inches of water-gauge between the bottom of the downcast and upcast shafts, this is indicative of a ventilating pressure of $(5·2 \times 3) = 15·6$ lbs. per square foot of cross-sectional area of the roadway—that is, a pressure of 15·6 lbs. per square foot has been necessary to overcome the frictional resistance of the mine workings.

[1] *Annales des Mines*, 1898, series 9, vol. xiii. p. 331.

The water-gauge, in its simplest form, consists of a U-shaped glass tube, one of the two ends of which is open to the intake air, and the other to the return air of the mine, and the tube being partially filled with water, the difference of level of the same in the two branches indicates the difference in pressure. Thus, as a cubic foot of water may be taken to weigh 62½ lbs. (actually at 60° F. a cubic foot of pure water weighs 62·33 lbs.), a square foot of water 1 inch deep will weigh 5·2 lbs.,[1] so that 1 inch of water-gauge indicates a difference in pressure of 5·2 lbs. per square foot of cross-sectional area of roadway.

There is a difficulty in respect of the practical use of such a simple form of water-gauge as that described above, in that there is a constant oscillation of the water, due to the force of impact of the air on the water surface, and to slight variations in pressure. Various means have from time to time been devised to overcome this difficulty, and to design an instrument which shall give a steady and reliable reading, such, for instance, as contracting the bend of the glass tube, and so narrowing this passage for the water, and by bending the end exposed to the intake air and diminishing the orifice (see Fig. 23), these precautions have undoubtedly imparted steadiness to the water columns and secured a reduction in oscillation. But perhaps the most approved form of water-gauge for common use in mines is that illustrated in Fig. 24, the special features of which are as follows:—The tubes, instead of being the old-fashioned U shape, consist of two

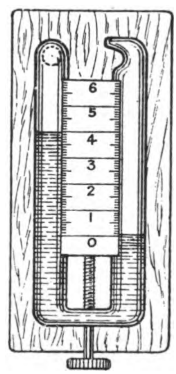


FIG. 23.—Water-gauge.

[1] According to Act of Parliament, 5 Geo. IV. c. 74, a cubic inch of pure water at 62° F. weighs 252·458 grains.

separate pieces of straight boiler-gauge glass, the lower ends, *a*, *a*, of which are tightly fitted into a brass block, the water columns, in the glass tubes, connecting through the block by a small hole in a tap (*b*), so that a considerable contraction is provided in the connecting piece; the tap, also, can be turned off, thus registering the height of the water column, and the water-gauge brought into a good light to be read. The top of the intake air side is covered by a perforated brass cap, the other end is bent and fitted with a cone-shaped brass tube to allow of a tight fitting when inserted into a hole in a wooden door, or being fastened to india-rubber tubing. The scale being behind the tubes renders more accurate reading possible than if it were placed in front. This scale can be raised or lowered by means of the screw (*c*), the whole is fixed to a wooden frame (*d*), which is provided with a spirit-level (*e*) to ensure the instrument being placed in a truly horizontal position when taking a reading.

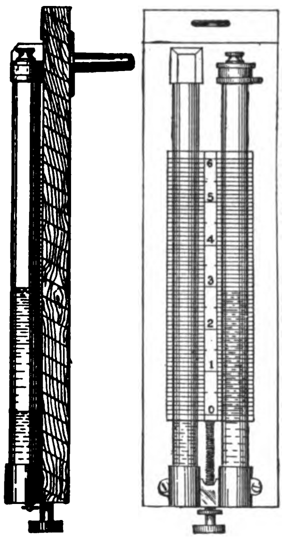


FIG. 24.—The Davis Water-gauge.

For determining very delicate variations in pressures and in laboratory experimental work, where great accuracy may be necessary, the author has used the arrangement shown in Fig. 25, in which a long piece of glass tubing B is bent into the U shape, and fixed in an inclined position, one end being connected to the return air-way or fan drift, as the case may be, through a tightly stoppered vessel A by means of india-rubber tubing C, the object of this vessel being to ensure steadiness in the water column. The difference in the reading a and b represents

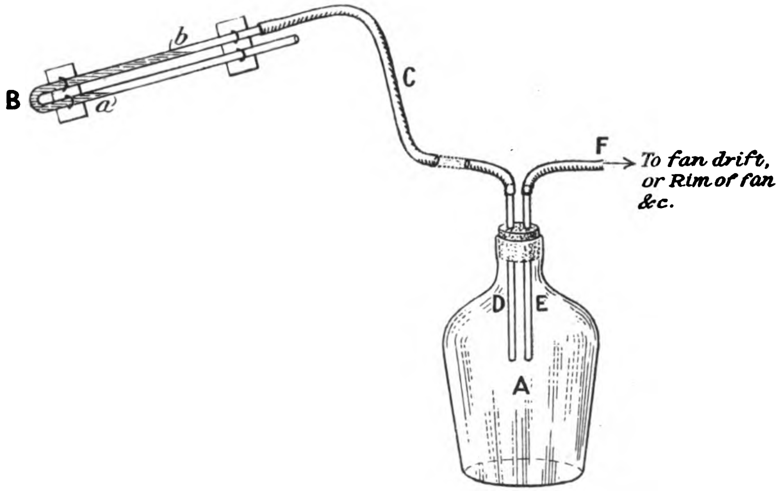


FIG. 25.—Apparatus for Determining Small Variations in the Difference of Pressure when Great Accuracy is required.

the apparent water-gauge, from which the true height of water-gauge can be calculated thus:—

Supposing the angle made by joining the point reached by the level of the liquid in the upper branch of the tube with that reached by the liquid in the lower branch of the tube is 30°, viz. the angle ABC (see Fig. 26), and the length AB is the apparent water-gauge. The true reading will be the length AC, and AC=AB × Tan CBA. The length AB being determined from the scale reading, this being 2 inches in the above figure.

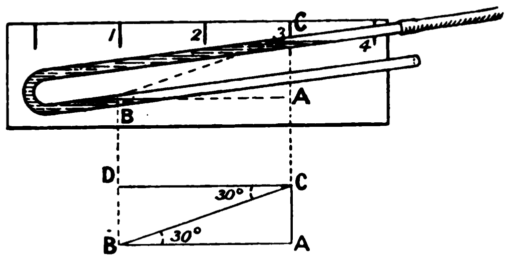


FIG. 26.—Diagrammatic Representation of the Method of Calculating the True Height of Water-gauge when measured by Inclined Tubing.

CHAPTER VII

THE CREATION OF THE VENTILATING CURRENT: VENTILATION BY FURNACE, FANS, ETC.

Natural Ventilation—Ventilation by Furnace—Determination of the Motive Column—Horse-power of the Ventilating Current—Types of Furnaces—Steam-jet and Waterfall—Mechanical Ventilation—Types of Fans—Calculations in respect of Fans.

Methods of Ventilation.—Ventilation of mines may be natural or artificial; natural ventilation being that due to the difference of temperature between the air in the mine and the air at the surface; artificial ventilation being obtained either by drawing the air from the mine by means of furnaces, steam jets, or exhausting fans, or forcing the air into the mine by means of fans or air-pumps. Furnaces and exhaust fans are, however, the chief means of artificial ventilation in practical use.

Natural Ventilation.—Natural ventilation, as the sole means of ventilation, is chiefly restricted to metalliferous mines. As it depends on the variation of temperature—hence the difference in density of the mine air as compared with that of the surface air—it is obvious that the direction of the air-currents in the mine will be subject to variation, the direction being largely dependent on the seasons. The humidity of the external atmosphere is also, to some extent, a determining factor. Dry air entering a wet shaft will absorb moisture quickly, become cooled, and, owing to the resulting increase in weight, will descend. The nature of the climate, whether normally dry or humid, is therefore of some importance in respect of the matter of ventilation, more especially when this is not augmented by artificial means. In a dry climate the natural ventilation will be better than in one normally humid.

In many metalliferous mines, especially in those where the lode worked is of a hard and crystalline character, necessitating the use of power drills, the natural ventilation of the mine is augmented by the exhaust from the compressed-air driven rock-drills in the headings, stopes, and cross cuts, but the total effect of this augmentation on the mine does not amount to much, and is almost entirely local in so far as regards useful effect.

Ventilation by Furnace.—The area of coal-mines, owing to the comparatively flat character of the stratification of the coal in the great majority of cases, is much greater than that of the majority of metalliferous mines. This, and the further fact that coal-mines generate noxious gases to a far greater extent than non-carboniferous deposits, led to the necessity, as mines grew in extent, of introducing artificial means of producing air-currents, and the obvious development was in the direction of augmenting the natural ventilation by applying heat to the return air-current in the upcast shaft, and, besides increasing the amount of the current, of giving it a positive and permanent direction in the mine. For many years this continued to be almost the sole means of artificial ventilation in the coal-mines of Great Britain —if we except the steam-jet and waterfall, of which more anon—and is still adopted with success in some extensive mines, though largely superseded by mechanically driven fans.

The Motive Column.—The power of the furnace, then, is due to the difference in the weight of two equal columns of air of the height of the upcast shaft (the reason for taking the depth of the *upcast* shaft as the measure of the height of the two columns is explained on p. 109), and the measure of the power is the height of a column in the downcast shaft, which would be equal in weight to the difference of the weight of the two columns.

To calculate this it will be necessary to determine the average temperature of the air in the upcast shaft, and the *average* temperature of a column of air of equal height in the downcast, and calculate the respective weights of the two columns. Supposing them to be 1 foot square (see p. 109), the difference equals the weight of the motive column, and the *height* of this can then be found in terms of the downcast air, or, briefly, the following formula may be used :—

Let M = the height of the motive column in feet of air of the temperature of the upcast shaft.

T = the temperature of the upcast air in degrees Fahrenheit.

t = the temperature of the downcast air in degrees Fahrenheit.

D = the depth of the upcast shaft in feet.

Then $M = D\,\frac{T - t}{T + 459}$[1]

Supposing the depth of the upcast shaft is 846 feet,

[1] The calculation can be expressed in another form, viz.—

$$h = D\,\frac{T - t}{Tt}$$

Where h = the pressure due to the weight of the motive column.

D = depth of the upcast shaft in feet.

T = the absolute temperature of the air in the upcast shaft in Fahrenheit degrees.

t = the absolute temperature of the air in the downcast shaft in Fahrenheit degrees.

And the truth of this formula can be demonstrated thus :—

and the temperature of the upcast air is 150° F., and that of the downcast air 50° F.

$$\text{Then } M = 846 \frac{150^\circ - 50^\circ}{150^\circ + 459^\circ} = 13{\cdot}9 \text{ feet.}$$

Having determined the height of the motive column it is possible to arrive at the theoretical velocity of the air-current in the upcast shaft. For the velocity of the air is the same as that of a body falling through a height

If t = the absolute temperature of the air in the downcast shaft, which in the Fahrenheit scale is taken at 492·6° below freezing point (32°), (*e.g.* if the temperature were 50° F. as above, this would be $492{\cdot}6^\circ + 50^\circ = 542{\cdot}6^\circ$).

T = the absolute temperature of the air in the upcast shaft.

D = the height in feet of the two columns of air = the depth of the upcast shaft.

Then, the barometric pressure being equal, the volume of the air varies proportionately to its *absolute* temperature, so that were the column of air in the upcast shaft cooled it would shrink and become $D \frac{t}{T}$. Hence the downcast column contains more air than is required to balance the upcast column, exceeding the amount in the latter by $D \frac{T - t}{T}$.

$$\text{Hence } M = D \frac{T - t}{T}.$$

Or thus:—

We know (see p. 26) that $W = D \frac{1{\cdot}3258 \times B}{459 + t}$ = weight of a column of air having a base 1 foot square and depth temperature of the downcast shaft.

$W' = D \frac{1{\cdot}3258 \times B}{459 + T}$ = weight of a column of air having a base of 1 foot square and of a depth and temperature of the upcast shaft in which t and T are the *observed* temperatures in the downcast and upcast shafts respectively.

$$\text{For } M = D \frac{1{\cdot}3258 \times B}{459 + t} - D \frac{1{\cdot}3258 \times B}{459 + T}$$

$$= D \frac{(1{\cdot}3258 \times B) \overbrace{(459 + T)}^{T} - (1{\cdot}3258 \times B) \overbrace{(459 + t)}^{t}}{\underbrace{(459 + t)}_{T} \underbrace{(459 + T)}_{t}}$$

or representing $(459 + T)$ by T

and $(459 + t)$ by t

and bringing the result to the temperature of the downcast by dividing by the weight of a cubic foot

$$\frac{1{\cdot}3258 \times B(T - t)D}{Tt} \times \frac{t}{1{\cdot}3258 \times B}$$

$$\text{we have } M = \frac{(T - t)}{T} D$$

equal to the height of the motive column, which is calculated by the formula—

$$v^2 = 2\,gh,$$

or in a reduced form $v = 8\sqrt{h}$

Where v = the velocity in feet per second

h = the height in feet through which the body falls

and g = the force of gravity or 32·2.

Or taking the figure in the example above—

$$v = 8\sqrt{13 \cdot 9} = 29 \cdot 76 \text{ feet per second.}$$

The power due to the motive column can be expressed in inches of water-gauge. Thus—

One cubic inch of water weighs 0·036 lb., and 1 inch of water-gauge represents a pressure of 5·184 lbs. per square foot. One square foot of water 1 inch deep weighing 5·184 lbs., and 1 cubic foot of air at 60° F. weighs 0·0766 lb.

$$\text{Therefore } \frac{M \times 0 \cdot 0766}{5 \cdot 184} = \text{inches of water-gauge.}$$

The horse-power of the air-current can be calculated, if we know the quantity of air, and the height of the motive column.

$$\text{H.P. of current} = \frac{Q \times M \times 0 \cdot 0766}{33{,}000}$$

In calculating the height or weight of the motive column, the point of chief importance is the determination of the vertical height of the upcast. The following examples will make this clear:—

Taking in the first instance the case of a horizontal seam and an uneven earth surface, as in Figs. 27 and 28. In Fig. 27 the downcast shaft AB is 400 yards in depth, and the upcast shaft 200 yards, but the first 200 yards in depth (reckoning from the surface) of the downcast balances the hypothetical part of the upcast column, viz. DE, the line *aa′* marking the level of equal atmospheric pressure. In Fig. 28 the horizon of equal atmospheric pressure is again determined by the level of the top of the upcast.[1]

In the case of horizontal deposits, therefore, the weight of the motive column is determined by the loss of weight

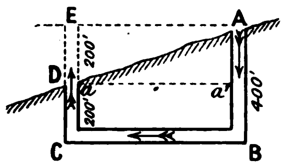


FIG. 27.

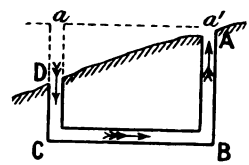


FIG. 28.

FIGS. 27 and 28.—Diagrammatic Representation of Shafts of Unequal Depth sunk to a Horizontal Seam.

as between two imaginary columns of air, of the height of the atmospheric envelope, due to the expansion of the air in one part of one of the columns, or, put in another way, by the difference of weight in respect of two columns of air measured by the height of the upcast shaft.

Supposing the seam being worked is inclined, and suppose the surface is level, as in Figs. 29A and 29B. In Fig. 29A the downcast shaft AB is 400 yards in depth, and the upcast shaft CD is 200 yards deep, and the line *aa*[1] marks the level of equal atmospheric pressure. It might be thought necessary in this case to suppose the upcast shaft as commencing from the level of B, the deepest point in the mine, and that at which the atmospheric pressure is greatest, and that the average tempera-

[1] This is not strictly true, as there will be some slight correction to make on account of alteration of density of the downcast air due to friction and probable slight increase of temperature.

ture of the upcast should be that calculated as between the points B and D, the artificially heated air being that contained in the column CD, the furnace being situated at C. But instead of taking DE as the depth of the upcast, the correct method of estimating the ventilating power would be to divide the calculation, and first calculate the height of the motive column from the depth of the upcast shaft DC, and, as a separate calculation, determine the value of the natural ventilation for the column a'B being colder than the hypothetical column EC (which will have the temperature of the air in BC),

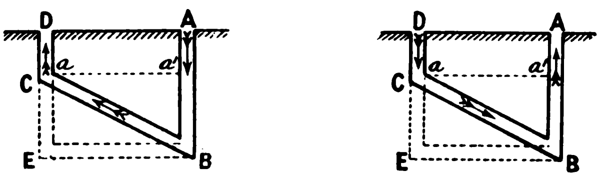


FIG. 29A. FIG. 29B.

FIGS. 29A and 29B.—Diagrammatic Representation of Shafts sunk from a Horizontal Surface to an Inclined Seam.

will more than balance it, as the air, after leaving the point B, has traversed the workings, and extracted much of the rock temperature.

In Fig. 29B, AB is the upcast, but in this case, seeing that the hypothetical downcast is DE (and in calculating the motive column it is supposed that the temperature of the downcast air is that of the surface air), it will be necessary to correct the calculation by deducting therefrom the effect of the natural ventilation, due to the temperature of the mine, or of taking as the temperature of the downcast air the average between D and B.

It will be seen from the above examples why, when ventilation of mines by furnace was more common than at present, it was often deemed advisable to increase the depth of the upcast shaft by the erection on the surface of a chimney.

As in calculating the height or pressure of the motive column a unit of 1 square foot is adopted, we are not concerned for the moment as to the relative sizes of the two shafts.

The pressure increases as the depth of the upcast shaft—provided the difference in temperature between the two shafts be maintained—and the ventilation as the square root of the depth of the upcast, so that by adding a stack of, say, 30 feet to the top of an upcast shaft 900 feet deep, the ventilation capabilities would be increased in the proportion—

$$\sqrt{930} : \sqrt{900} = \sqrt{\frac{930}{900}} \text{ or } 1\cdot016 \text{ times}$$

or thereabout.

The Furnace.—When collieries were worked by single shafts they were divided by a partition (bratticing) into a downcast and upcast side, and a furnace placed at the bottom of the upcast side, or at the bottom of a cupola or chimney connected with the upcast side, and this latter arrangement is still sometimes adopted to ventilate a sinking shaft (Fig. 30), though undoubtedly for this purpose a small forcing fan is preferable (see p. 163, vol. ii., *Sinking of Shafts*).

However, the furnace is now, where used in permanent ventilation, placed at the bottom of the upcast shaft, and Figs. 31, 32, and 33 show respectively in plan, front elevation, and section the extensive ventilating furnace, and its position in respect of the upcast shaft, which was constructed a number of years ago at a large colliery in the North of England. This furnace put 271,356 cubic feet of air per minute into circulation at a water-gauge of 2·6 inches, and consumed on the average 16 tons 6 cwts. of coal in the twenty-four hours.[1]

It will be observed that the furnace was surrounded by a brick-arched travelling-way, so that at no point was it

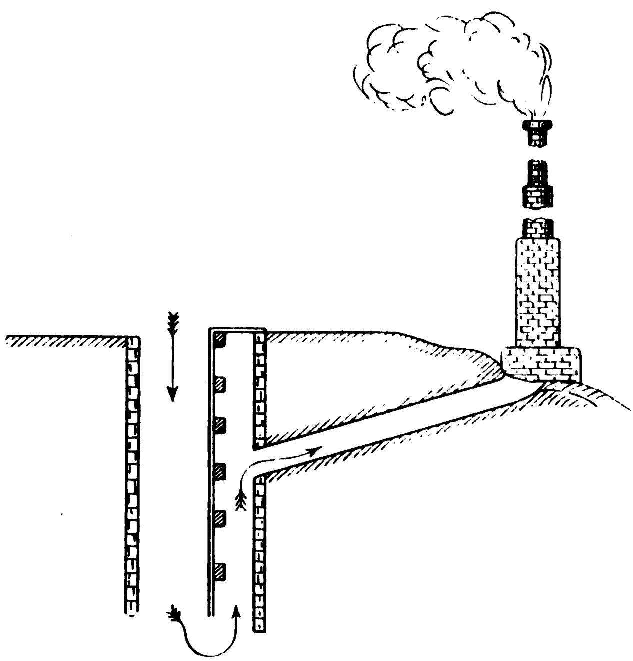


FIG. 30.—An Old Method of Ventilating a Mine or Sinking Pit by means of a Furnace placed at the Surface.

in contact with the coal sides, the lateral arms of this road being 8 feet wide, and the cross connecting road under the furnace drift 6 feet wide by 6 feet high. There was a travelling road on either side of the furnace drift, and also round the shaft at the point where the drift entered it. The drift had an inclination of 1 in 2, and was 12 feet wide

[1] The author has observed that furnaces require more firing in summer than in winter in order to maintain the same current, the reason for which is obvious.

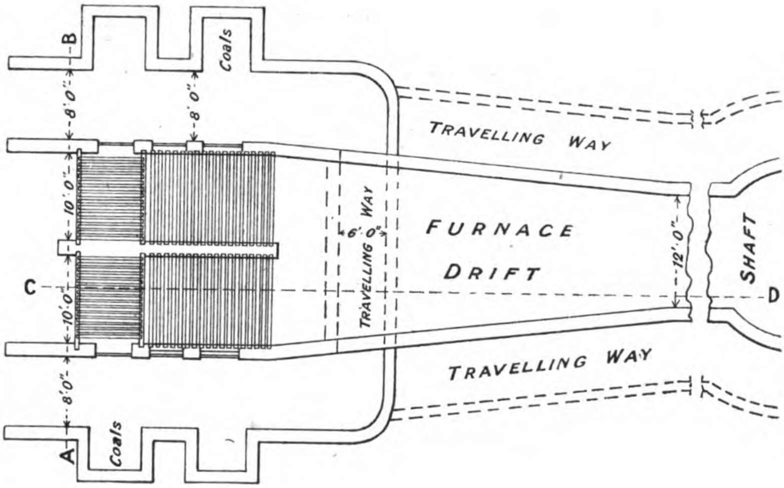


FIG. 31.—Plan of Ventilating Furnace.

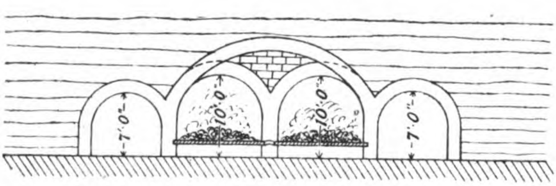


FIG. 32.—Front Elevation of a Ventilating Furnace showing Travelling Roads (Air-spaces) on either side.

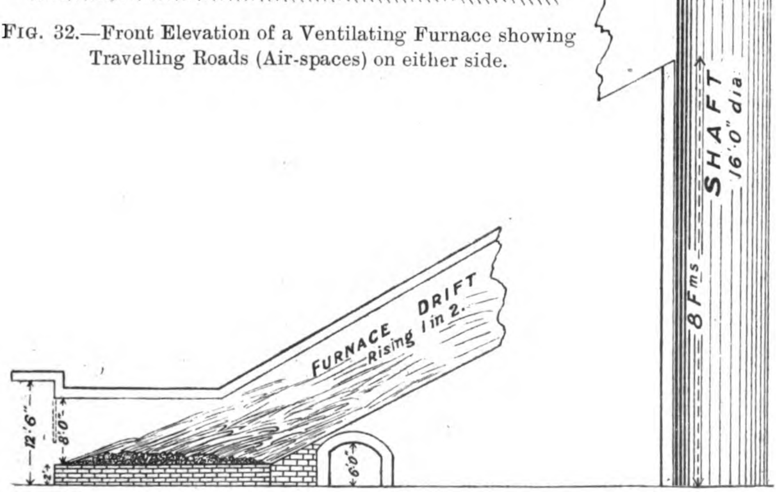


FIG. 33.—Section of Furnace, Furnace Drift and Shaft.

at the point at which it entered the shaft, the perpendicular height of the drift gradually increasing from 8 feet to 12 feet, the increased area being for the purpose of allowing of the expanded condition of the heated air and gases and conducing to increased draught. The width of the road in front of the furnace was 20 feet 6 inches and its height 12 feet 6 inches. This may be taken as illustrating an almost ideal furnace arrangement, but that it was found necessary ultimately to remove the middle partition, as the iron expanded to such an extent as to injure the brickwork, and to have the furnace with an open face, and with one instead of two fire-grates.[1]

Cost of Constructing Furnace.—The cost of making the ventilating furnace illustrated and described above was as follows :—

Building—						
Fire bricks	48,000 @ 44/		=	£105	6	8
Common bricks	72,000 @ 36/6		=	131	8	0
Plate bricks	106,000 @ 34/		=	180	4	0
Fireclay	16 tons @ 10/		=	8	0	0
Fireclay lumps	7615 @ 1/		=	380	15	0
Do.	26 tons @ 24/		=	31	4	0
Labour				910	13	8
Stone Work—						
Labour				626	0	0
Castings—						
	Cwts. Qrs. Lbs.					
11 bearing bars	96 1 0	@ 3/9	=	18	0	11
83 engine bars	148 0 26	@ 3/9	=	27	15	10
	Total cost			£2419	6	5

The late Mr. T. E. Forster, in giving evidence on the subject of ventilation before a Committee of the Lords in 1849, stated that the cost of erecting a furnace capable of

[1] The mine is now ventilated by a fan.

producing a current of 48,760 cubic feet of air per minute would be £212, and that the annual expense of working the same would be £267, 10s.

The Application of the Furnace to "Fiery" (Gassy) Mines.—In some mines the return air is liable to become dangerously vitiated with firedamp by reason of the occurrence of sudden outbursts or blowers of firedamp, or by the existence of large goaves (wastes), which on a sudden fall in the atmospheric pressure will discharge large volumes of "gas" into the return air-ways—especi-

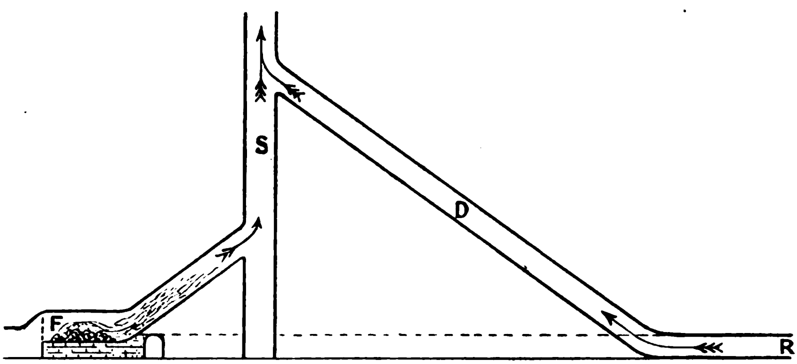


FIG. 34.—Illustrating the Use of the "Dumb" Drift, the Furnace being Fed with Fresh Air.

ally if such goaves are of an "open" character. In such cases if furnace ventilation is resorted to it is eminently desirable that the fire should be fed with intake air, instead of the return air of the mine, the latter being conducted by means of a "dumb" drift to a point in the upcast shaft well above the reach of the flame of the furnace. This arrangement is sketched in Fig. 34, in which F is the furnace, and the return air-current is shown as carried into the shaft S by means of the dumb drift D from the main return R.

With, however, the exception of Wallsend Colliery, the author can call to mind no case of an explosion having occurred at a mine furnace, which non-occurrence may perhaps be explained by the fact that in all probability, if the return air-current is a strong one, there is not the same tendency for the flash of the ignited mixture of gas and air to pass far back into the return air road. The mine would have to be very foul indeed to allow of the air when it reached the furnace being in an explosive condition. The late Mr. Mathias Dunn, in an unpublished diary, alludes on one occasion to the gas "flashing" as the air passed over the furnace at Felling Colliery.

Undoubtedly the best method of inducing ventilation in the case of a "fiery" colliery is by means of a fan, for in the event of a colliery explosion wrecking the mine, the re-establishment of the ventilation would be a slow and dangerous undertaking were it to depend on a furnace; not only might the furnace be wrecked, but to light a fire might be a matter of considerable risk; whereas the fan being at the surface, and some distance back from the top of the shaft, is usually very little or not at all injured by the force of the explosion, and can be put into operation after but little delay if the fan-drift has been injured.

Ventilating Efficiency of the Furnace.—An instance of the ventilating power of a furnace has already been quoted (p. 112), which was equivalent to 111 horse-power, but this must be regarded as an extraordinary case. The power of the average mine ventilating furnace will be considerably below this figure.[1]

[1] Although so-called refuse coal was burned in the case quoted, it cannot be said that this coal was unsaleable, hence the cost of working the furnace was greater than appeared.

There is a limit to the efficiency of furnace ventilation, because of the expansion of the air in the upcast shaft due to heat, and if this is carried too far the consequent velocity of the current in the shaft will so augment the frictional resistance that a point may be reached when the greater the heat applied to the upcast air, the less will be the air circulated in the mine. Péclet states that when the upcast air has been expanded to twice its original volume, the limit of furnace ventilation has been attained.[1]

No more air should pass through the furnace than is required for the combustion of the fuel. Theoretically, about 150 cubic feet of air are necessary for the combustion of 1 lb. of average coal. In practice, however, it takes about twice this amount of air. The actual consumption of fuel in some pit furnaces is given in an interesting paper in vol. vi. of the *Transactions of the North of England Institute of Mining and*

[1] The amount of fuel *theoretically* required for a furnace may be determined thus:—Heat produced by burning 1 lb. of furnace coal = 12,000 heat units. (A heat unit = heat required to raise 1 lb. of water 1° F.)

It takes less heat to raise 1 lb. of air 1° F. than it does to raise 1 lb. of water 1° F.

The relative heat required to raise a body to a temperature of 1° is called its *specific heat.*

The specific heat of water is 1.

The specific heat of air is 0·2374.

Weight of air that 1 lb. of coal will raise 1° F. is as 0·2374 : 1 :: 12,000 : x.

$$x = 50{,}000 \text{ lbs.}$$

Thus to take an example:—

100,000 cubic feet air per minute.

Intake temperature, 50° F. Upcast temperature, 130° F.

Heat required from the furnace = 130 - 50 = 80° F.

100,000 cubic feet at 50° F. weighs 7811 lbs.

$$\therefore \text{ fuel required per minute} = \frac{7811 \times 80}{50{,}000} = 12{\cdot}49 \text{ lbs.}$$

12·49 × 60 = 749·4 lbs. of fuel per hour.

Ventilating pressure is 10·61. Foot pounds, 1,061,000. Horse-power, 32·15.

Mechanical Engineers, from which the following figures are taken :—

TABLE V.—*Particulars as to Fuel Consumption at some Durham Mine Ventilating Furnaces.*

	Depth in Feet.	Coal in lbs. per H.P. Utilised per Hour, excluding Power due to Heat of Mine.	Coal in lbs. per H.P. Utilised per Hour, including Power due to Heat of Mine.	Quantity of Air per Minute in Cubic Feet.
Thornley $\frac{5}{4}$ Seam, Co. Durham .	556	85·5	37·5	45,756
Thornley, Hutton Seam, Co. Durham	868	162·4	57·1	26,574
Walker Colliery, Northumberland	960	30·5	15·6	44,800
Castle Eden, Co. Durham .	1,038	29·1	28·3	42,326
South Hetton, Co. Durham . .	1,212	27·2	15·5	132,895
Wearmouth, Co. Durham . .	1,800	29·5	7·9	70,500
Average . .	..	60·7	27·6	..

The Steam-jet and the Waterfall.—Mention has been made of the steam-jet and waterfall as means of producing ventilation, but except as temporary measures these need not be seriously considered.

The steam-jet consists merely in discharging a vertical jet of high pressure steam at any point up the shaft. The effect of the discharge is to impel the outflow of the air. The measure of this impulse may be calculated thus :—

V = the volume of the discharged steam in cubic feet.
p = the effective pressure in lbs. per square foot at which the steam is discharged.
U = the units of work in foot pounds that can be obtained from the discharge of the steam without expansion.

$$U = pV.$$

If, however, the jet is placed some distance down the shaft there would have to be added the effect due to the heating of the air column, though against this addition there would have to be put the friction due to increased velocity owing to the presence of the steam.

The steam-jet was first introduced for the ventilation of mines by Mr. John Buddle in the year 1811, as a means of increasing the ventilation at Hebburn Colliery near Newcastle-upon-Tyne, where there was a great discharge of gas, and the furnace could not be used in con-

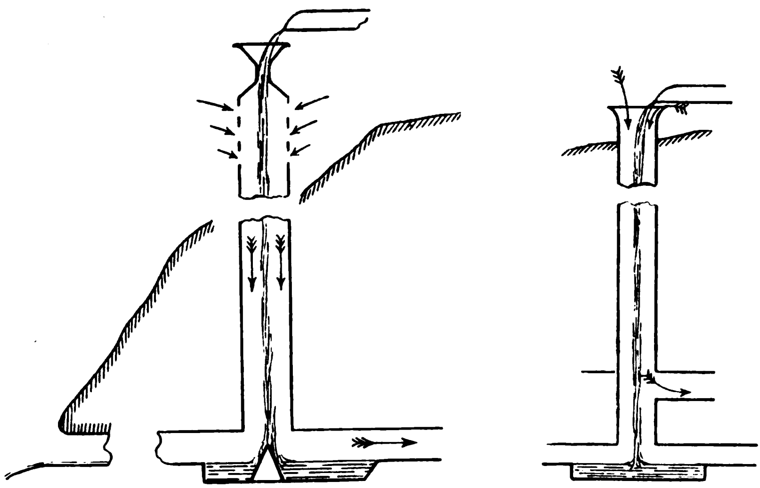


FIG. 35. FIG. 36.

FIGS. 35 and 36.—Diagrammatic Representation of the Application of the "Waterfall" as a means of Ventilating a Mine.

sequence. Another notable instance of the use of the steam-jet was at Seaton Delaval Colliery, Northumberland, where it was introduced by Mr. T. E. Forster in the year 1848. Mr. J. Taylor, who examined the Seaton Delaval steam-jet arrangement came to the conclusion[1] that although the jets with the furnace produced a much larger current than the furnace alone, the consumption of fuel in proportion to the amount of current was

[1] *Working of Collieries*, by Mathias Dunn, 2nd ed., 1852, p. 139.

greater. That is to say, the furnace produces more air than the jets in proportion to the fuel consumed.

The Waterfall (Figs. 35 and 36) is the reverse of the

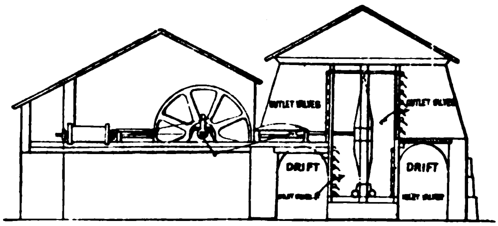


FIG. 37.—The Nixon Ventilator (from Lupton's *Mining*).

steam-jet, in that when used it is employed to draw air down into the mine, or act by compressive, as opposed to exhaustive, ventilation. Unless the mine is so constituted that free drainage is possible (see Fig. 35) the water has to be pumped to the surface again, which

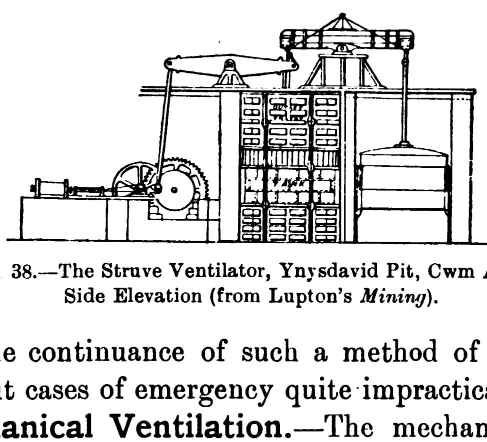


FIG. 38.—The Struve Ventilator, Ynysdavid Pit, Cwm Avon, Side Elevation (from Lupton's *Mining*).

renders the continuance of such a method of ventilation for any but cases of emergency quite impracticable.

Mechanical Ventilation.—The mechanical ventilators used at mines can be classed as follows :—

Mechanical ventilators
- (1) Reciprocating machines
- (2) Rotary machines
  - (*a*) Those acting by displacement.
  - (*b*) Those acting by centrifugal force.

In the first class, viz. reciprocating machines, sometimes also termed varying capacity machines, and usually applied to exhausting the air from the mine, there are Nixon's ventilator (Fig. 37) at Nixon's Navigation (Glamorganshire), the Struve ventilator (Figs. 38 and 39), or the Hartz blower (or duck machine of Cornwall).

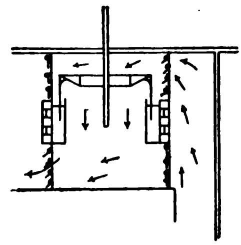


FIG. 39.—Struve Ventilator: Section (from Lupton's *Mining*).

The Hartz Blower (Fig. 40).—This consists, as the figure shows, of two boxes, one moving inside the other; the larger and outer box is nearly filled with water, and through the bottom of both boxes passes a pipe which acts as an exhaust air shaft. The outer box is stationary, whilst the inner box, being suspended from a bar projecting from the pump rods, is lowered and raised with the motion of those rods. At the top of the exhaust pipe is a valve and another at the top of the moving box, both of which open in the same direction; but whilst on the upward motion of the box the pipe valve opens, that of the moving box is closed. On the descent the reverse is the case, and the air drawn out of the pipe in the upward motion is exhausted from the box in its downward motion. By reversing the valve this apparatus can be made to act as a blower.

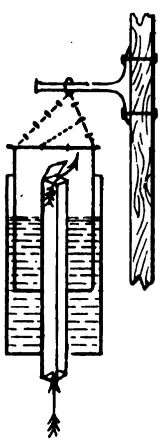


FIG. 40.—Hartz Blower.

Rotary Ventilators: (*a*) *A Displacement Ventilator.*—A rotary air-pump which acts by displacement may be mentioned, as it is sometimes used to ventilate sinking shafts or mines of very small area, viz. the Roots ventilator, blower or exhauster. In this form of ventilator two pistons of similar shape, as shown in *aa*, Fig. 41, are fixed to parallel shafts, and made to rapidly revolve in opposite directions within an iron or steel casing, the air (if the ventilator is acting as an exhauster) being drawn in at the bottom of the casing and expelled at the top. The clearance between the pistons and between the pistons and the side of the casing with large ventilators of this type, is not more than one-eighth of an inch, but as the pistons are kept in position by gearing, there is no danger of their coming in contact one with the other.

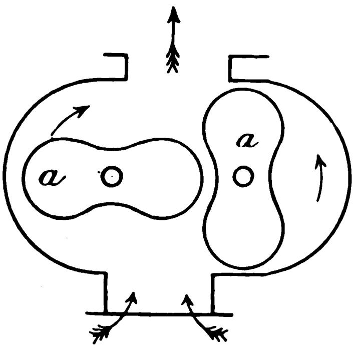


FIG. 41.—The Roots Blower.

(*b*) *Centrifugal Ventilators or Fans.*—This class of machine includes the more important ventilators in use at the present day at mines, where, although they can be applied as forcing fans, they are most commonly used as exhausters. But the fact should not be lost sight of that forcing air into a mine has some advantage over ventilation by exhaustion; the French Commission came to the conclusion that there was a considerable theoretical advantage in respect of the former mode, because it required a little less power, and produced a plenum of pressure which tended to keep back blowers of firedamp and choke-damp, and with a falling barometer the more rapid evolution of gases could be met by increasing the air compression in the mine. Were, however, the forcing system to be adopted, in order to maintain the natural principle of ascensional ventilation, the fan would have to be placed at the deepest shaft, which would usually be that at which the winding is done, whereas an exhausting fan may be placed at any shaft. Further, on the sudden stoppage of a forcing fan the atmospheric pressure in the mine does not increase as is the case on the sudden stoppage of an exhausting fan.[1]

Types of Fans.—Until about thirty years ago there were but three distinct types of ventilating fans in use at the mines of the United Kingdom, viz. the Waddle, the Schiele, and the Guibal. Thus, for instance, when the North of England Institute of Mining and Mechanical Engineers, and the Midland Institute of Mining, Civil, and Mechanical Engineers, in the year 1888, appointed a Committee to experiment and report on mechanical ventilators,[2] these three types only were experimented upon, although when the report was published other types had been in use for some years. The subject of mechanical ventilators might well occupy the attention of one or other of the mining engineering societies, British, American, or continental, with a view to arriving at some conclusions respecting the relative merits of the various types.

The exigencies of space will not allow of the author describing all the types of ventilating fans in use. He proposes confining his remarks to a brief description of the following :—

1. The Waddle.
2. The Schiele.
3. The Guibal.
4. Walker's Fan.
5. Bumstead & Chandler's Fan.
6. The Capell.
7. The Sirocco.
8. Barclay's Fan.
9. The Jeffrey Fan.
10. The Rateau Fan.
11. Parsons Turbo Exhauster.

---

[1] Since 1882 a forcing Guibal fan, 7 metres (22·96 feet) in diameter, has been in operation at the bottom of the Alexander Shaft at Von Arnim's Colliery (Upper Silesia), the object being to keep back the large quantities of choke-damp given off from the workings. The results are said to have been favourable. Be this as it may, "forcing" ventilation has been but little used in the district.

[2] "Mechanical Ventilators," by M. Walton Brown, *Trans. Inst. M.E.*, vol. xvii. pp. 482–576.

The Waddle Ventilator.—The first Waddle fan was erected over forty years ago at Bonville's Court Colliery in Pembrokeshire, and although of somewhat primitive construction, has been running regularly ever since, is still in good condition, and has cost little in repairs, although run at a speed of 100 revolutions per

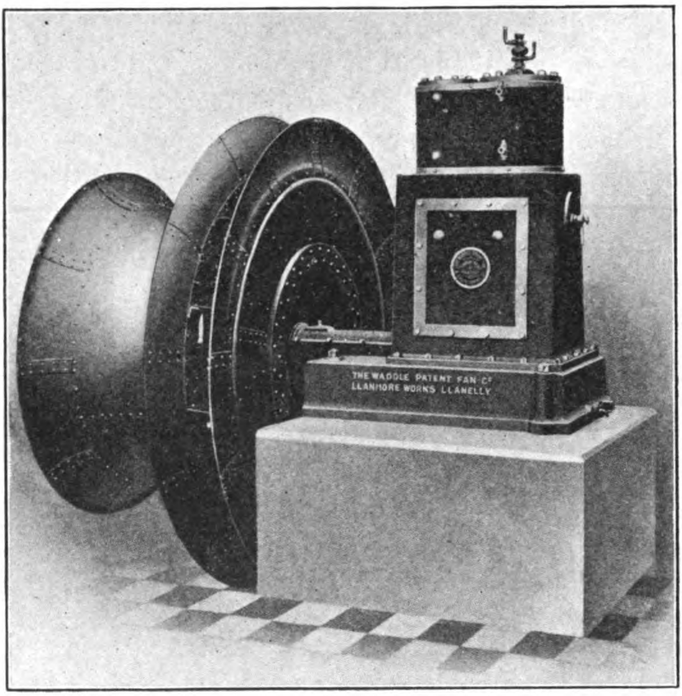


FIG. 42.—Self-contained Waddle Fan and Vertical Engine.

minute—facts which argue efficient design, good construction, and sound erection.

The Waddle fan as a whole consists of a light hollow disc (see Fig. 42), the two sides of which are braced together by the blades, which are riveted to them, so that instead of the blades revolving within a casing, as is common to enclosed types of fans, in the Waddle the revolving casing carries the blades with it.

The air is drawn in at one side only, a fact which allows of the connection from the upcast shaft being a short straight passage. In the case of double inlet fans bends are necessary in the fan drift, so adding considerably to the cost for masonry, and to some extent acting as a hindrance to the flow of the air.

The periphery is open all round, and the air is discharged equally from every part of it, hence the load

FIG. 43.

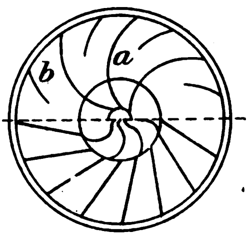


FIG. 44.

FIGS. 43 and 44.—Arrangement of Blades in a Waddle Fan. FIG. 43.—End View. FIG. 44.—Side View. Lower part shows side plates; upper part divested of these to show blades.

on every blade, and on every part of it, is steady and unvarying, so conducing to prevent vibration. On leaving the blade tips the air enters the diffuser or outlet at a greatly reduced velocity, so that the kinetic energy of discharge is utilised within the fan. Originally the blades were made curved and arranged as shown in Figs. 43 and 44, but in recent designs they are radial. In the modern form also the outlet is divergent, that is, the projecting rims of the casing are inclined outward, instead of being straight.

The dimensions of a typical fan as used for ventilating a large colliery are given in Figs. 45 and 46, but fans are made in sizes varying from 3 feet diameter up to 45 feet diameter. They are constructed of steel through-

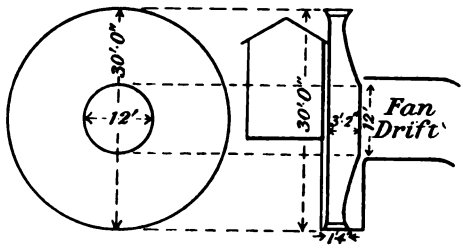


FIG. 45. FIG. 46.

FIGS. 45 and 46.—Showing Dimensions of a Typical Waddle Fan.

out, and are generally direct-driven by means of steam-engine or electric motor, or indirect, through the medium of belt or rope, by means of an electric motor. The comparative arrangement of the fan and engine and fan

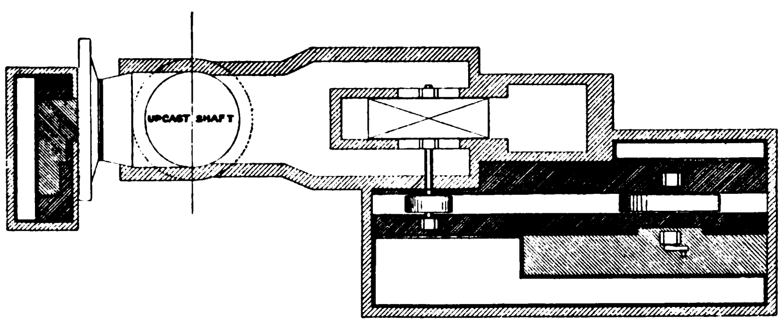


FIG. 47.—Comparative Diagram showing Masonry required for a Waddle Self-contained Fan and Engine, and Belt or Rope Driven Fan and Engine of equal capacity.

drift in respect of a self-contained direct-driven fan and of a rope-driven fan is shown in plan, Fig. 47.

Fig. 48 shows a Waddle fan direct-driven by a horizontal engine.

TABLE VI.—*Particulars relative to various Standard Sizes of Waddle Fans.*

Diameter in Feet.	Diameter and Stroke of Engine.	Volume in Cubic Feet per Minute.	Water-gauge in Inches.	Average Quantity of Masonry in Complete Foundations and Building.
				Cubic Yards.
4	6 × 4	35,000	2	8
6	9 × 7	60,000	2	12
7½	10 × 7	80,000	2	14
9	12 × 10	100,000	3	20
10	14 × 10	120,000	3	22
12	14 × 12	145,000	3	35
15	15 × 15	180,000	3	50
18	18 × 18	250,000	3	70
21	20 × 21	320,000	3½	95
25	24 × 24	450,000	4	130

Efficiency.—The following are the tabulated results which have been obtained from some of these fans :—

TABLE VII.—*Results obtained from a 6 feet diameter Self-contained Waddle Fan driven by a 9 by 7 in. Steam Engine, with varying "Equivalent Orifices."*

Equivalent Orifice.	Revolutions.	Volume.	Water-gauge.	Mean Pressure.	I.H.P.	Air H.P.	Useful Effect.
8·0 sq. ft.	380	35,013	2·7	22·5	19·0	14·9	78·4
13·0 sq. ft.	355	51,623	2·1	27·0	21·3	17·1	80·3
21·5 sq. ft.	364	69,886	1·45	32·3	26·1	16·0	61·3

TABLE VIII.—*Particulars as to Standard Sizes of Electrically-driven Waddle Fans.*

Diameter of Fan.	Volume in Cubic Feet per Minute.	Water-gauge.	Revolutions for Volume and Maximum Water-gauge mentioned.	B.H.P. of Motor for 2 in. Water-gauge.
Feet.		In.		
3	25,000	1–4	970	12
4	40,000	1–4	730	18
6	60,000	1–4	490	28
7½	85,000	1–4	390	38
9	120,000	2–6	385	54
10	150,000	2–6	350	65
12	200,000	2–6	290	90
15	300,000	2–6	240	130
18	400,000	2–6	200	180

The practice is now to make the fans self-contained as regards fan and engine, and of smaller diameter, and to drive them at higher velocities. The old 40 feet diameter fans were not so economical as the newer design, as the following results given by the two fans at Morfa Colliery in Glamorganshire show. Both fans were on the same upcast shaft, so that the conditions which they had to meet were precisely the same in each case, and both fans were direct-driven in accordance with the usual practice with the Waddle fan.

RESULTS GIVEN BY FANS AT MORFA COLLIERY ON SAME UPCAST SHAFT.

TABLE IX.—40 *feet Fan and Engine of Old Make.*

Revolutions.	Volume in Cubic Feet per Minute.	Water-gauge in Fan Drift.
53	61,000	2·5 inches.

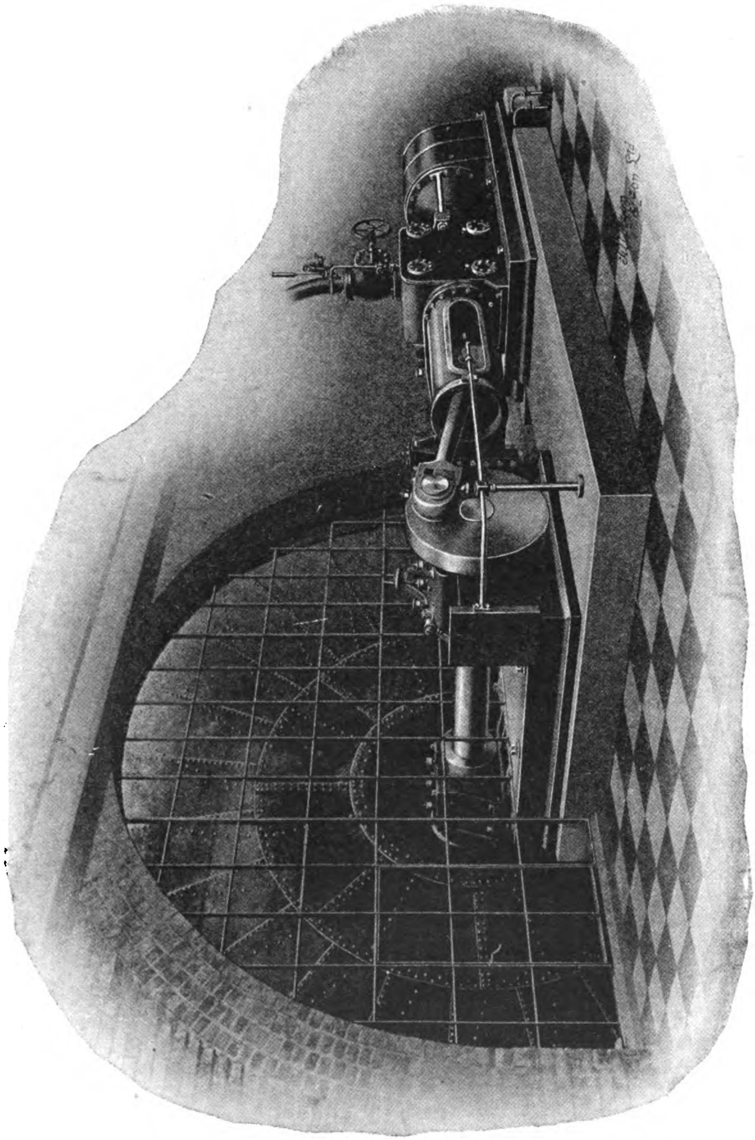


FIG. 48.—Direct-driven Waddle Fan.

TABLE X.—21 *feet Improved Self-contained Fan and Engine.*

Revolutions.	Volume in Cubic Feet per Minute.	Water-gauge in Large Chamber over Upcast 20 feet from Fan.
136	88,700	5·2 inches.
153	97,200	6·2 „
160	108,500	6·7 „
168	110,200	7·4 „

The following results were obtained from a fan at Houghton Colliery, near Durham.

The fan was 21 feet diameter, of the self-contained fan and engine type.

TABLE XI.—*Results given by a Waddle Fan at Houghton Colliery, Co. Durham.*

Revolutions.	Volume.	Water-gauge.
106	272,000	1·6 inches.
116	303,000	2·2 „

A very fine result is that obtained with a 35 feet fan at the Cambrian Collieries, Clydach Vale, in South Wales, which is driven by one tandem compound engine with 22-inch and 36-inch cylinders, and 3 feet stroke. This fan, under ordinary work, gives a volume of over 400,000 cubic feet of air per minute, with a water-gauge of 4½ inches.

Cost.—To give some indication of the cost of a fan and engine of the Waddle type, the following case may be quoted. A fan (old type) 31·6 feet in diameter and 14 inches wide at the tips of the blades, with an inlet of 14 feet 1 inch diameter, and driven by a non-condensing horizontal engine with one cylinder 24 inches by 24 inches. Cost for fan and buildings, £940. The cost of upkeep (stores) of a well-erected fan should not exceed £25 per annum.

The Schiele Ventilator.—The Schiele (Fig. 49) is a small diameter, high-velocity, encased fan, the casing being of masonry. The blades, which are curved, are made of iron or steel, and are not of the same width throughout, being widest in the middle and decreasing towards both extremities. It has double inlets, and the chimney is of the *évasée* or expanding type (see p. 133). It will be seen that the fan is not in the centre of the casing, but is placed eccentrically. It is made in sizes of from 5 to 20 feet in diameter. The smaller sizes are usually driven at a velocity of about 500 feet per minute, and the larger sizes at 110 feet per minute.

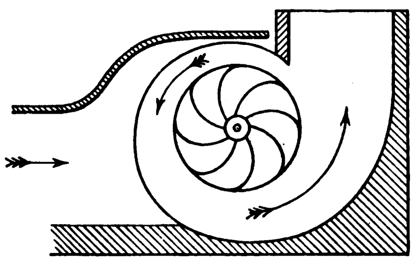


FIG. 49.—The Schiele Fan.

The following are the particulars of one of these fans at Wingate Grange Colliery, near Durham :—

	Feet.	Inches.
Diameter	12	0
Width at inlet	3	4
Width at periphery	2	0
Diameter of inlet	7	3

This fan is driven from a single-cylinder horizontal engine 25 inches in diameter, with a 24-inch stroke, by rope gearing, in the ratio of 2·6 to 1. The fan and buildings cost £1423, 12s. 4d. The annual cost in stores is £38, 1s. 11d., and of repairs about £4, 3s. 6d.

The Guibal Fan.—The Guibal fan was designed by the late M. Guibal, an eminent Belgian mining engineer. In this fan the rectangular blades or vanes, of which there are eight to ten, and which are constructed of wood, are not arranged radially, but in the manner shown in Fig. 50—that is, they are bolted to arms which are fixed to an octagonal cast-iron boss. The blades have about ½ inch clearance with the side casing, and 1 to 2 inches with the top or arch. This was the first fan to have the expanding chimney or *évasée* (*b*), which lessens the velocity of the air as it travels to the point of discharge, and to be equipped with the sliding shutter (*a*), which acts as a regulator. If the shutter opening is too large, eddies are formed within the fan space, but by careful regulation of the shutter the size of passage best suited to the requirements of a fan in respect of velocity and mine conditions can be obtained. The original form of shutter, having a straight-cut edge, had the effect of producing a certain amount of shock, but the Walker shutter, which is now adopted, being formed like an inverted V, allows of a gradual instead of a sudden change as each fan blade passes the enclosed part of the casing. The degree of opening which gives the best effect for a given case must, of course, be determined by experiment.

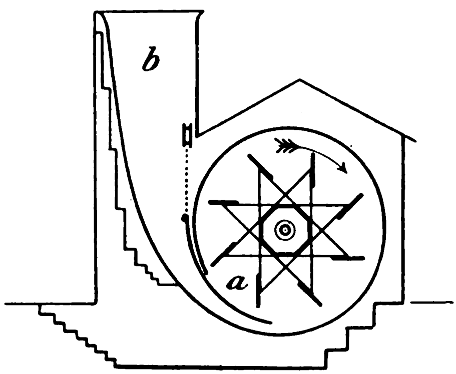


FIG. 50.—Guibal Fan.

The dimensions of a typical Guibal fan would be 36 feet in diameter, 12 feet wide, and diameter of inlet 13 feet. It is a single-inlet fan, and the buildings are arranged somewhat in the manner indicated in Figs. 51 and 52.

The Use of the Expanding Chimney.—The effect of the expanding chimney is such that the velocity of the air, by the time it reaches the outer atmosphere, is reduced to one-quarter or one-fifth of its original value,

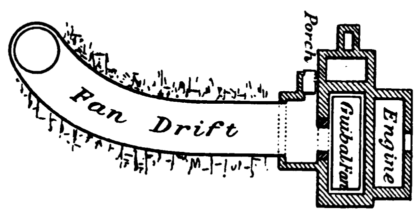


FIG. 51.—Plan.

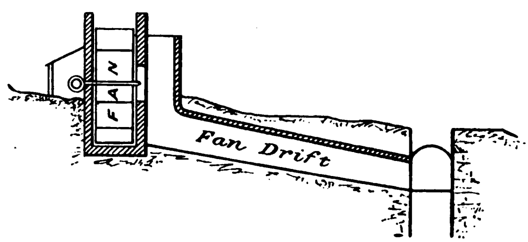


FIG. 52.—Section.

FIGS. 51 and 52.—Plan and Section showing Arrangement of a Guibal Fan Building in relation to the Upcast Shaft.

and the *vis viva*[1] to one-sixteenth or one-twenty-fifth, conditions which are obviously favourable to the effective utilisation of the force applied in driving the fan.

[1] *Meaning of the term "vis viva."*—The distance through which a body will fall from rest *in vacuo* in one second of time is 32·2 feet $= g$.

Let $v =$ the velocity of body in seconds.

Then $v = gt$.

Let $s =$ the space through which the body falls.

Then $s = \frac{v^2}{2g}$ or $v^2 = 2gs$. And $s = \frac{gt^2}{2}$.

The amount of work stored up in a body of weight w, when moving with a velocity v, is measured by the expression—

$$\frac{wv^2}{2g}.$$

Formerly it was the practice to call this quantity half the *vis viva* of the body, The effect can be demonstrated by a simple calculation :—

Let w = the weight of the air.
v = the velocity at which it is moving.

Then the amount of work produced $W = \frac{wv^2}{2g}$.

Taking two supposititious cases—

(1) Let $w = 10$
$v = 200$.

Then $W = \frac{10 \times 40,000}{64 \cdot 4} = 6211$

(2) But if the velocity be reduced by half—

$$W = \frac{10 \times 100,000}{64 \cdot 4} = 1552$$

The following are the particulars of a Guibal fan at a large colliery in Durham :—

Diameter	36 feet
Width at inlet	12 ,,
Width at periphery	12 ,,
Diameter of inlet	13 ,,

Direct-driven by a single-cylinder horizontal engine. Diameter of cylinder and stroke of engine, 30 inches.

and the term *vis viva* is still very commonly used, although beginning to be replaced by the term kinetic energy—the *vis viva* of a body being twice its kinetic energy.

When a body is in motion in any line, whether straight or curved, and has a given linear velocity, we estimate the work stored in it by half the product of the mass into the square of its velocity. Thus the work stored up in a heavy weight placed at the end of a revolving bar depends only on the mass and the square of the linear velocity. The force to the centre in no way influences the result, except so far that it is necessary to maintain the motion.

Uniform circular motion is impossible unless the body be pulled in towards the centre by a force, and there being no action without an equal and opposite reaction, it is evident that we can imitate in a body at rest the conditions which obtain during circular motion by supplying a force equal and opposite to this centre-seeking force.

The fan and buildings (see Figs. 51 and 52) cost £2280, 16s. 8d. The annual cost of stores is £29, 12s. 11d., and repairs about £4, 3s. 6d.

The Walker Fan.—The Walker fan is one of the more recently designed types, and has come into use at

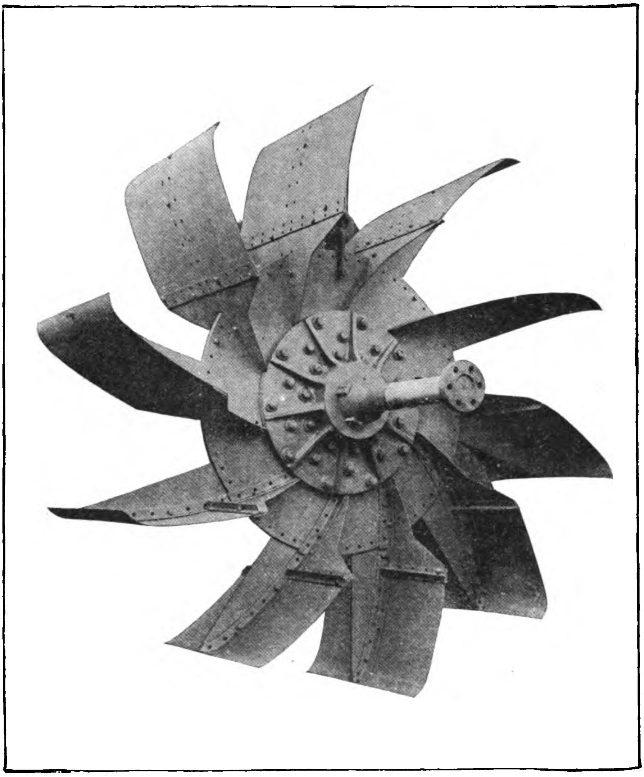


FIG. 53.—"Wheel" of Walker's "Indestructible" Fan.

many collieries. The details of the fan are shown in Fig. 53. It is constructed of steel throughout, and the blades, which number ten, it will be seen, are somewhat of the Guibal shape, but instead of being of the same width throughout, are cut away somewhat opposite the point where they join the disc plate. At the back the blades are supported by angle irons. The fan is exceedingly strong and very light, and is known as the "Indestructible," and has a double inlet.

Fig. 54 is a line drawing showing the general arrangement of fan, which, as will be seen, is rope-driven.

Fig. 55 shows the arrangement of the fan wheel in the casing. In this illustration the fan is shown having eight blades.

Results.—The following results which have been obtained with the fan have been supplied to the author by the makers, Messrs. Walker Bros. Ltd. of Wigan.

TABLE XII.—*Electrically-driven Ventilating Fans in the Midland District.*

Volume of air	320,429 cub. ft. per min.
Water-gauge near top of upcast shaft	4·3 inches
Horse-power in air	217
Electrical horse-power (input)	312·4
Brake horse-power in motor (output)	276
Combined efficiency of motor and fan	69·45 per cent. (70)
Efficiency of fan, including friction of driving ropes	78·6 ,,
Efficiency of fan, excluding friction of motor and ropes	82·5 ,,

TABLE XIII.—*Newcastle-on-Tyne District.*

Volume of air	312,630 cub. ft. per min.
Water-gauge taken near top of upcast pit	5 inches.
Horse-power in air	246
Electrical horse-power (input)	332
Brake horse-power of motor (output)	298·8
Combined efficiency of fan and motor	74 per cent.
Efficiency of fan, including friction of driving ropes	82·3 ,,
Efficiency of fan, excluding friction of motor and ropes	86·6 ,,

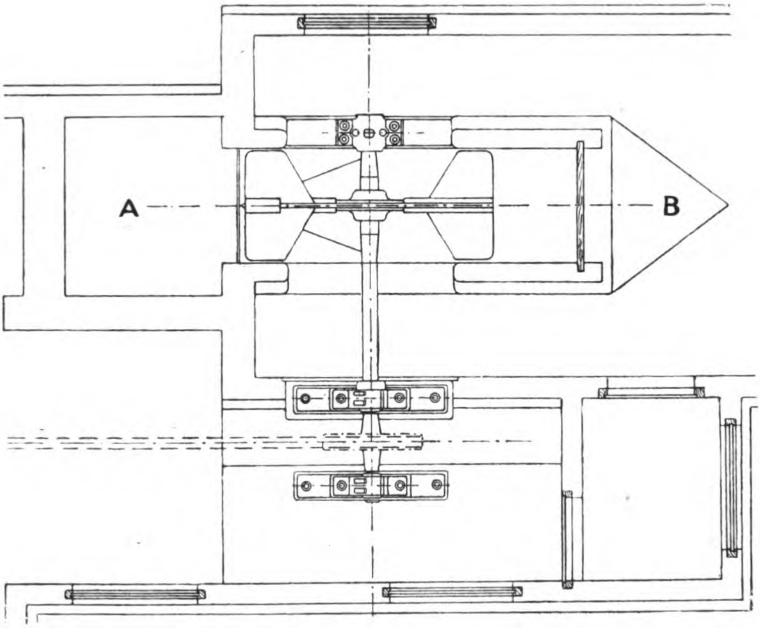


FIG. 54.—Plan showing Arrangement of Fan and Engine in a Rope-driven Walker Fan.

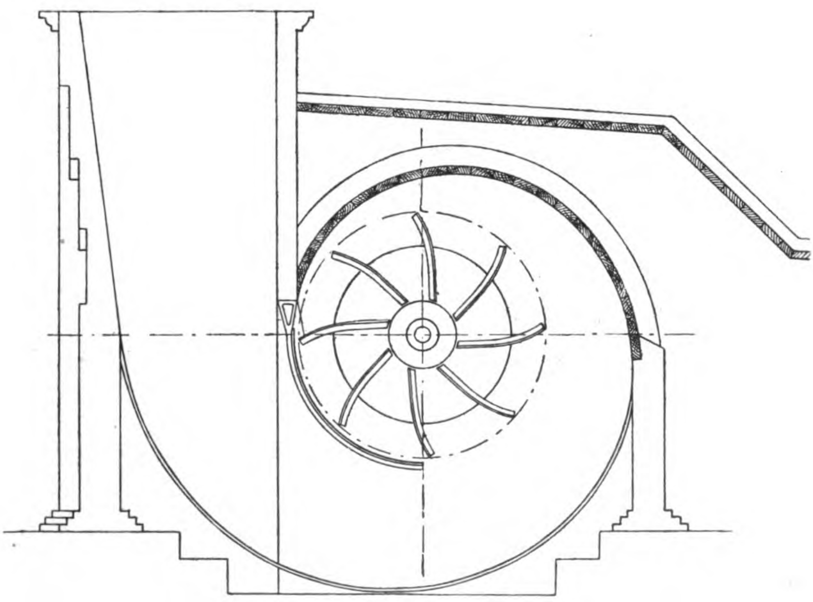


FIG. 55.—Vertical Section showing Walker Fan and Fan-casing.

TABLE XIV.—*Fifeshire District.*

Volume of air	302,892 cub. ft. per min.
Water-gauge taken near top of upcast shaft	3·8 inches
Horse-power in air	181
Electrical horse-power (input)	262
Electrical horse-power, output (90 per cent. efficiency)	236
Combined efficiency of fan and motor .	69 per cent.
Efficiency of fan, including friction of driving ropes	77 „
Efficiency of fan, excluding friction of motor and ropes	80 „

TABLE XV.—*Staffordshire District. Rope-driven from Cross Compound Steam Engines.*

Volume of air	299,318 cub. ft. per min.
Water-gauge in main shaft	8·27 inches
Horse-power in air	390
Horse-power in engines	605
Combined efficiency of fan and engines .	64·3 per cent.

The Chandler Fan.—The Chandler fan[1] (built by Messrs. Bumstead & Chandler of Hednesford, Staffordshire) was, the author believes, first applied to colliery ventilation about the year 1889, but it had been used for the purposes of forced draught in other directions for some years previously.

A suitable size for a fairly large colliery is 10 feet diameter over the blade tips. The fan consists of a central disc of steel boiler plate, and is keyed to a steel shaft carried by outside bearings. On each side of the disc are twelve curved blades (as shown in Figs. 56 and 57) riveted to steel angles, which are again riveted to the disc. The shape of blade which has been found with this fan to

[1] See "The Chandler Patent Fan," by R. S. Williamson, *Trans. Inst. M.E.*, vol. iii. pp. 171–174.

give the best results is, according to Colonel Williamson, that which may be represented by a much-flattened **S**. The central disc, besides acting as a support for the blades, serves the purpose of separating the two air-

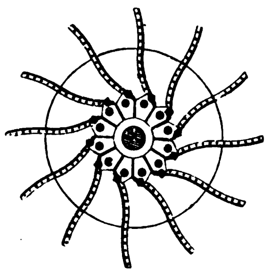
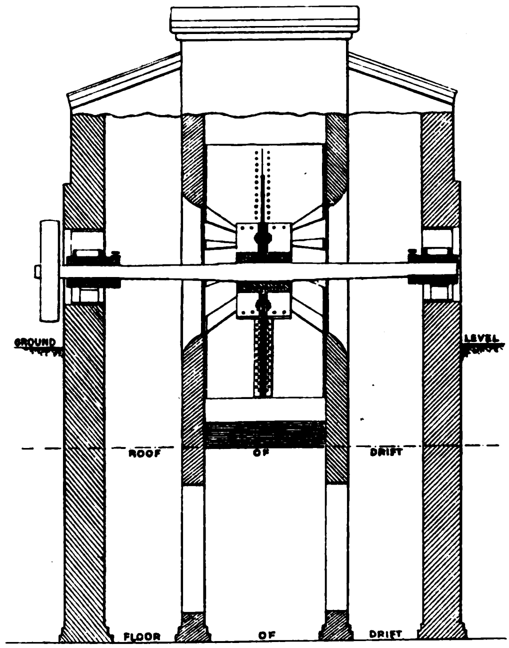


FIG. 56.



SECTION.

FIG. 57.

FIGS. 56 and 57.—The Chandler Fan. Fig. 56 showing the Arrangement of the Fan Blades; Fig. 57 Cross Section of Fan Chambers. (From *Trans. Inst. M.E.*, vol. iii. p. 174.)

currents until the fan motion has caused them to flow in parallel lines.

Fig. 58 is an external view of the fan and engine-houses.

Results.—The following are the particulars of a test made with this type of fan in April 1909, at the Coppice Colliery, Cannock, Staffordshire. The fan, which is 12 feet diameter, and was put down some years ago, is direct-driven by a vertical enclosed engine, having forced lubrication to all its working parts, through a flexible coupling fitted on the end of the fan shaft. During the tests the volume of air passing through the drift was

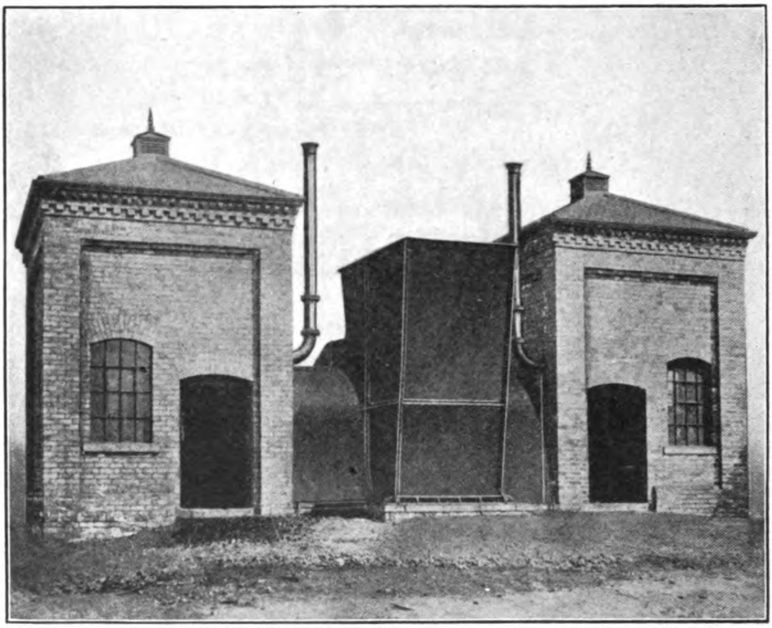


FIG. 58.—The Chandler Fan: External View of Fan-casing and Engine-houses.

measured, the drift being divided into twenty squares, two anemometers which had been recently corrected being used.

The engine has one cylinder 22 inches diameter by 15 inches stroke, and can work up to 250 revolutions, if wanted, at any time. The results were as shown in Table XVI.

TABLE XVI.—*Result of Tests, 30th March* 1909.

	Fan and Engine. Revolutions per Minute.	Average. (3 Sets of Cards.) I.H.P.	Water-gauge in Drift.	Velocity of Air. Feet per Minute.	Volume of Air. Cubic Feet per Minute.	H.P. in Air.	Engine Efficiency. Per Cent.	Fan Efficiency. Per Cent.	Combined Efficiency. Per Cent.
Test 2 . .	205	178·8	5·5	2,089	152,600	132·0	93·0	79·0	73·8
Test 3 . .	205	178·8	4·75	2,293	167,000	124·8	93·0	75·0	69·8

The makers state that it is a frequent occurrence for these forced lubrication engines to give 93 per cent.,

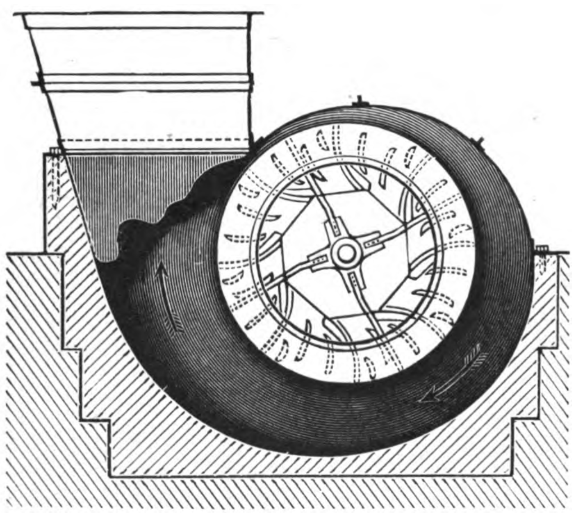


FIG. 59.—Capell Fan: Casing uncovered to show Position of "Wheel."

and even 94 per cent. efficiency at full load. They also state that this fan will easily pass 220,000 cubic feet of air per minute if the mine and drift will permit of the volume entering the fan, and this can be done at a moderate speed.

The Capell Fan.—The most recent development of this type of fan is illustrated in Figs. 59, 60, 61 and 62. All the vanes are fixed, and so move with the fan-wheel as it revolves, the concave side of the vanes opposing the air, in which fact the fan differs from the original type of Capell ventilator. The fan is fitted with main and tail wings, the main wing constituting a kind of scoop. The

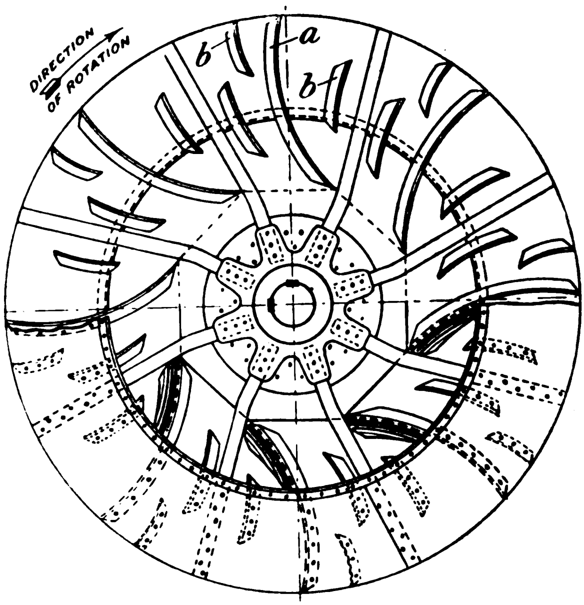


FIG. 60.—Capell Fan: British Type.
a, Main Vane; *b*, Tail Vane.

great point of the tail wings (*b*) is the prevention of the re-entry of air behind the concave main wings (*a*). Figs. 60 and 61 represent the British type and Fig. 62 the American type of the same patent. In the British type the tail wing is divided (*b*, *b*), and is an air-passing appliance, whereas in the American fan the same results and a rather higher water-gauge is secured with an attached tail wing (b') and an extra intermediate wing (b). The fans are made to be either direct, rope, or belt driven, and for exhausting or blowing.

The fans are in use at mines in sizes varying from 7 feet by 2 feet to 18 feet by 5 feet 6 inches.

A Capell fan, but not of the latest type, is at work at Neumühl Colliery, Rhine-Prussia, Germany, which gives

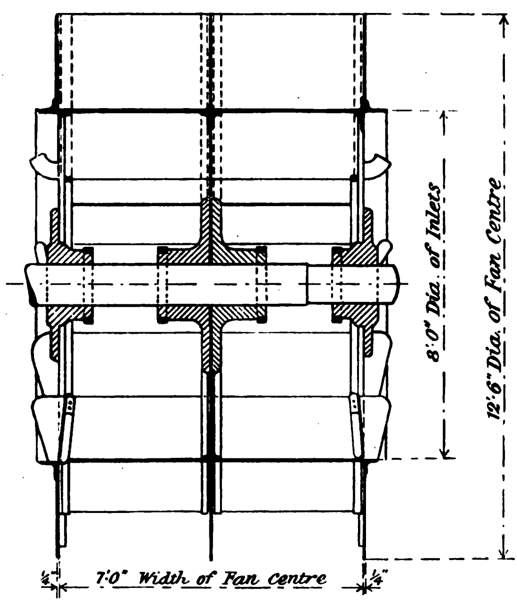


FIG. 61.—Capell Fan : Cross Section of British Type.

a volume of 328,000 cubic feet of air per minute with a water-gauge of 15·8 inches, and the author was informed that the useful effect is 80 per cent. The size of this fan is 16 feet 3 inches diameter by 5 feet 4 inches wide, and is driven at the rate of 270 revolutions per minute.

The following tests of three of the new type of ventilators have been supplied by the makers, viz.:—Test of fan, diameter 12 feet 6 inches, width 7 feet, rope-driven, at Shotton Colliery, County Durham. Date of test, 5th October 1907.

Revolutions	265
Water-gauge	8 inches
Cubic feet per minute	376,000
Horse-power in air	472
Horse-power indicated	579
Useful effect	81·5 per cent.

Test of fan, 16 feet by 8 feet, at Sterling Company's Mine, Uniontown, Pennsylvania, under Messrs. Cunning-

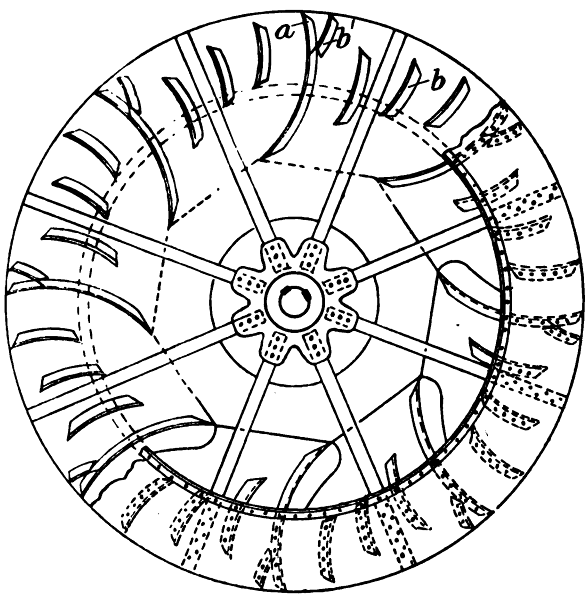


FIG. 62.—Capell Fan: American Type.
a, Main or Scoop Vane; *b'*, Tail Vane attached to Main Vane; *b*, Tail Vane.

ham and Maccanche, State Mine Inspectors. Fan direct-driven. Date, 6th October 1907.

1.	Revolutions, fan and engine	150
	Cubic feet per minute	387,379
	Water-gauge	3·8 inches
	Useful effect	78 per cent.
2.	Revolutions, fan and engine	158
	Water-gauge	4·3 inches
	Cubic feet per minute	430,479
	Useful effect	78 per cent.

Test of fan, diameter 11 feet, width 7 feet, at Mine of Caledonian Coal Company, Newcastle, New South Wales. Test, September 1907.

Revolutions per minute	270
Water-gauge	5 inches
Cubic feet per minute	265,000
Useful effect	70 per cent.

The Rateau Ventilator.—The Rateau fan[1] or turbine consists of a cone-shaped casting, on which wings or vanes are fixed. The mechanism is shown in Fig. 63, the two special features of which are the shape of the vanes, and the use of a diffuser. A represents the fan body, the surface (*a*, *b*) of which is attached to the boss B, which is on the end of the shaft C. D, D are the vanes, of which there are thirty, made of steel sheets. These are of special shape, and are fixed to the fan body at A. Each vane has four edges, viz. *ab*, *bc*, *cd*, and *ad*. The edge *ab* is attached to the fan body *c*, forms part of the periphery of the fan wheel; the edge *cd* moves close to the casing E, which is a continuation of the conical-shaped inlet *ij*, and the edge *ad* moves in the vicinity of the actual inlet F to the fan. The air enters the fan at the edges *ad* of the vanes, and passes out at the edges *bc*.

[1] "The Rateau Ventilator," by M. Walton Brown, *Trans. Inst. M.E.*, vol. iii. pp. 410–414.

The diffuser, which is built partly of masonry and partly of cast-iron, is composed of two parallel plates HH, surrounding the periphery of the fan, and a spiral, the height of which increases from the beginning of the diffuser to the base of the expanding chimney, the diffuser I, I, and the expanding chimney J, which completes the diffusion. Altogether the arrangement is not unlike that of a centrifugal pump.

The following are particulars of the belt-driven Rateau fan at the Aubin Colliery, Cransac, Aveyron :—

Extreme diameter	6 feet 6·74 inches
Width at rise of fan	0 „ 6·29 „
Diameter of inlet	3 „ 11·24 „
Area of inlet	11·41 square feet
Area at top of expanding chimney .	40·90 „
Diameter of pulley	3 feet 3·37 inches
Volume produced by the vanes . .	26,587 cubic feet.

The Sirocco Fan.—This ventilator, which heralded a distinctly new type, made its appearance a few years ago, since when it has been largely adopted throughout mining districts.

The distinctive feature of the fan is that it has an inlet of nearly the diameter of the fan itself, absolutely clear of obstruction ; numerous blades, very shallow radially and very long axially, being ranged round the periphery. The outer edges of the blades are curved forward in the direction of rotation, the air passages between the blades being open at the ends towards the inflowing air. This arrangement of blades is said to impart to the discharging air a velocity 80 per cent. in excess of that of the circumferential speed of the blades.

The fan is constructed throughout of steel boiler plate. They are made in a large number of sizes, varying from 2 ft. 11 in. to 8 ft. 4 in. in diameter, and designed for

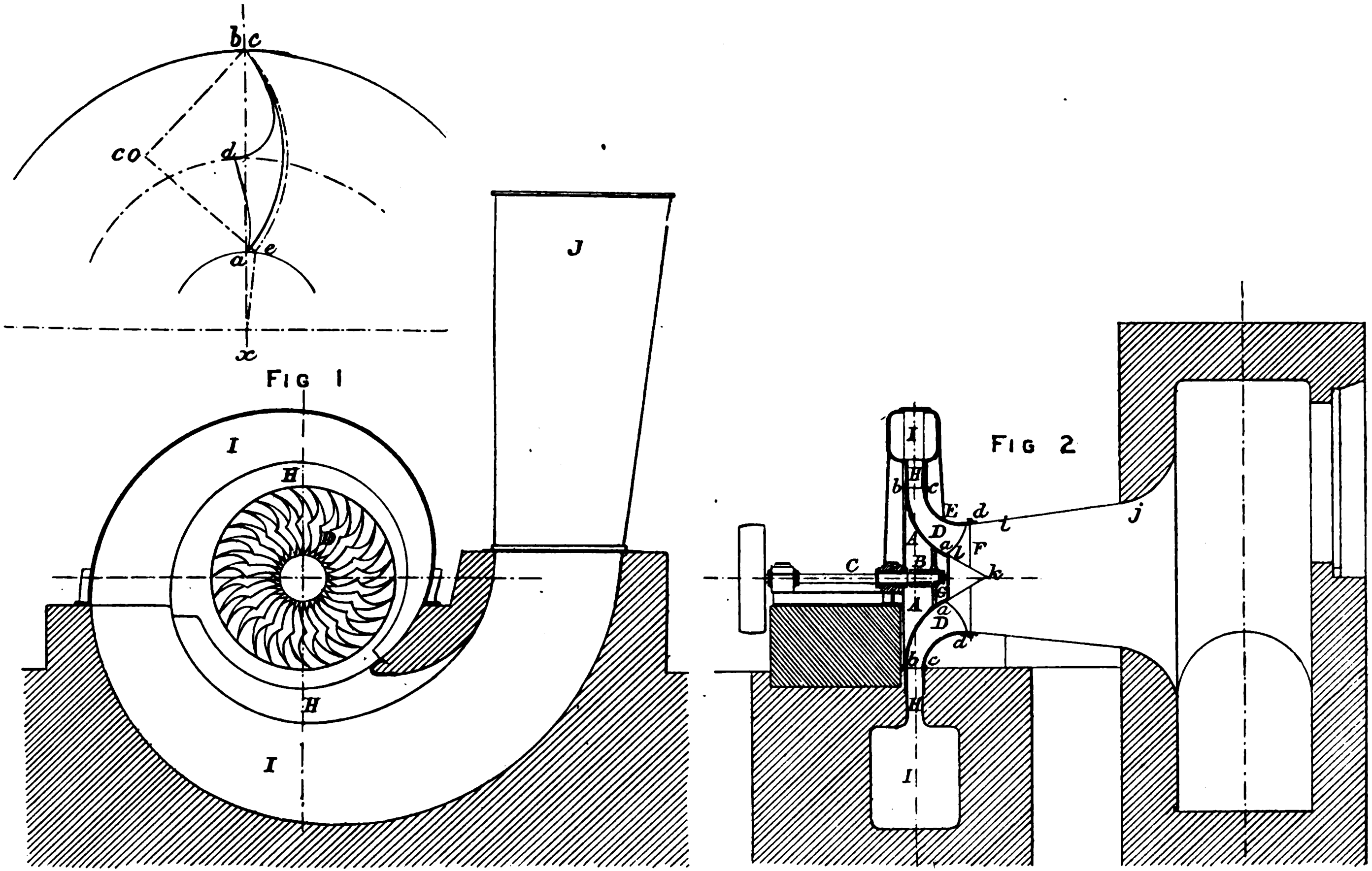


FIG. 63.—The Rateau Ventilator. (1) Transverse Section; (2) Longitudinal Section; (3) Construction of Blades.
(From *Trans. Inst. M.E.*, vol. iii. p. 414.)

single (Fig. 64) or double (Figs. 65 and 66) inlet. The velocity at which the fan is driven varies, as with all

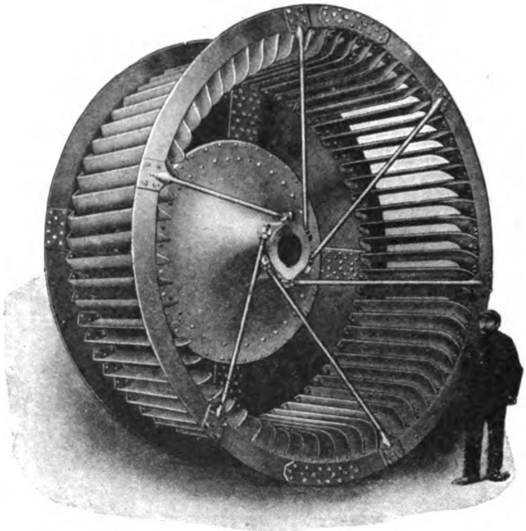


FIG. 64.—Single-inlet Sirocco Fan.

other fans, according to size; being of small diameter, it is a high-velocity fan.

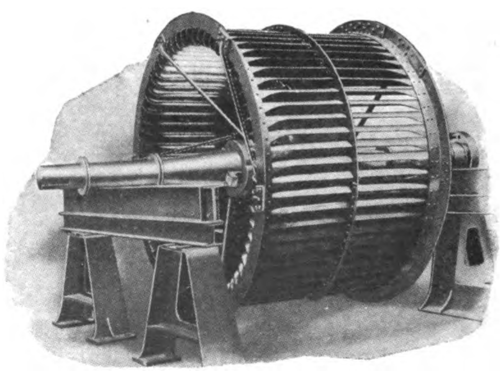


FIG. 65.—Double-inlet Sirocco Fan.

Figs. 67 and 68 show the arrangement of fan buildings and fan-drift in the case of single and double inlet fans respectively. The makers (Messrs. Davidson

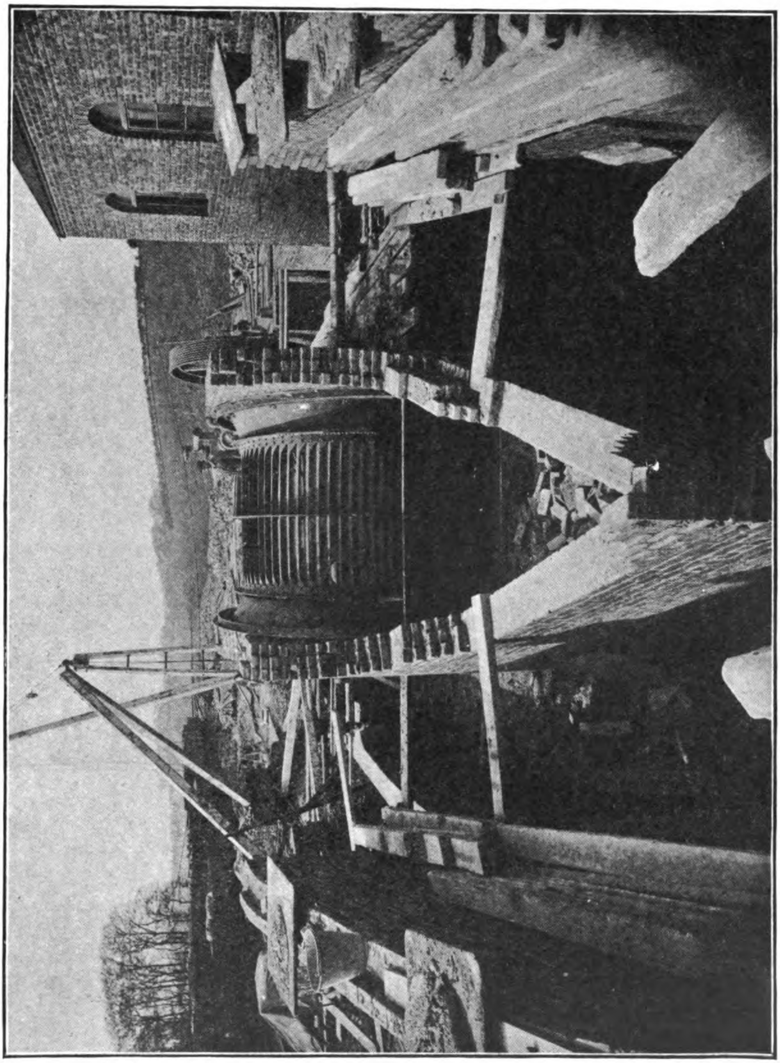


FIG. 66.—Double-inlet Sirocco Fan placed in position: Fan Buildings in course of construction.

& Co., Ltd., of Belfast) guarantee an efficiency of 70 per cent.

The following approximate speeds and outputs of air at 2, 2½, and 3 in. of water-gauge have been supplied by the makers :—

TABLE XVII.—*Approximate Outputs and Speeds of "Sirocco" Patent Centrifugal Cased Fans for Mine Ventilation, at 2, 2½, and 3 in. Water-gauge.*

Size of Fan.	2-in. Water-gauge.		2½-in. Water-gauge.		3-in. Water-gauge.	
	Revs. per Min.	Volume of Air, Cub. Ft. per Min.	Revs. per Min.	Volume of Air, Cub. Ft. per Min.	Revs. per Min.	Volume of Air, Cub. Ft. per Min.
	SINGLE-INLET TYPE.					
In.						
35	515	26,500	570	28,500	625	31,500
40	450	33,500	500	36,500	545	40,500
45	400	43,000	440	47,000	485	51,500
50	365	53,500	400	58,000	440	64,000
55	330	65,000	360	71,000	395	78,000
60	300	78,000	330	85,000	365	93,500
65	280	92,500	310	100,500	340	110,500
70	260	170,500	285	117,000	315	129,000
75	240	123,500	265	134,000	290	147,500
80	225	141,000	250	153,500	275	168,500
85	210	160,000	235	173,500	260	191,000
90	200	179,000	220	194,500	240	214,000
95	190	200,000	210	217,500	230	239,000
100	180	222,500	200	241,500	220	266,000
	DOUBLE-INLET TYPE.					
35	515	53,000	570	57,000	625	63,000
40	450	67,000	500	73,000	545	81,000
45	400	86,000	440	94,000	485	103,000
50	365	107,000	400	116,000	440	128,000
55	330	130,000	360	142,000	395	156,000
60	300	156,000	330	170,000	365	187,000
65	280	185,000	310	201,000	340	221,000
70	260	215,000	285	234,000	315	258,000
75	240	247,000	265	268,000	290	295,000
80	225	282,000	250	307,000	275	337,000
85	210	320,000	235	347,000	260	382,000
90	200	358,000	220	389,000	240	428,000
95	190	400,000	210	435,000	230	478,000
100	180	445,000	200	483,000	220	532,000

Barclay's Drum Pattern Fan.—A fan which has been built and installed during the last fifteen years is that constructed by Messrs. Barclay & Sons of Kilmarnock, N.B. It will be observed that it is somewhat similar to

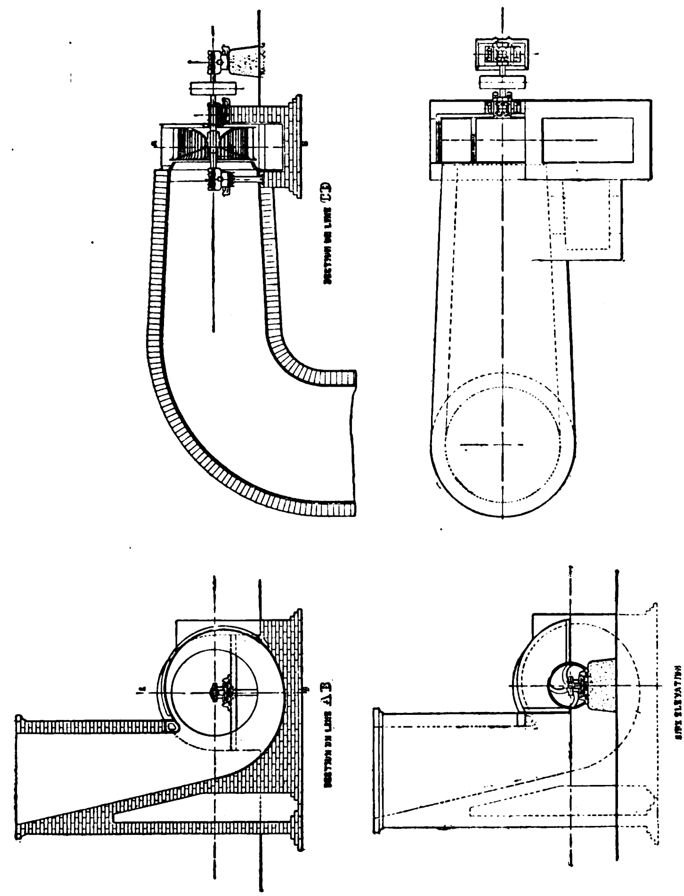


FIG. 67.—Arrangement of Sirocco Fan for Mine Ventilation: Single-inlet, Upcast Type.

the Sirocco type (see Figs. 69 and 70), and it is claimed for it by the makers that the shape and lead of the main vanes is such that the effective impulse is given to the air in the body of the fan. The fan inlet, like the

Sirocco, is clear of all obstruction. The ventilators are made of riveted boiler plate, and are very strongly con-

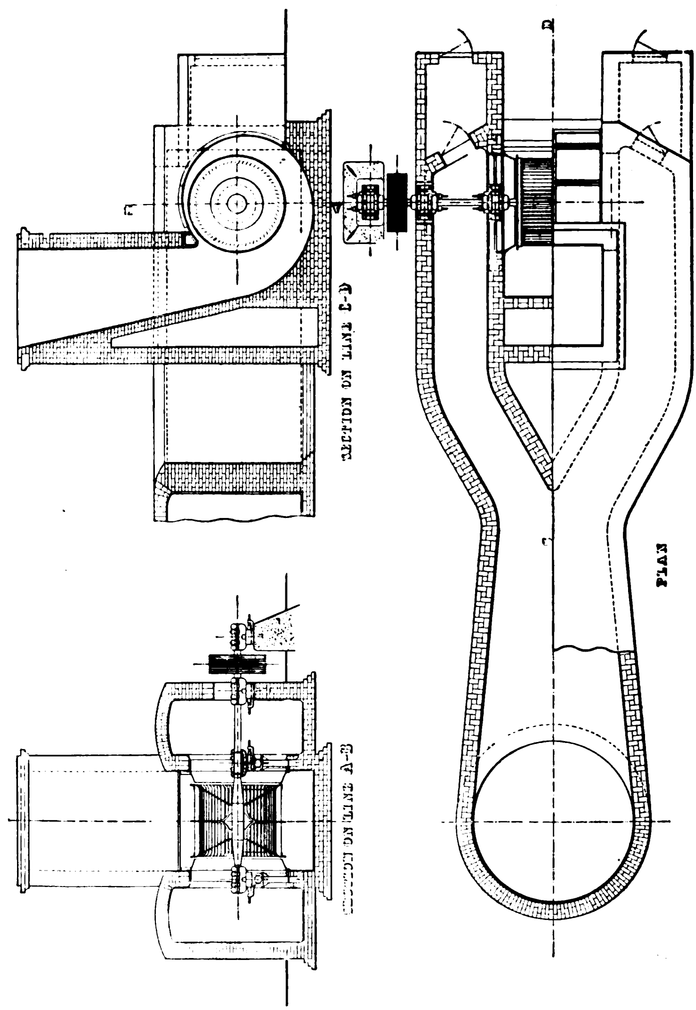


FIG. 68.—Arrangement of Sirocco Fan for Mine Ventilation: Double-inlet, Upcast Type.

structed, have strong shafts and ample bearings; they are of the single or double inlet type.

A Barclay drum fan was recently ordered 10 feet in

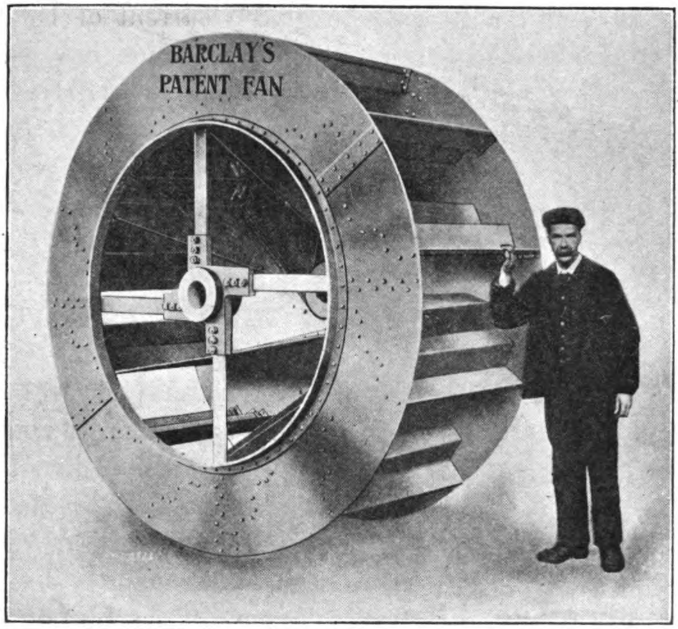


FIG. 69.—The Barclay Single-inlet Drum Fan.

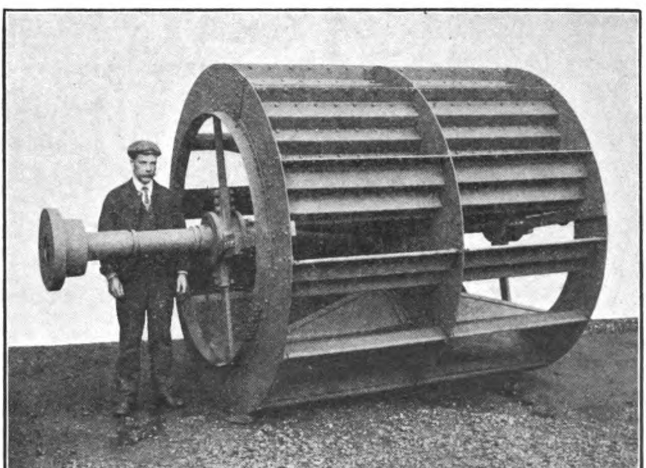


FIG. 70.—The Barclay Double-inlet Drum Fan.

diameter and 7 feet wide, to give a current of 180,000 at 240 revolutions per minute; water-gauge not stated. And a direct-driven 20 feet diameter fan is giving 200,000 cubic feet per minute. This firm build also the Capell type of fan.

FIG. 71.

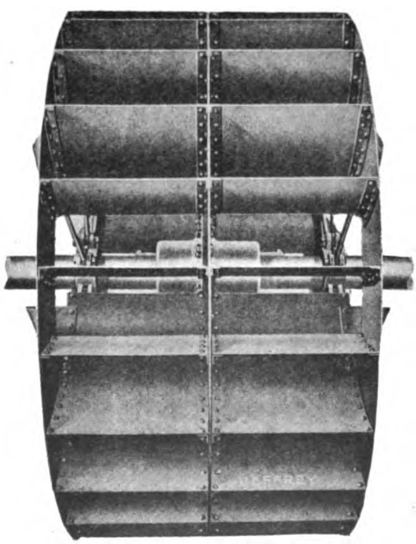


FIG. 72.

FIGS. 71 and 72.—The Jeffrey Centrifugal Fan Wheel: Double-inlet Type.

The Jeffrey Centrifugal Fan. — This ventilator (Figs. 71 and 72) is of American make, being manufactured by the Jeffrey Manufacturing Company of Ohio, U.S.A., and is the outcome of a series of tests instituted for the purpose of ascertaining the exact curvature and position of vanes, which would give the best results under conditions peculiar to mines. The makers claim for it that it develops large air volumes against high water-gauges at low speeds. The special features of the fan are the shape and position of its vanes, its conical scoops and the practically unobstructed inlet. The vanes are six in number and curved, discharging the air radially. In the single-inlet form the vanes are fixed to the sides of the fan-wheel, but in the double-inlet form there is a dividing central disc. There is very little central obstruction, and the inrushing air is caught by conical scoops, placed as shown in Fig. 71, which prevents the gushing of the air from the inlet.

The manner of bracing adopted is also shown in Fig. 71.

The fan is constructed entirely of steel mounted on a hammered steel shaft.

The following table shows results of tests made with the above 10 ft. by 5 ft. double-inlet blowing-fan installed at the works in Columbus, Ohio:—

TABLE XVIII.—*Tests made with a Jeffrey Fan.*

Revolutions per Minute.	Fan Discharging into Free Air.					Outlet Closed.	
	In Water-gauge.	Area of Discharge.	Velocity of Air in Feet per Minute.	Volume No. Cub. Ft. per Minute.	Volumetric Capacity.	In Water-gauge.	Manometric Efficiency.
					Per Cent.		Per Cent.
50	0·15	34	1,530	52,020	265	0·22	67
100	0·6	34	3,070	104,380	266	0·9	66
150	1·4	34	4,700	159,800	271	2·0	70
200	2·5	34	6,280	213,520	272	3·6	71
250	3·8	34	7,750	263,500	268	5·3	72
300	5·6	34	9,390	319,260	271	8·0	70

A recent test made with the above 10 ft. by 5 ft. fan, installed for the Bituminous Coal Company of America, gave the following results:—

TABLE XIX.—*Tests made with a Jeffrey Fan.*

Revolutions.	Area.	Velocity.	Volume.	Gauge.	Volumetric Capacity.
	Sq. Ft.			In.	Per Cent.
95	50	1,700	85,000	0·3	240
144	50	3,000	150,000	1	265

The fans are built in four types, viz. for blowing, exhausting, blowing reversible type, and exhaust reversible type (see Fig. 73). The last two correspond to the first two, except that the shutters inside the casing are arranged to allow of reversing the air-current.

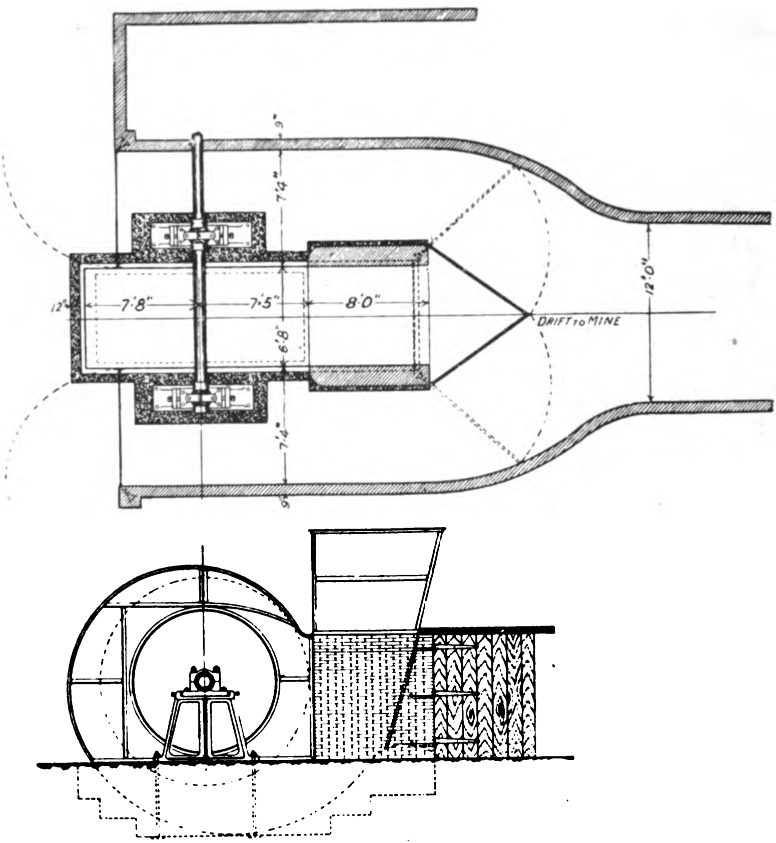


FIG. 73.—The Jeffrey Fan: General Arrangement of a Large Double-inlet Exhaust Reversible Fan.

The Parsons Turbo-Ventilator.—The points of importance and peculiarity in this type of ventilator are (1) in respect of the motor, and (2) the ventilator itself. The motor is a steam-driven turbine of the well-known

Parsons type (Fig. 74), and (2) the exhauster is a fan designed on the propeller principle (Fig. 75).

I am indebted to the makers, Messrs. Parsons & Co., of Newcastle-upon-Tyne, for the following description of the turbine and exhauster:—

The steam turbine in its simplest form may consist of one or more nozzles directing jets of steam tangentially on to suitable buckets or blades mounted on a ring or a wheel. This is the impulse type. Or again, it may consist of one or more nozzles carried on an arm or disc pivoted at its centre, the steam issuing tangentially from these nozzles, and by its reaction causing the disc to rotate. This is the reaction type of turbine. In practice both these types are combined in the Parsons compound steam turbine, which consists of alternate rows of fixed and moving blades. Each set of fixed and moving blades is like a simple turbine complete in itself. The steam on entering the cylinder passes through the first row of fixed guide blades, which direct it in a number of streams in a tangential direction on to the first row of moving blades, thus imparting to them a rotational force due to the impact of the steam. The curved shape of the passage through the blades reverses the direction of the steam, so that it issues from the moving blades tangentially but in the opposite direction, and by its reaction imparts additional rotational force to the moving blades. The steam then reaches the next rows of fixed and moving blades, where a similar operation takes place at each.

The energy to give the steam its high rotational force at each successive row of blades is supplied by the drop in pressure, the steam expanding gradually by small increments. Since the number of rows is comparatively large, the velocity of the steam through each is moderate, and no cutting of the blades can take place.

The steam turbine which is shown in section in Fig. 74 consists of a cylindrical case with rows of inwardly projecting blades, within which revolves a concentric shaft with rows of outwardly projecting blades.

The steam enters at A on the lower half of the cylinder, thus leaving the upper half quite clear of steam pipes and all obstructions and facilitating dismantling. It then passes successively through the different rows of fixed and moving blades, as explained above, and leaves the cylinder through the exhaust pipe B.

In order to give increased passage-way for the steam as it expands, the shaft is made with three steps of different diameter, the heights of the blades being also increased.

The steam, in addition to its rotational force, exerts a pressure endways along the shaft on the surface of the blades and the shoulder of the shaft. This is balanced by the dummy pistons C′, C″, C‴, as shown in the section. They are made of diameters corresponding to the different parts of the turbine they balance, and are supplied with the corresponding steam pressure through the pipes P′, P″. The shaft thus runs in complete balance endways, and can be moved backwards and forwards with a light lever even when the turbine is running under full load.

In order to prevent steam leakage, grooves are turned in these pistons, into which project, without, however, touching the moving parts, suitably shaped strips of brass caulked into grooves in the cylinder. The whole form a labyrinthine passage offering great resistance to the escape of the steam, most of which is carried round and round by the skin friction of the dummy pistons, producing a most effective screen against leakage.

The two glands D, where the shaft leaves the turbine casing, are constructed in precisely the same manner. The steam for packing them is obtained from the exhaust

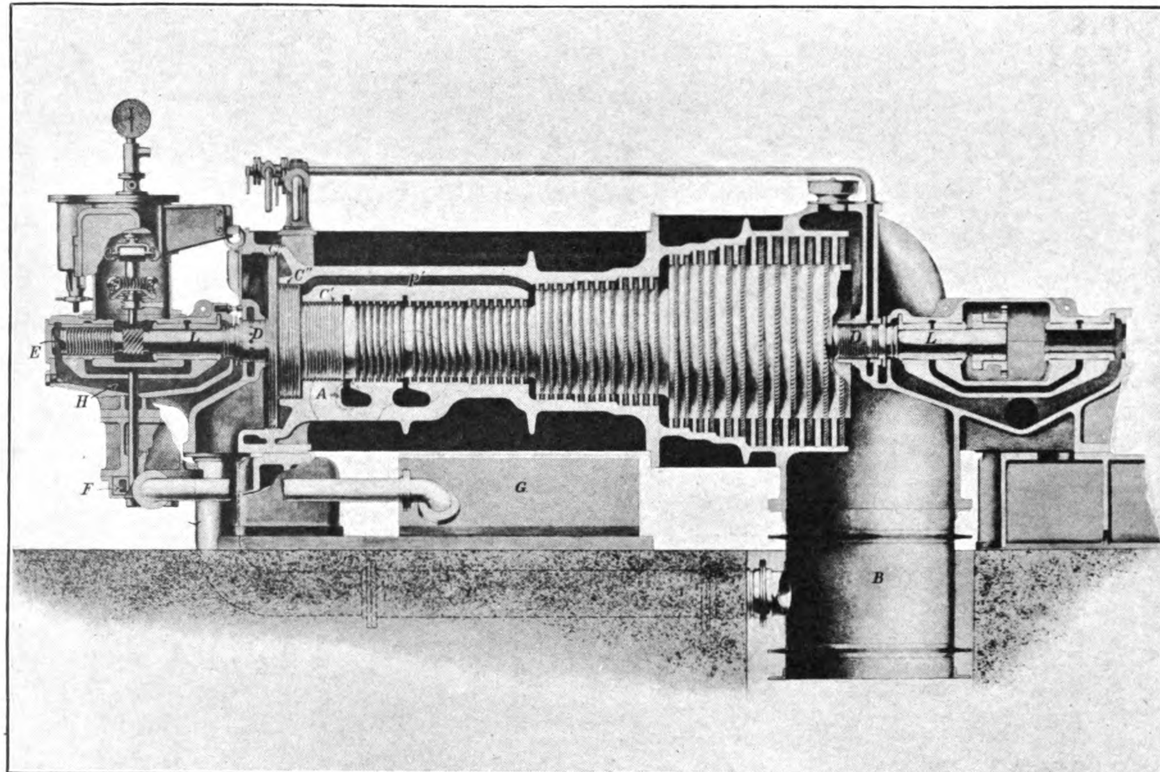


FIG. 74.—Section through Parsons Steam Turbine.

of the steam relay, a live steam connection being fitted for use before starting up. An ejector is also fitted to draw the excess steam away from the glands.

The coupling between the turbine and generator is of the flexible claw type, to allow for slight differences in alignment of the two portions of the plant.

The thrust-block E, at the end of the turbine shaft, merely keeps it in place with the right clearance between the fixed and moving parts of the glands and dummies, and adjustment is made in a few minutes with a small liner behind the thrust-block.

The shafts themselves are solid steel forgings in the smaller sizes, while in the larger sizes they are built up of hollow steel drums machined inside and outside, and then carefully shrunk together and pinned.

The blades are composed of a special brass alloy which long experience has proved to be the most suitable. This is rolled and drawn to the required section, and highly polished, so as to ensure the friction with the steam being as little as possible. For the first few rows of blades, copper is used instead of brass, as being more suitable for superheated steam. Turbines fitted with this blading have been opened up after running many years, and not the slightest cutting action has been visible on any of the blades.

The tips of the blades are thinned, so that in the case of their touching the ends are merely ground away or slightly bent over. This improvement has been found to afford additional security, without at the same time affecting the steam consumption.

For many years past the blades have been caulked one by one into grooves in the cylinder and shaft, a method which has proved satisfactory. Messrs. C. A. Parsons & Co., however, have devised and are also using several methods of assembling and holding the blades in complete sectors of suitable lengths ready for insertion in the cylinder and spindle. These sectors can all be prepared beforehand and kept ready in stock, so that the actual operation of fixing the blades in the grooves of the cylinder and spindle is done very quickly.

In one form, applicable also to existing turbines which have been already bladed by the older methods, the standard blades and packing pieces are provided with holes by which they are strung on a wire of suitable section; they are then driven up tight in a former curved to the right radius for the part of the cylinder or spindle which they have to fit; the ends of the sector are secured by riveting or any other suitable method. The complete sector is then ready for fixing in the turbine by caulking in the usual manner.

In all these methods the full strength of the blade is maintained unimpaired right into the root fixing, there being no swaging or spreading out of the root or possible cracking of the metal, while at the same time the hold on the root is so firm that there is absolutely no fear of the blades ever coming loose.

L, L. These in the larger sizes are lined with white metal. In the smaller sizes, where the number of revolutions is much greater, the tubular type is used, consisting of a central bearing bush, prevented from turning by a lug cast on it at one end. On this bearing are slipped concentric tubes fitting loosely into one another, between which the oil circulates and acts as a cushion, effectually damping out all vibration which may arise from slight want of balance or alignment. Since there is always a film of oil between the journals and the bearings, these should run for years with practically no wear.

The whole of the lubrication of the bearings, thrust-block, worm, and governor gear, &c., is automatic, the oil being drawn by the pump F from the tank G. The pump is of the rotary type, consisting of two wheels gearing into one another, which sweep the oil under considerable pressure round the outer part of the casing. It is placed low down, so that it is constantly flooded, and since it has no valves and is driven by a prolongation of the governor shaft, its failure is practically impossible. For flooding the bearings before starting up an additional small hand-pump is fitted.

A cooler is also provided to reduce the temperature of the oil.

Since there are no rubbing contacts inside the cylinder, no lubrication is required there, a fact which can be appreciated by those who realise the troubles of lubricating the cylinder and piston-rods of ordinary engines, especially when running with superheated steam.

Since the same oil is circulated over and over again by the pump, and since no lubrication is required in the cylinder, the total consumption of oil is exceedingly small.

The governor gear has been re-designed, but still retains the most valuable features of the older form, with all the improvements that experience can suggest.

The governors themselves are of the centrifugal type, with the springs acting directly between the balls, and are both mounted on the same spindle H, driven off the main turbine shaft by worm and worm-wheel.

Steam is admitted to the turbine in a series of gusts by the periodic opening and closing of a double-beat valve, operated by a steam relay in mechanical connection with the turbine shaft, the duration of each gust being regulated by the position of the governor balls and collar. One end of the governor lever controls the plunger valve of the steam relay, and the other end is furnished with a small roller, which runs on the collar of the governor. This collar has one side higher than the other, and as it turns round raises and lowers the end of the governor lever, and thus gives the periodic motion to the valve.

For hand adjustment of speed, a separate collar, spring, and lever are fitted to the governor. In addition, a runaway valve is also fitted, being held open against a spring by a catch and double-trigger arrangement, so designed as to reduce pressure on the point of the catch to a minimum. In this way the release of the valve takes very little force, and is extremely regular in its action, always cutting out at the speed for which it is set. There is also a lever provided, by which the valve can be tripped by hand.

The work done by the governor itself is merely a slight alteration of the mean position of the lever end, and the relay plunger attached to it, the amount of travel remaining constant. The volume also of steam in the turbine is comparatively small, and passes very quickly through to the exhaust, so that the turbine itself responds at once to the governor, and the energy stored up in the rotating parts being large, the variations in speed, momentary and permanent, are very small. For these reasons the turbine is specially adapted for such severe loads as traction and railway work.

The exhauster, Fig. 75, shows this with the front valves of the inlet and outlet bends, as well as the central cover removed, in order to show the arrangement of the internal parts. It will be seen that the exhauster is direct coupled and that the shaft carries the screw propellers, which are arranged to run at any speed up to 8000 revolutions per minute. Between the propellers there are arranged sets of stationary guide blades, which are suitably curved so as to give maximum efficiency. These guides serve to take the spin out of the air as it leaves the first propeller, and to direct it on to the second one, where it receives a further compression, and so on. To increase the water-gauge it is only necessary to add one or more propellers. Remarkably high water-gauges have been secured with this arrangement. Not many of the Parsons ventilating sets of the propeller type have as yet been installed, as the number of cases in which the volume of air and water-gauge required is suitable for their economical operation is only small. But having developed a satisfactory form of reduction gear for use with steam turbines, any of the well-known types of centrifugal ventilators can be driven by means of the steam turbine, either from high pressure, mixed pressure, or exhaust steam, coupled to a slow-speed fan by this reduction gear.

The following particulars concerning a Parsons ventilating set of the propeller exhauster type at Florence Colliery in North Staffordshire, for which I am indebted to Mr. G. A. Mitcheson, the General Manager of the Stafford Coal and Iron Company, and the Florence Coal and Iron Company, are of considerable interest, as pointing to the application of this type of ventilator to meet conditions of an abnormal character.

The colliery is one of the deepest in Great Britain, the lowest working seam being 859 yards from the surface. The shafts are narrow (downcast 14 feet in diameter, upcast 12½ feet in diameter), and a very large volume of air was required against a high water-gauge. The turbo-ventilator was constructed to give 250,000 cubic feet of air per minute, against a water-gauge pressure of 12 inches, the exhauster being driven at a speed of 2500 revolutions

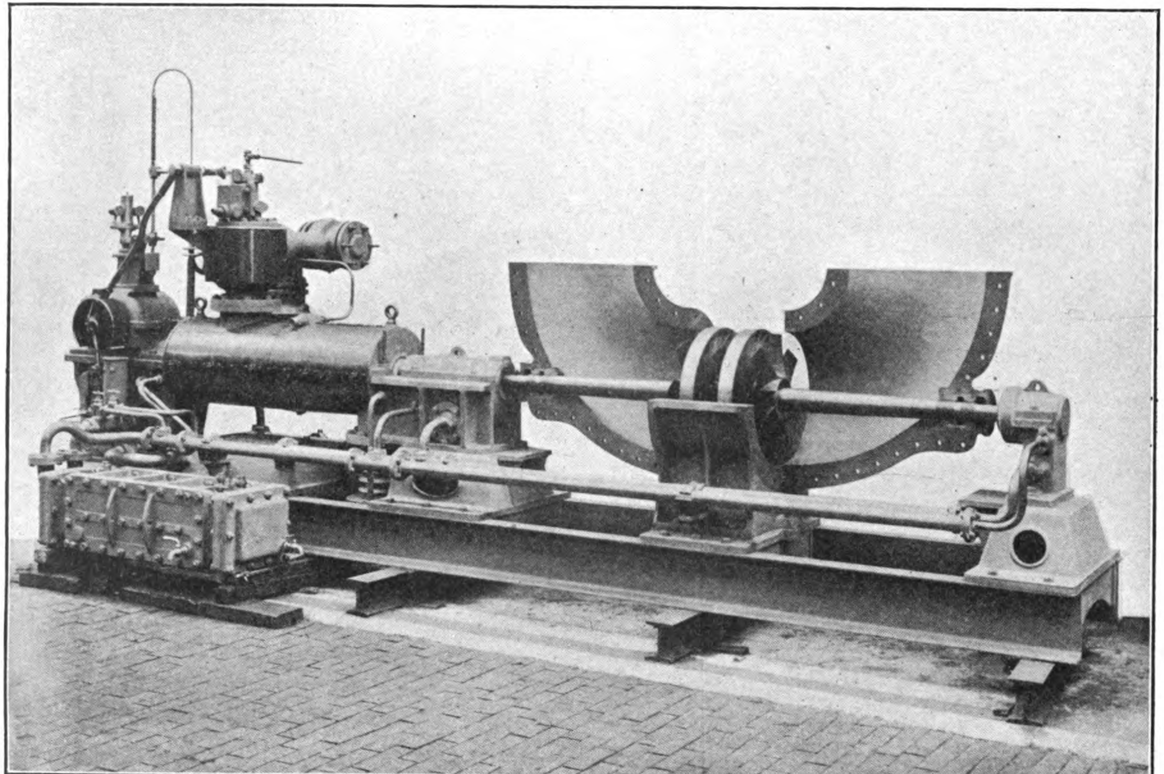


FIG. 75.—Parsons Turbo-Exhauster: View showing Internal Construction.

per minute. The guaranteed steam consumption, with steam on the stop valve at a pressure of 130 lbs. per square inch and a vacuum of 27 inches, was 15,000 lbs. of steam per hour, or with a volume of 200,000 cubic feet per minute against a water-gauge of 8 inches and a speed of 2000 revolutions per minute, 9100 lbs. of steam per hour. The "fan" discharged through a "disperser" into an uptake shaft (see Figs. 76 and 77), but it was found that

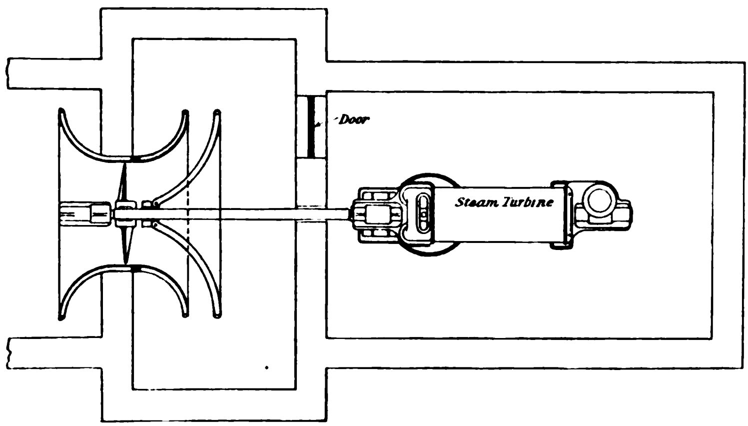


FIG. 76.—Plan of Parsons Turbo-Exhauster, Fan and Engine-house, and Steam Turbine at Florence Colliery.

when the air was discharged in that way it occasioned such an unpleasant sound that it was deemed necessary to erect a structure in which "baffles" were placed to break up the air waves. This structure, which is of large dimensions, allows of the air-current being greatly reduced in velocity before it is discharged into the external atmosphere. The "fan" has been tested up to its full capacity, and answers all requirements, but at present is not running against more than a 7-inch water-gauge, owing to the fact that there is a considerable leakage from the surface into the upcast shaft. An air-current of

293,000 cubic feet of air per minute has repeatedly been measured in the fan drift, a very satisfactory result.

The Reversing of the Air-current.—The necessity of being in a position to reverse the air-current in the mine at short notice has of late years been more and

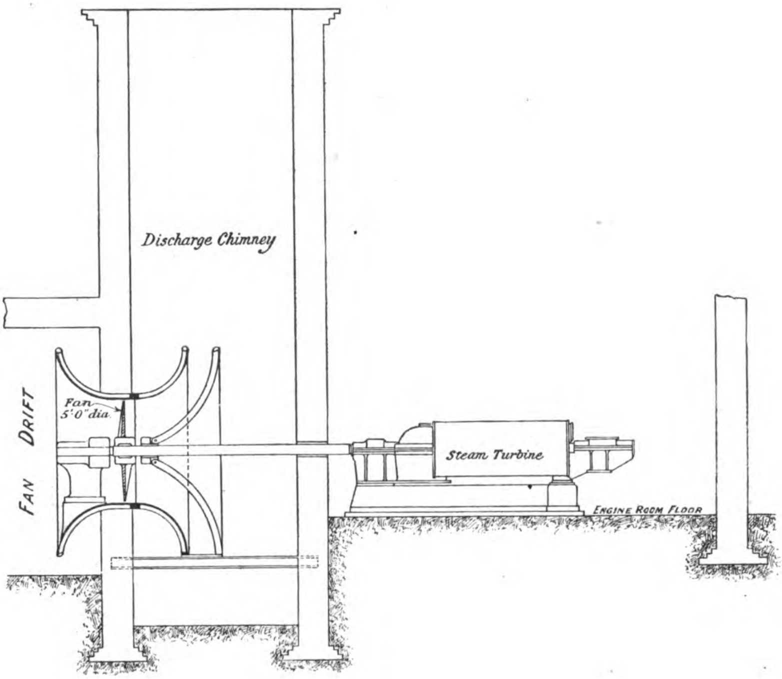


FIG. 77.—Cross Section of Fan Chamber at Florence Colliery showing Arrangement of Parsons Turbo-Exhauster and Steam Turbine.

more impressed on the minds of mining engineers. Events may occur, or combinations arise, such as the existence of underground fires in particular, in which the position of the fire being known, the reversal of the air-current may be the means of saving the lives of those below ground. It is impossible in the limited space of a text-book to elaborate the argument, and point to all the possible combinations rendering necessary the reversal. Before such a proceeding as reversing the ventilation of a mine is resorted to, the most earnest consideration will have to be given by the manager of the mine to all the circumstances of the particular case. But as to the desirability of the ventilating plant at the surface being so arranged as to enable the reversal, when determined upon, being put into operation at a moment's notice, there can be no two opinions.

Fig. 78 shows an arrangement for this purpose where

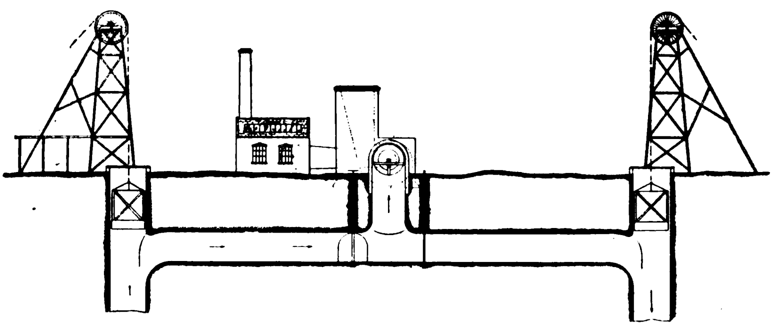


FIG. 78.—Showing how a Fan (a Sirocco) may be arranged for Reversing the Air-current Underground.

the downcast and upcast shafts are within a reasonable distance of each other. The fan suction is connected to a vertical shaft from the bottom of which a drift is led to the upcast and downcast shafts respectively. To reverse the-air current, doors are arranged on each side of the vertical shaft, so that by adjusting these the air can be drawn from either the upcast or downcast shaft at will.

In the case of winding in both shafts provision requires to be made for closing in the pit top to prevent short circuiting of the air. In cases where it would not be convenient to cover in the pit top, double doors can be arranged in the shaft immediately above the drift leading to the fan, thus forming an air lock, and preventing an inrush of air from the surface. This change can be made without in any way affecting the efficiency of the fan. The arrangement is shown carried out with a Sirocco fan.

Auxiliary Ventilation.—There are occasions when fans have to be used underground, and arranged to act in conjunction with the surface fan. Thus, if the workings are a considerable distance from the shafts, and the leakage of air is great, it is sometimes deemed advisable to instal a fan some distance inbye, and apply the same, either as a blower in the intake, or as an exhauster in the return air-way. Such fans being driven either by endless steel wire rope, or more commonly by compressed air or electricity.

The late Mr. Walton Brown arrived at some valuable results in respect of auxiliary ventilation with an underground fan from some experiments which he carried out at a colliery in Durham.[1]

In this instance the colliery was ventilated by a surface fan (Schiele 8 feet diameter) exhausting from three seams, which may be referred to in order of depth as Nos. 1, 2, and 3, No. 1 being nearest the surface. An auxiliary fan (Scheile 5 feet in diameter) was placed in No. 2 seam near the upcast shaft.

The general result of the experiments may be summarised as follows:—

The ratio of the underground fan revolutions to the surface fan revolutions being increased from 0 to 2·445, the volume of air in No. 2 seam rose 78 per cent., the total volume of air from the mine rose 28 per cent., whilst the

[1] "Experiments on an Auxiliary Ventilating Fan," by the late Mr. Walton Brown, *Trans. Inst. M.E.*, vol. xxxiv. pp. 494–509.

ventilation in No. 1 and No. 3 seams fell 9 per cent. Mr. Brown concluded his paper with the following interesting observation :—

"The theoretical law that the volume is in ratio to the cube root of the power is confirmed, and the experiments prove that the law holds good even when the total power is obtained from two separate fans.

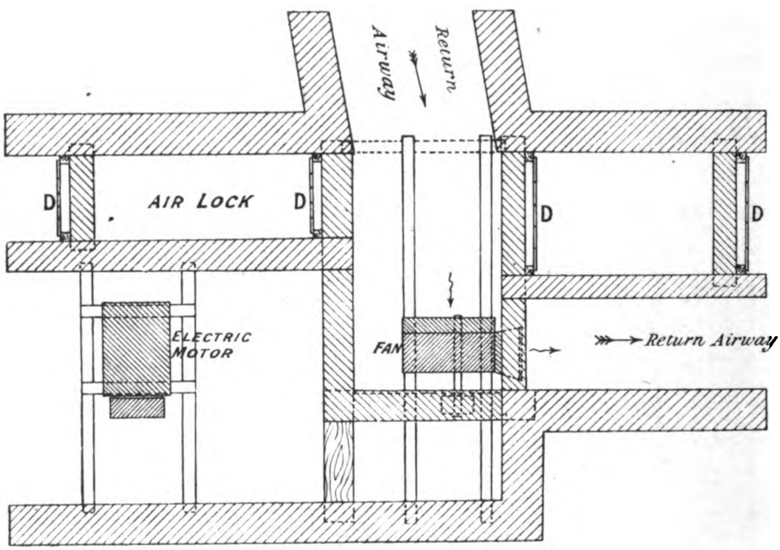


FIG. 79.—Arrangement of a Sirocco Fan Underground in the Return Air-way as an Auxiliary Ventilator, Florence Colliery, Staffordshire.

But the speed of the rear fan must not unduly lag behind the speed of the other. Whether two separate ventilating installations are as economical as one single installation designed for the entire work is another question."

Fig. 79 shows in plan the general arrangements made in 1907 in placing a Sirocco fan underground at a deep colliery in Staffordshire. This fan passed about 32,000 cubic feet of air at a water-gauge of 4·9 inches. The result of the carrying the arrangement into effect was that the total volume of air in the mine was augmented, but it was necessary to increase the number of revolutions of the surface (Waddle) fan, owing to the increased ventilation necessary, but the increase in volume was greater than that due to the increased power put into the Waddle, the effect being that the underground fan acted as a "helper up" to the surface fan.

The cost of the underground fan was £48, and the

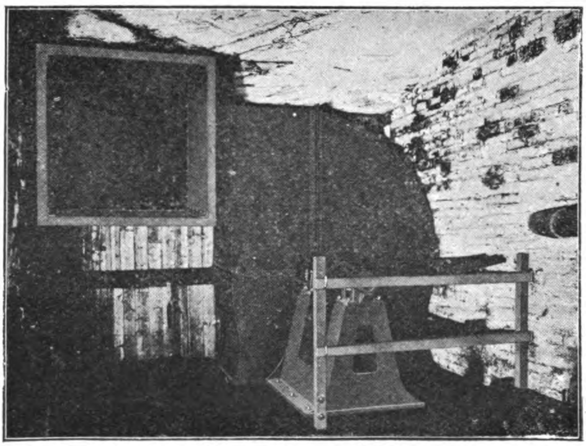


FIG. 80.—Sirocco Fan placed Underground at Peel Hall Colliery, Lancashire.

total cost of fan and motor and installing same was about £500.

Fans may also be used underground in a more local capacity, viz. the ventilation of a particular district, or to ventilate dead ends, the atmosphere in which may not be much affected by the main air-current.

Fig. 80 shows the arrangement of an underground fan at Peel Hall Colliery in Lancashire. The fan is a 45-inch diameter Sirocco single inlet fan, placed at the pit bottom, and discharging into the upcast shaft. The fan is driven by steel chain belting from a small steam engine, and is capable of producing a current of 50,000 cubic feet per minute at a 3-inch water-gauge.

The Theory of Ventilation by Centrifugal Fans: The Perfect Mechanical Ventilator.—Centrifugal ventilators possess this advantage over other forms of mechanical ventilating machines, that they do not require the use of valves, and a larger percentage of the power put into the machine is usefully consumed; in other words, they have a higher useful effect.

If the case of a fan with radial vanes is taken, whatever may be the radial velocity of the air passing from the fan, it must flow from the vane tips at a tangential velocity equal to the peripheral speed of the fan.

Now if u = the *peripheral* speed (= the linear velocity of the vane tips) in feet per second.

If the fan is exhausting into the atmosphere per second a weight of air = W lbs.

The momentum[1] of the air $= \frac{W}{g} \times u$ = foot pounds per second. And, as by Newton's second law of motion, change of motion is proportional to the acting force, and takes place in the direction of the straight line in which the force acts. By *motion* we are to understand motion as measured by momentum—

Hence the force propelling the air from the fan $= \frac{Wu}{g}$.

The third law of motion states that action and reaction are equal and opposite. Therefore there must be acting on the circumference of the fan and tending to retard its progress a like force, viz. $\frac{Wu}{g}$, and force multiplied by velocity is work, measured in foot-pounds.

[1] Momentum = mass × velocity.

Therefore this retarding force (the overcoming of which is the work done by the fan having a velocity u)

$$=\frac{Wu}{g}\times u=\frac{Wu^2}{g}$$

Hence every pound of air (*i.e.* every unit of air) leaving the fan is possessed of $\frac{u^2}{g}$ more energy than it possessed on entering it. So that the following conclusion is arrived at—a perfect ventilator would be such as would raise every pound of air $\frac{u^2}{g}$ feet high.

If h = equivalent head or column of air, assumed at atmospheric pressure.

Then with *radial vanes* $h=\frac{u^2}{g}$ where u is the velocity of the tips of the vanes, and the fan revolves within a casing having an expanding chimney, from which the air is discharged into the atmosphere, with a velocity low enough to be neglected. Hence, in a perfect ventilator the air column representing the initial vacuum h would theoretically be twice the head that corresponds with the circumferential speed of the fan. But imperfections met with in practice require the insertion of a fractional coefficient k less than unity, which Murgue calls the "manometric efficiency," to distinguish it from the fan's mechanical efficiency or useful effect.

When the *blades are inclined backwards* the theoretical depression, expressed in water column, contains a negative expression, and is only equal to

$$h=\frac{u^2}{g}-\frac{uv\operatorname{Cos}a}{g}$$

where v = relative velocity of air upon the blade, and a = the inclination of blade. This accounts for the fact that certain ventilators have been shown to have abnormal efficiencies.[1]

The Useful Effect of Fans.

Let E = the useful effect in horse-power.

Then if Q = the volume of air in cubic feet per minute exhausted by the fan.

h = the height of the water-gauge.

d = the weight of a square foot of water 1 inch deep.

$$\text{Then } E = \frac{Qhd}{33{,}000}$$

But as (p. 176) $h \propto kv^2$ when k is the manometric efficiency and v is the velocity of the air.

The useful effect may be stated as—

$$E \propto vh$$

$$\text{That is} \quad E \propto kv^3$$

Or the power varies as the cube of the velocity of the air-current.

In determining the comparative efficiency of fans it is necessary to take into consideration the efficiency of the motors driving them. The coal consumption in pounds per horse-power should not be above 4, and even that is high. It would not be unreasonable to require of the makers a guarantee that the consumption shall not exceed 2 lbs. per indicated horse-power.

Power required to Drive a Mechanical Ventilator.—As the power P required to put into circulation a given quantity of air is from 1·65 to 3 times the useful effect E in the air.

$$P = 1{\cdot}65\,\frac{vhd}{33{,}000} \text{ to } 3\,\frac{vhd}{33{,}000}$$

The Resistance of the Mine: Murgue's

[1] *Trans. Inst. M.E.*, vol. xxi. pp. 369–370. For fuller details see *Trans. Inst. Civil Engineers*, vol. lxvi. pp. 266–277, and *Bulletin de la Société de l'Industrie Minérale*, Series 2, 1880, vol. ix. pp. 22–71.

Theory.—M. Murgue, the eminent French authority, evolved the theory that the resistance offered by any mine to the flow of air through it may be likened, for the purposes of comparison, to an orifice in a thin plate; that is to say, any mine which is ventilated by a known volume of air under a certain depression or water-gauge is the exact equivalent to an orifice made in a thin plate, which allows of the passage of the same volume of air under the same depression.

By this means resistances of all mines can be compared from complete closure up to, say, a large railway tunnel of 200 square feet sectional area.

Having then the volume of air produced by any ventilator through an equivalent orifice (the latter a matter of calculation), a curve can be constructed where the abscissæ are the equivalent orifices, and the ordinates the volumes of air produced.

By comparing the curves of different ventilators the type of fan best suited for the particular mine can be discovered.

For the purpose of experiments a normal velocity should be taken to which all the fans tested (supposing comparisons are intended) will have to be reduced, say a tangential velocity of 6000 feet per minute at the tops of the vanes.[1]

As has been shown (p. 173), in a theoretically perfect

[1] In speaking of the velocity of revolution of mechanical ventilators, two expressions are frequently used—tangential and angular velocity.

By tangential velocity is meant the velocity of a point on the circumference, *i.e.* speed of the periphery, and this is calculated by the diameter × 3·1416 × revolutions per minute.

By angular velocity is understood the rate at which the fan turns through a given angle. Thus in order that a small fan may have the same tangential velocity as a large fan, it must be made to revolve at a correspondingly quicker rate. The work of a fan depends on its tangential velocity. The tangential velocity being equal, the angular velocity of the fans is in inverse ratio to their diameter.

centrifugal ventilator, the velocity of the circumference produces the theoretical depression.

$$h = \frac{u^2}{g}$$

u being the tangential velocity in feet per second of the fan.[1]

To convert this height into water-gauge, let

d_1 = density of water.
d = density of air.
12 = inches per foot.

$$\text{Then } h = \frac{u^2}{g} \times \frac{d_1}{d} \times 12$$

But in actuality this perfection is not realised, so it is necessary to multiply by a fraction representing the maximum manometric efficiency of the fan.

Thus if k represents this fraction,[2] the actual water-gauge would be—

$$h = k\frac{u^2}{g} \times \frac{d_1}{d} \times 12$$

Then if a = the area of the equivalent orifice
and 0·65 = constant due to the *vena contracta*.[3]

---

[1] The theoretical depression due to the speed of the periphery in the case of a perfect fan is equal to twice the height of a column of air necessary to generate such a velocity in a falling body. The theoretical depression under normal velocities is greater than the actual depression, because of the orifice passage, or obstruction to the flow of air caused by the fan itself, is not taken into account in determining the theoretical water-gauge.

[2] As stated above, $h = \frac{u^2}{g}$ is never actually obtained, but $h = k\frac{u^2}{g}$, k expressing the proportion of the initial depression to the theoretical depression is properly speaking the *useful effect in depression* in ventilation, and is called the *manometrical* yield.

[3] It can be determined by experiment that the volume of water or any other fluid flowing through a hole in a vessel containing the fluid, is not that which would be determined by the product of the rate of efflux and the area of the aperture, but a certain fraction of this theoretical discharge. This is due to a contraction taking place at the point of issue, which is termed the *vena contracta*, so that the actual area of discharge is something less than unity. For water 0·62 is used as the coefficient and 0·65 in the case of air.

$$a = \frac{\text{volume of air per second}}{0{\cdot}65\sqrt{\dfrac{h}{2g \times 12} \times \dfrac{\text{density of a cubic foot of water}}{\text{density of a cubic foot of air}}}}$$

This formula is arrived at by the following train of reasoning—

$$v^2 = 2gh \quad [1]$$
$$\therefore \quad v = \sqrt{2gh}.$$

Consequently the *theoretical* volume of air passing through any orifice of an area a will be—

$$\text{Volume } V = va = a\sqrt{2gh} \qquad (1)$$

and if h represent the height of the depression in inches

[1] This formula is so frequently referred to in mining text-books, that demonstration of its truth may be acceptable to the elementary student. It can be reasoned thus—

Uniform Acceleration.

Let f = increase of velocity for each second.
v = velocity at the commencement of the time.

Then velocity at the end of—

$$\begin{aligned} \text{1st second} &= v+f \\ \text{2nd second} &= v+2f \\ \text{3rd second} &= v+3f \\ \text{and at time } t &= v+ft. \end{aligned}$$

Now in Fig. 81 let AC $= v$, and Aa, ab, bc, &c., represent *intervals* of time, T.

The ordinates at the successive intervals will be obtained by adding to the preceding ordinate the constant quantity fT, represented by $a_1\, a^1$, $b_1\, b^1$, &c. Then if AB represent the complete period of time t, ordinate BD $= v + ft$.

$\therefore$ CD is a straight line.

The space passed over in time t is represented by the area A, B, D, C—that is, by rectangle A, B, E, C, and the triangle C, E, D,

$$\text{or } \overbrace{\text{AC}}^{v} \times \overbrace{\text{AB}}^{t} + \overbrace{\tfrac{1}{2}\text{ED}}^{ft} \times \overbrace{\text{CE}}^{t}$$

$$\therefore \quad s = vt + \tfrac{1}{2}ft^2$$

If at the point of starting velocity v = nothing, then the space is represented by the triangle C, E, D.

$$\therefore \quad s = \tfrac{1}{2}ft^2.$$

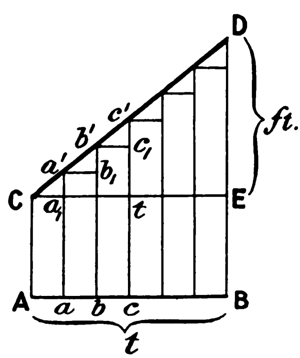


FIG. 81.—Diagrammatic Explanation of Uniform Acceleration of Falling Bodies.

Falling Bodies.

A ball of lead and a scrap of paper fall through air with very different velocities.

If bodies fall in a tube from which air has been expelled, the *velocity acquired* of water of density d_1 and d is the density of the air, and h_1 motive column in feet of air—

$$h_1 = \frac{h}{12} \times \frac{d_1}{d} \qquad (2)$$

Or substituting the value h_1 in (1)—

$$V = a\sqrt{\frac{2gh}{12} \times \frac{d_1}{d}}$$

And allowing for the *vena contracta*—

$$V = \cdot 65a\sqrt{\frac{2gh}{12} \times \frac{d_1}{d}}$$

From which the value of a can be determined, as in the formula.

In order to compare two machines, they are regulated, as has been said, to the same speed (6000 feet per minute normal velocity) of periphery, or their results may be reduced to equal speeds, for the volumes vary as the revolutions, and the depressions (heights of water-gauge) as the square of the speeds.

There is an obstruction in the fan itself, which Murgue terms the "orifice of passage."

The characteristic curve of a perfect fan would be a straight line; the nearer the curve approaches to a straight line the more perfect is the fan.

The natural ventilation of the mine has to be taken

---

is the same for all bodies, *i.e.* the motion of all bodies *in vacuo* is uniformly accelerated.

The force producing the motion is "gravity," and the acceleration is indicated by the letter g. So g=number of feet added to the velocity of a body moving freely *in vacuo* for every second of time. (In London a velocity of nearly 32·2 feet is added in every second when a body moves *in vacuo*.)

Putting g for f,

$v = gt$ and $s = \frac{1}{2}gt^2$

Then $v^2 = g^2t^2$

and $S = \frac{1}{2}gt^2$ or $2S = gt^2$

But $v^2 = g^2t^2$

So $v^2 = 2Sg$ or $v^2 = 2gh$

(h and s being the same).

into consideration in conducting experiments on fans. As in the calculation it increases the volume of air, but diminishes the size of the equivalent orifice—the one factor, however, about balances the other.

Thus, as air expands 0·002036 of its volume for every 1° F., if the length of the column of air is 50 feet, and the temperature of the upcast shaft is 20° warmer than the downcast—

$$20 \times \cdot 002036 \times 50 = 2\cdot 036 \text{ feet}$$

so that 52·036 cubic feet of the warmer air will balance 50 cubic feet of the cooler air, and the velocity of efflux of the heated air column when pressed by the greater weight of the cold air column will be equal to—

$$8\sqrt{2\cdot 036} = 11\cdot 036 \text{ feet per second.}$$[1]

Suitability of a Fan to a Mine.—Let o = the orifice of discharge of a fan in square feet. Then if $\frac{a}{o}$ is equal to, or greater than, unity, the fan is too small for the resistances of the mine. When the value is less than unity, 0·5 is considered as representative of a fair working ratio. The most favourable ratio is 0·3.

By this means the dimensions of a fan can, to some extent, be calculated.

The speed per second of the rim of the fan u, required to produce volume Q and water-gauge h, is determined by the formula—

$$u = \frac{Q}{40}\sqrt{\frac{a^2 + o^2}{2k \times a^2 \times o^2}}$$

in which k is a fraction representing the maximum manometric efficiency produced by the fan.

[1] *i.e.* $h = \frac{v^2}{2g}$ or $v = 8\sqrt{h}$.

Calculation of Dimensions of Fans and Fan Engines.

To find the diameter of the cylinders of the engine for driving fans.

If D = the diameter.
Q = the quantity of air in cubic feet per minute.
P = the power necessary to drive the fan.
h = the height of the water-gauge in inches.

$$\text{Then } D = \frac{Q \times h}{\cdot 7854 \times P \times 250 \times g}$$

To find the diameter of fans for a given quantity of air.

The following formulæ have been given, but are of value only in the case of moderate volumes of air :—

In the case of large fans—

$$D = \frac{Q - 30{,}000}{8000} + 18$$

In the case of small fans—

$$D = \frac{Q - 50{,}000}{25{,}000} + 8$$

In all fan calculations it must be remembered that the water-gauge varies as the squares of the quantities, and the horse-power as the cube of the quantities.

Scheme of Fan Experiments.—In experimenting with fans numerous data have to be provided before the results of the experiments can be worked out and tabulated. The following form is therefore given, with notes thereon, as likely to be of some help to young experimenters :—

Mine.......................... Date...............
No........... Experiment..............

CONDITIONS

Temperature of external atmosphere.	Wet bulb.........	Dry bulb.........
Temperature of air in fan drift.	Wet bulb.........	Dry bulb.........
Temperature of air at bottom of shaft.	Wet bulb.........	Dry bulb.........
Pressure—atmosphere, *i.e.*	Inches of mercury.	

OBSERVATIONS

Duration of experiment ....minutes.
Mean revolutions of engine...................per minute.
Mean revolutions of fan ....per minute.
Mean revolutions of anemometer ....per minute.
Water-gauge, No...... mean depression at inlet of fan ......in.
 „ „ fan drift ......in.
 „ „ ......feet from fan ......in.
 „ „ in.
 „ „. at bottom of shaft ......in.
Indicator diagrams, mean pressure in lbs. per square inch.

No.	Back end.	Front end.	Average.
„	....	....	....
„	....	....	....

CALCULATIONS

Weight of a cubic foot of air of the atmosphere ......lbs.
Weight of a cubic foot of air of fan drift ....lbs.
Weight of a cubic foot of water...........................lbs.
(Calculated thus—
Weight of air, &c., from Glaisher's tables,
say air in drift averages...............58° (saturated)
air at surface averages............45°.
Weight of 1 cubic foot of saturated air at 58° F. temperature and 29·75 inches barometer, from Glaisher's tables ......531 grs.

$$\frac{531}{7000} = 0{\cdot}07587 \text{ lb.}$$

Weight of 1 cubic foot of water at 45° F. = 62·442 lbs.
Indicator diagrams (spring, say, $\frac{1}{20}$th).
Calculated thus—

$$\frac{\text{Area of indicator diagram} \times 20}{\text{diameter}} = \text{mean pressure.}$$

(3) Velocity of circumference of fan, viz.:
Tangential velocity (see footnote, p. 175)
= diameter of fan × 3·1416 × revs. per minute.
N.B.—When comparing one fan with another a *normal* velocity is assumed throughout the experiments of say 6000 feet per minute.)

(4) Mean volume of air............
(viz. corrected revolution of anemometer × cross-sectional area of drift).

(5) Volume of air at normal velocity............
(6000 ÷ length of circumference of fan in feet = revolutions of fan at normal velocity.)
∴ volume of air at normal velocity = observed volume in cubic feet per minute × $\frac{6000}{\text{speed of periphery}}$

(6) Water-gauge at normal velocity.
(As velocity ^{2} : normal velocity ^{2} : : observed water-gauge : water-gauge at normal velocity.)

(7) Theoretical water-gauge.................. (see p. 176).

(8) Manometric efficiency of fan............ (see footnote, p. 176).

(9) Equivalent orifice .... (see p. 176).

CHAPTER VIII

THE DIRECTION OF UNDERGROUND AIR-CURRENTS—VOLUMES AND VELOCITIES—STOPPINGS, DOORS, AIR-CROSSINGS, REGULATORS, VENTILATION PLANS, VENTILATION OF MINES UNDER DIFFERENT SYSTEMS OF WORKING—VENTILATION OF METALLIFEROUS MINES

Underground Ventilation.—The end to be secured by underground ventilation is the carrying of an adequate volume of air—as pure as possible—through those parts of the mine in which persons have to work or pass, with as little loss by leakage as possible. Involved in this consideration are the questions of quantity, purity, and velocity.

In determining the quantity of air necessary, the mine manager will be guided by the number of men and animals that have to be served by the current; and the nature of the mine in respect of the production of fire-damp or other gases, absorption of oxygen by the coal and strata, humidity, and temperature: the extent to which shot-firing is carried on, the distance of the working places from the shafts, and consequently the loss by leakage which the volume suffers between the point of inlet and the working face. In arranging the number of splits he will be guided by the number of separate districts and their size.

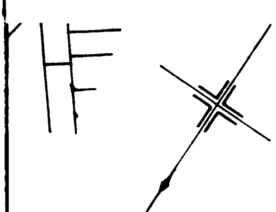
A mine, unlike a factory, is never stationary in respect of the area covered by its operations; it is either advancing or retreating, or, it may be, that it is advancing in some directions and retreating in others. Hence the ventilation is also a matter of variation. Not only, therefore, is a general ventilation plan desirable (see coloured plate), but it is advisable to keep a record of projected ventilation schemes, so that when it is necessary to alter the ventilation of any district, owing to its advancement or retreat, the manager, having sketched out the alterations he deems advisable, these, with the date on which they are to be effected, being recorded, there is available, for the officials that have to carry out the same, a definite scheme to guide them. An example of what is meant by this is shown by Figs. 82 and 83.

The author has found this a most useful arrangement, and one highly educational to under-officials. It is difficult to explain ventilation schemes by word of mouth, and the misplacement of a stopping or door, or the omission of the same, may make all the difference in the scheme. Ordinarily an explanation of any desired change is accompanied by a chalked out sketch underground, but it is far better that such should be afterwards carefully considered and sketched in the manager's office, and a record kept in the form of plans.

Adequate Ventilation: The Volume of Air necessary and the Velocity of the Current in the Mine.—This matter has already been touched upon in pages 16, 27, and 80 of this volume, but may again be referred to more fully.

Dr. Haldane has stated[1] that 100 cubic feet of air would suffice to keep a man alive for ten hours, this would be equivalent to 288 cubic inches per minute, but as

[1] *Causes of Death in Colliery Explosions*, 1896, p. 33.



Longma

Vincent Brooks, Day & Son Ltd lith.

the number of respirations of a healthy man, between the age of 25 and 30 years, is 16 per minute when passive,

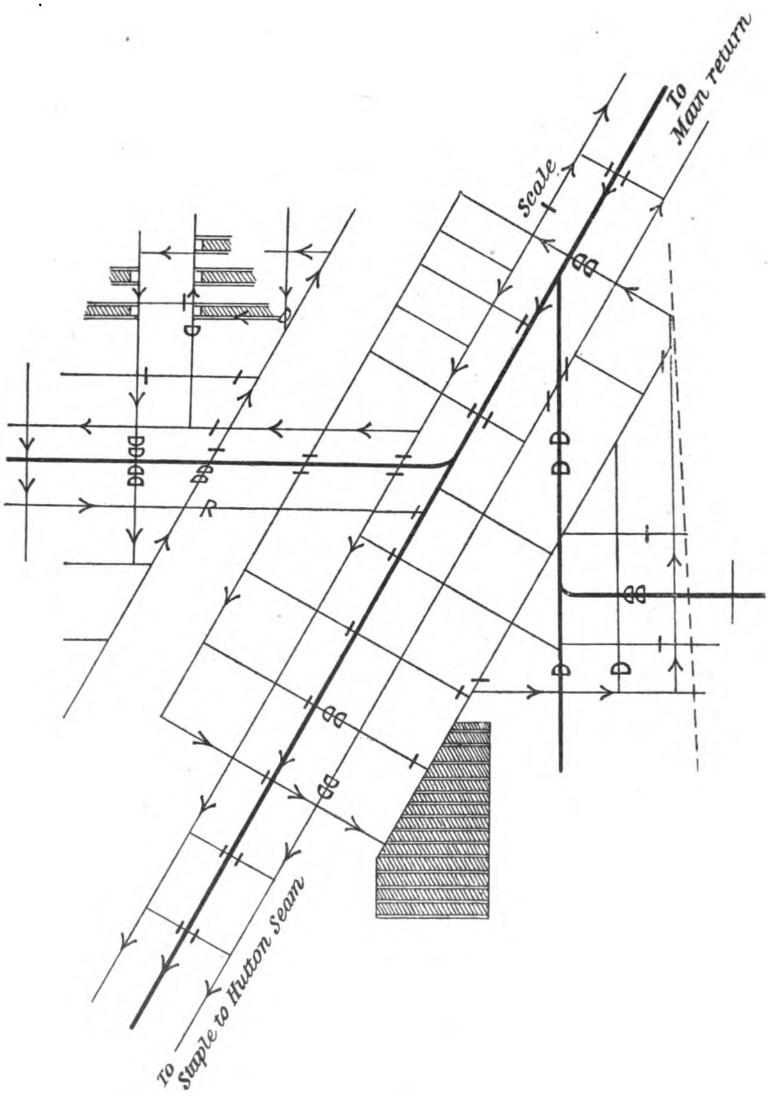


FIG. 82.—Sketch Plan of an existing System of Ventilation of "Bord-and-Pillar" and "Double-Stall" Workings, all of which are ventilated by the same Air-current.

and he takes in 34 cubic inches of air with each respiration, he will require, under normal conditions, 544 cubic inches per minute, so in the case put by Dr. Haldane he would have to breathe the air twice over; but as Dr. Clowes has shown, air may be breathed three or four times over before it becomes uncomfortable to the breather. As he breathes, he vitiates the air by exhaling carbon dioxide to the extent of 21·76 cubic inches per minute, and as ordinary air contains but ·04 per cent. of this gas, $21·76 \div ·04 = 544$ cubic inches of fresh air will be necessary to purify that which is vitiated, and $544 + 544 = 1088$ cubic inches of fresh air per minute will be required by each person in the mine when quiescent. This figure should be multiplied by 32 when the person is working = about 20 cubic feet per minute.

Horses when working will require 90 cubic feet per minute, and each safety lamp will require $\frac{1}{5}$ cubic foot per minute. But the rate at which noxious gases are given off naturally in the mine has an important bearing on the volume of air necessary to properly ventilate the mine, and those occasioned by blasting have also to be taken into consideration. It may be taken that a minimum of 500 cubic feet of air is required for every shot fired in the mine to properly dilute the gases due to the explosion. The firedamp should be kept as low as possible, and it is desirable that it should be kept down to $\frac{1}{2}$ per cent. in the main return air-way.

Working on these figures, Mr. E. W. Thirkell, in his very interesting paper summarising the researches of Special Commissions,[1] arrives at the following useful conclusion:—

[1] "Adequate Ventilation, and Noxious Gases: with Special Reference to the Recommendations of the English, French, Prussian, and Austrian Firedamp Commissions," by E. W. Thirkell, *Trans. Inst. M.E.*, vol. xiii. p. 403.

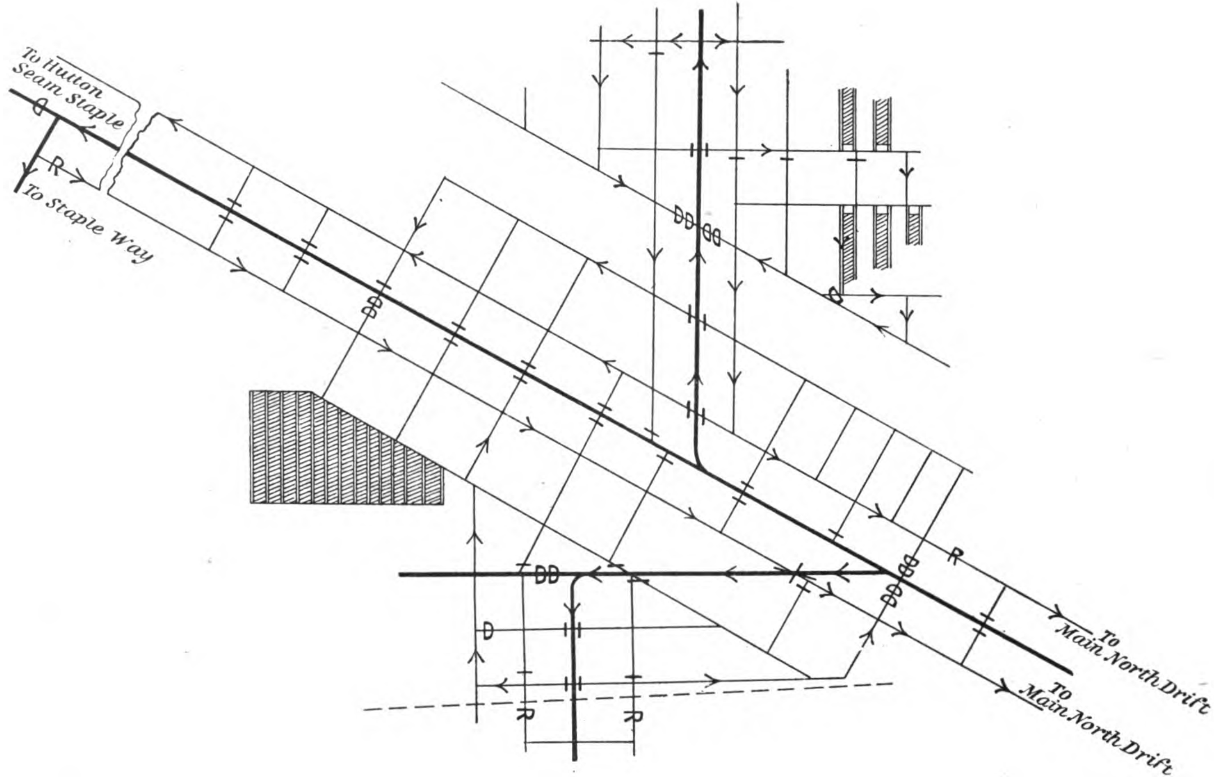


FIG. 83.—Sketch Plan of a projected change in the Scheme of Ventilation of the Workings shown in Fig. 83, showing the Ventilation of each District by a separate Split necessary owing to the development of the Workings.

TABLE XX.—*Scale of Volumes of Air required for Persons, Animals, &c., under Different Conditions, in respect of Gas.*

Each person requires ordinarily when at work	20 cub. ft. per min.
Each person when in a working place giving off $\frac{1}{2}$ per cent. of gas	40 „ „
Each person with $\frac{3}{4}$ per cent. of gas, to reduce it down to $\frac{1}{2}$ per cent.	55 „ „
Each person with 1 per cent. of gas, to reduce it down to $\frac{1}{2}$ per cent.	80 „ „
Each person with $1\frac{1}{8}$ per cent. of gas, to reduce it down to $\frac{1}{2}$ per cent.	107 „ „
Each person with $1\frac{1}{4}$ per cent. of gas, to reduce it down to $\frac{1}{2}$ per cent.	115 „ „
Each person with $1\frac{1}{2}$ per cent. of gas, to reduce it down to $\frac{1}{2}$ per cent.	160 „ „
Each horse requires	90 „ „
Each safety lamp requires	$\frac{1}{5}$ „ „

The Austrian Commission[1] stated that the return air should not contain more than $1\frac{1}{2}$ per cent. of firedamp and $\frac{1}{2}$ per cent. of carbon dioxide, and hence that the quantity of air required is as follows:—

In Mines coming under Class I.—*i.e.* those producing little firedamp (under 1 per cent. $CH_4 + CO_2$ in the return air), at least 70 cubic feet per man per minute, calculated on the largest shift, and counting a horse as equal to four men.

In Mines under Class II.—*i.e.* those producing a moderate amount of firedamp (1 to 2 per cent. of $CH_4 + CO_2$ in the return air[2]), at least 105 cubic feet per man per minute.

In Mines under Class III.—*i.e.* those producing much firedamp (more than 2 per cent. of $CH_4 + CO_2$ in the return air), at least 140 cubic feet per man per minute.

The Prussian Commission[3] state that the volume of

[1] *Trans. Inst. M.E.*, vol. iii. p. 542.

[2] That is to say, 1 to 2 per cent. of $CH_4 + CO_2$ calculated to exist were the mine ventilated by a volume of air based on the standard of Class I.

[3] *Trans. Inst. M.E.*, vol. iii. p. 1105, vol. iv. p. 631, vol. v. p. 500.

fresh air should amount to at least 70·62 cubic feet per head per minute on the maximum number of men in a shift, a horse being regarded for this purpose as equal to four men; furthermore, an allowance to the air supplied to each separate ventilating district should be made to the amount of 52·95 cubic feet per ton of average daily output of coal, and the gaseous contents of the return air-ways should be reduced to 1·5 per cent.

The *velocity* of the air in the shaft should not, according to the *Austrian Commission,* exceed 20 feet per second, or 33 feet per second if the shaft be used for ventilation only, and the return air should be calculated to exceed the intake in *volume* by 15 per cent. The *velocity* of the air-current in main roads should not exceed 20 feet per second, the space occupied by the tubs, &c., to be allowed for in this calculation. Not more than 100 workmen should be employed in one split of air, which is in accordance also with the Pennsylvanian (U.S.A.) Mine Law.

The *Prussian Commission* differ little from the Austrian in respect of velocities and volumes of air. They condemn high velocities, and say that air-ways should be of a cross-sectional area sufficient to allow of a low velocity. Main air-ways should not, in their opinion, be less than 32·29 square feet in area, and branch air-ways 21·52 square feet.

Main Features of Ventilation Systems.— Speaking generally, it is an advantage to arrange the return air-ways so that the main intake is skirted on either side by return air-ways. The *draw* of the intake air is then on both sides of the intake, *towards* the returns, a fact which acts as a preventative of the leakage of firedamp on to the intake, which is usually also the haulage road (and, very frequently, the travelling way also), and so the scene of much traffic, and hence a part of the mine where the air should be as pure as possible. It is not intended by this that there should be any preventable leakage of intake air to the returns, but if there is leakage (and do what one will, there will always be a certain amount of leakage), it should be ensured that it is *from* the intake or intakes *to* the returns. But such an arrangement cannot always be secured. Thus in a mine worked by the longwall system, where the coal is entirely removed, it is very difficult, and in many cases impossible, to secure this desideratum, owing to the openness of the goaf, so that no definite rule can be laid down in this respect.

The air-currents in a mine must have a definite and assured course, and must not be liable to variation or reversal owing to variations in pressure. This direction is secured by means of stoppings, doors, and air-crossings; and the quantity allowed to the several districts is in some measure determined by regulators or by scales.

Stoppings.—These are either permanent or temporary. Permanent stoppings constructed of stone or brick and lime are used in those cases where the air-current is to be maintained in a particular course for a long period, *e.g.* along main intakes of the mine and districts of the mine, and in the case of "bearing up" stoppings carrying the air on to the regulator. They are usually made 9 inches or 14 inches thick, built flat as a wall, the side, top, and bottom being let into firm ground, so as to prevent leakage, and the stopping well plastered over for the same reason. These stoppings should be easy of access, and not be stowed up and hidden by rubbish, as it is most necessary that excessive leakage may at once be detected and the stopping repaired. Stoppings are sometimes built as shown in Fig. 84, the object being to render them so strong as to resist explosive force. After a colliery explosion, as a rule, all or nearly all of the stoppings within the zone of great explosive violence are blown out, and have, during the work of rescue and reclamation of the mine, and at the expense of great loss of precious time, to be repaired in a temporary manner in order to restore the ventilation and enable the rescuers to get into the workings. The double form of stopping with the space between the walls packed with stones, constitutes a very strong form of stopping, and such as are suitable for main air-ways.

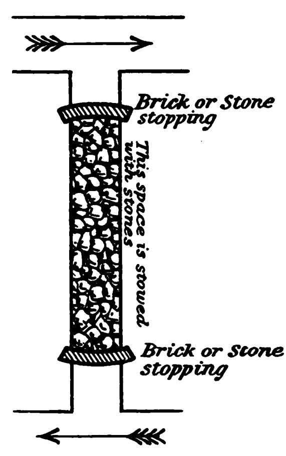


FIG. 84.—Strong Form of Stopping suitable for Main Roads.

It has been advanced that it may be advantageous to have the stoppings so constructed that they will be blown out should an explosion occur in the mine, as they would then act as safety valves, and allow of the explosive force being expended, and so tend to its being confined to fewer of the main roads of a mine. There might be some force in this contention were the explosion one due to firedamp alone, but in view of our later knowledge of colliery explosions, and the manner in which they are carried on through the mine by coal dust, the argument loses a good deal of its point. The stoppings used for "face airing" (*i.e.* keeping the air up against the face of the workings) being of a very temporary nature, as the face is always advancing, are made of either wood or canvas. Figs. 104, 105, and 106 show the relative positions of permanent, bearing up, and face stoppings.

Bratticing Air-pipes for Ventilation of Winning Places.—Face stoppings only serve to bear the air-current up to the innermost roads, but do not carry it against the actual coal face of winning places driven in advance of the general face, or in the formation of pillars in the bord-and-pillar system of working. The ventilation, therefore, of such places has to be carried out by means of air-boxes, air-pipes, or bratticing. Fig. 85 shows the mode of bratticing a "pair of places"; Fig. 86 four ways of carrying the current up to the face of three parallel winning places.

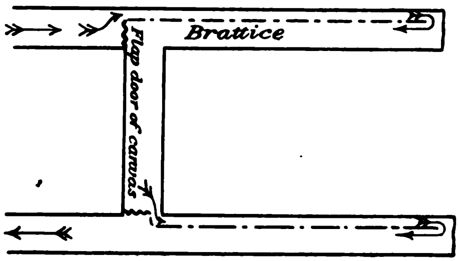


FIG. 85.—Ventilation by means of Bratticing of two Winning Places.

Doors.—Doors in mines, where used for directing the air-current, are usually of two kinds: separation or wooden doors, and check or hanging canvas doors. The former are used where the erection is to be of the more permanent character. Thus where used as a separation

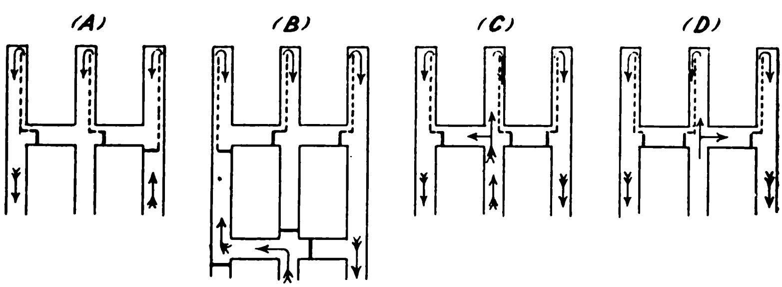


FIG. 86.—Four Ways of carrying the Air-current up three Winning Places.

In *A* and *B* the intake air is conducted up the narrow side of the brattice, while in *C* and *D*, in the middle places, it is carried up the wide side.

between air-currents in a road leading directly from the intake into the return air-way of a mine or district, the doors should be two in number, constructed of wood, and set in a light wooden, or better still, brick-built frame, with wooden jambs, and be locked. Such doors will in all probability have to remain for many years, but where doors are used in the workings, and for the purpose of forcing the air up against the face, *e.g.* at the outbye end of wooden or canvas bratticing in bord and

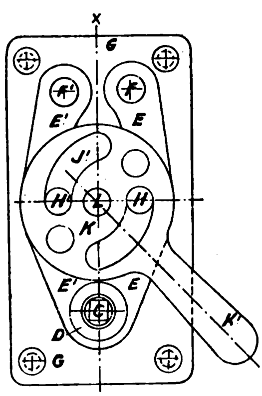


FIG. 87.—Noble's Lock for Mine Doors: Locked.

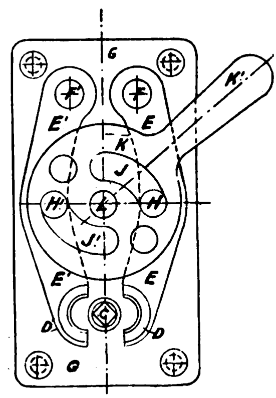


FIG. 88.—Noble's Lock for Mine Doors: Unlocked.

pillar "whole" workings, or in the gateways of longwall workings, they are usually of canvas hung from wooden "sconces" or frames, as the pressure of the air-current in the workings being small, the amount of leakage is not great; indeed, in some circumstances, it is an advantage that there should be a certain amount of leakage, especially in the case of longwall working (Fig. 99), where it is desirable that air should pass along the gates and cross-gates; and in double-stall workings for similar reasons.

The wooden doors when used as means of ingress and egress from one main ventilating road to another should be locked, and for this purpose a screw bolt is commonly used, which has to be unscrewed before it can be extracted, necessitating a special form of key which fits the square head of the bolt.

Mr. Noble, the manager of Charlaw Colliery, has, however, devised a better kind of unlocking arrangement,

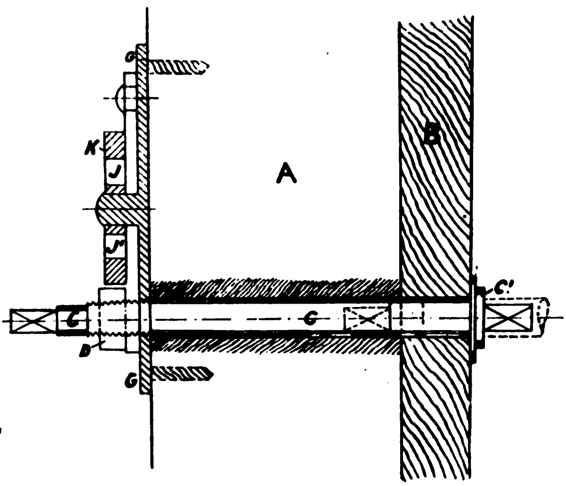


FIG. 89.—Noble's Locking Device for Mine Doors. Section along line *xx* showing relative positions of door-frame A, door B, and bolt C.

so that from one side the door can be opened without the aid of a key. The arrangement is as follows:—

Opening Locked Ventilation Doors without a Key.—The locking device mentioned above, and illustrated in Figs. 87, 88, and 89, consists of a screw bolt through the door and frame, and is screwed from the intake or outside, with a key in the possession of the authorised person, into a nut forming part of the unlocking device, the unlocking in cases of emergency being effected by a pair of clamping levers, preferably of the second order, on separate fulcrums, on each of which levers is formed one-half of the tapped nut for the screw bolt. Normally these levers are closed and secured by a double-ended slotted lever with a handle, but in an emergency this lever can be turned partially round, and, by the action of the inclined slots, separates the two levers on their fulcrums and opens the tapped nut, so the screw bolt is released from it, and the door can be pushed open from the return or inside, and persons pass from the return to the intake. Fig. 87 is a face view of the device as locked; Fig. 88 is a face view showing the device unlocked in case of emergency; and Fig. 89 is a section along line *xx* in Fig. 87. A is the door-frame, B the door fitting tightly against the frame A, C the bolt with collar C^1 on one end and screwed on the other end through a tapped nut DD^1 formed in two halves upon the ends of the levers EE^1, of which FF^1 are the fulcrums on a plate G secured on the frame A. HH^1 are studs upon the levers, which work in eccentric slots JJ^1 opening the levers EE^1 from the screw threads of the bolt C.

Air-crossings.—In order to carry one air-current across another one, bridges have to be made, which are commonly known as air-crossings or overcasts, and are constructed in various ways, according to the strength and durability desired.

There can be no doubt that all main air-crossings in the neighbourhood of the shafts should be constructed in the strongest possible way, "so that in the event of an explosion the ventilation to the different districts, which usually diverge from the main lines beyond such crossings, may be the more readily restored."[1] But it is doubtful whether any crossing short of one made in the solid rock

[1] *Colliery Working and Management*, by H. F. Bulman and R. A. S. Redmayne, 2nd edition, p. 145.

(see Fig. 91) would be of strength sufficient to withstand the enormous violence generated by some colliery explosions. In addition to that cut out of the solid strata, air-crossings may be constructed—

(*a*) One current of air may be conveyed across another

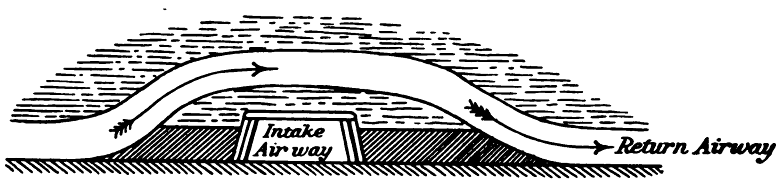


FIG. 90.—Air-crossing made in the Solid Rock.

by means of a pipe or pipes (see Fig. 91), which must, however, be regarded as only a form of temporary crossing, and should not be used in the case of the main and more important splits, but merely for conveying small volumes of air.

(*b*) The common type of air-crossing is constructed of stout deals (say 3 inches by 11 inches) supported by brick side walls 9 inches thick, the side walls on the return side being backed with débris derived from the space which has been excavated to make the air-crossing (see Fig. 92). The deals are usually dovetailed into each other and clamped with iron bands to prevent leakage. The cost of making such an air-crossing depends largely on the height of the coal seam, but in the case of a main air-crossing in a mine where the seam was about 3 feet 6 inches thick, necessitating therefore in its construction a good deal of stonework, the total cost would

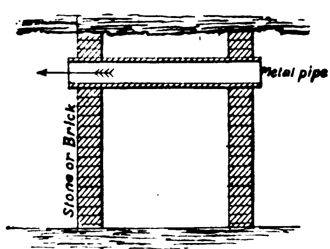


FIG. 91.—Metal Pipe Air-crossing. (From Bulman and Redmayne's *Colliery Working and Management.*)

be about £30, divided thus—as to stonework, £20; material and building, £10.

Sometimes, though rarely, the wooden top is made hinged like a door, either in one piece or in two halves,

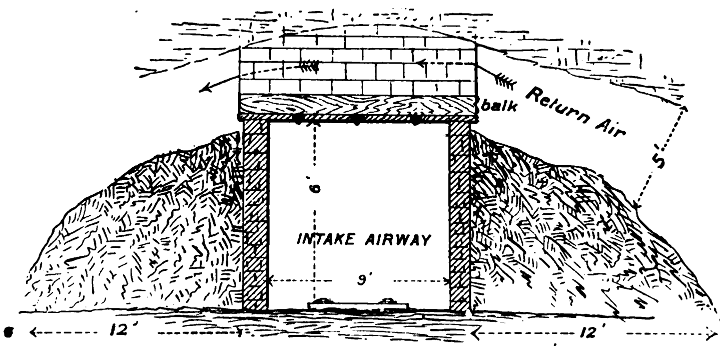


FIG. 92.—Wood-topped Air-crossing in Section. (From Bulman and Redmayne's *Colliery Working and Management.*)

opening upwards, and so arranged that they will readily fall back and close tight, the object being to allow of the dissipation of the violence caused by an explosion, and the preservation of the air-crossing intact; but judging from what one has seen of the effects of an explosion on doors in a mine, it is more than doubtful whether the object aimed at would be secured. The hinged tops would in all probability be blown to pieces.

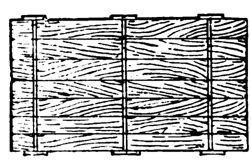


FIG. 93.—Plan of Top of Wood-topped Air-crossing. (From Bulman and Redmayne's *Colliery Working and Management.*)

(*c*) Sometimes, with the view of greater strength, the crossing is made arched as in Figs. 94 and 95, or an inverted-arch crossing, that is the ordinary arched crossing, with an inverted arch placed in the return, which is the strongest form of air-crossing that is made in mines short of tunnelling. Apart from the question of being explosive proof,[1] and it is doubtful whether even the inverted-arch crossing would withstand the violence of an explosion,

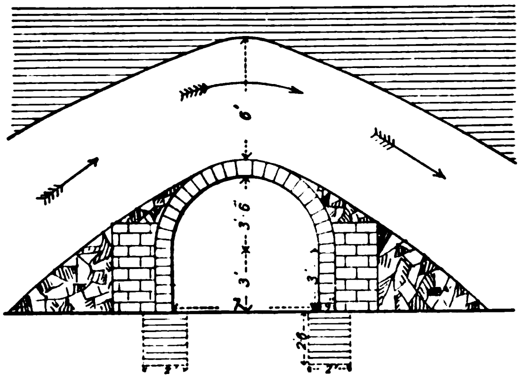


FIG. 94.—Brick-arch Air-crossing, in Section.
(From Bulman and Redmayne's *Colliery Working and Management.*)

this form of air-crossing is sometimes adopted where the strata are liable to movement due either to great pressure, or the presence of a soft and thick underclay floor

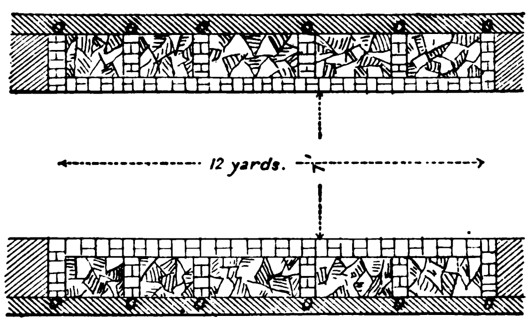


FIG. 95.—Brick-arch Air-crossing, in Plan.
(From Bulman and Redmayne's *Colliery Working and Management.*)

to the seam of coal. "Instances have occurred, however, where even this form of air-crossing has not been strong

[1] In the Elemore Colliery explosion, December 1886, several of the brick-arched air-crossings were completely blown out.

enough to resist the upward pressure of the floor, and has required renewal after a few months."[1]

Sometimes a door is placed in the side of the air-crossing as shown in plan and cross-section in Figs. 96 and 97, enabling the return to be entered at this point from the intake or *vice versa*.

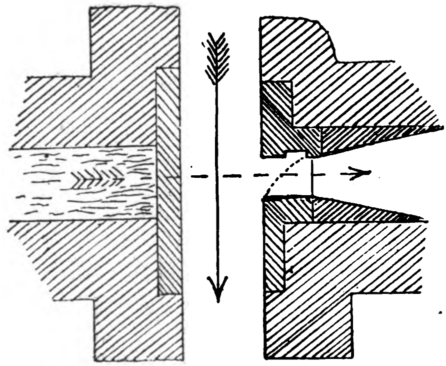


FIG. 96.—Special Form of Air-crossing, in Plan. (From Bulman and Redmayne's *Colliery Working and Management.*)

Regulators.—Some districts in the same seam by reason of closer proximity to the shafts, and hence more favourably circumstanced in respect of total frictional resistance, would, were not the flow of the air-current artificially obstructed, take an undue proportion of the total air-current available in the mine. It is necessary, therefore, to regulate the

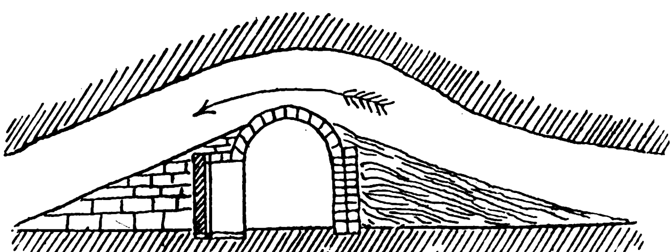


FIG. 97.—Special Form of Air-crossing, in Section. (From Bulman and Redmayne's *Colliery Working and Management.*)

splits, which is done by placing, usually for the sake of convenience, in the return air-ways a sliding door, by which the aperture (orifice of passage) can be

[1] *Colliery Working and Management*, p. 148.

diminished or increased as the requirements in point of area of workings, distance from the shafts, or generation of firedamp of the district may demand. The regulator commonly takes the form shown in Fig. 98. The slide, when moved into the position necessary to give to the district the desired volume of air, is locked in that position by means of a screw bolt. Of course the ideal condition would be to so arrange and advance each district of the mine that it should take from the main supply of air a split of a volume sufficient to meet its requirements without the interposition of regulators in any of the districts. But in practice it is seldom possible to carry out this ideal.

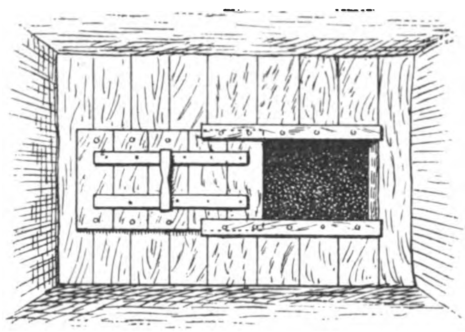


FIG. 98.—Regulator.

The Position of the Regulator.—The position of the regulator in the district is a matter of considerable importance. As a general rule it should be as near to the advancing face as is conveniently practicable, as the air pressure on the stoppings, from the intake to the return, is then greater than if the regulator were placed on the outbye side of these stoppings.

Leakage from the Air-current.—The loss by leakage of air from the current in its course from the bottom of the downcast shaft to the workings is considerable. Mr. Henry Palmer[1] mentions a case where this amounted to no less than 84 per cent., but this must be

[1] "Notes on a Ventilating Current," by Henry Palmer, *Brit. Soc. Min. Stud.*, vol. xi. p. 46.

regarded as an abnormal instance. Messrs. Pernolet and Aguillon, the representatives of a French Commission, instructed to report on the best means to prevent explosions of gas in collieries, state in their report, which appeared in the year 1881, that, in respect of British collieries, "from the daily registers we learn that in many pits from 25 to 50 per cent. of the air goes down one pit and up the other without penetrating the workings, being used for stables, boilers, &c. When not lost by simple leakages in doors, stoppings, and crossings, and such air as does go inbye has losses of its own, often caused by the intakes and returns running parallel to each other for many thousands of yards, only separated by stoppages of a very doubtful tightness." The advantage of the close proximity of parallel returns and intakes has, however, already been pointed out, and there is no reason why, with well-built stoppings and crossings, the leakage should amount to say more than 20 per cent. in a large mine. The less frequent the holings ("thirls, stentons," &c.) between the intakes and returns when parallel, the less will be the number of stoppings, and the less the leakage.

In collieries worked on the longwall system, in which the coal, ironstone, or fireclay is completely worked out in one working, the roads pass through the waste ("goaf" or "gob")—unless, indeed, the retreating system of longwall is adopted—in which case it is very difficult indeed to prevent considerable leakage of air, and adopt the "skirting" arrangement of intake and return air-ways, especially if the roof and floor are such as do not allow of the "waste" closing tightly and soon. Under these circumstances it will be found desirable to drive out the winnings as narrow places, leaving pillars of solid coal between intake and return, and work the longwall from a "barrier" place, leaving a wide barrier of solid coal between the return and the "waste" (see Fig. 99).

The leakage from the air-current in the case of an old mine may be such as to necessitate the forming of the main return air-way or air-ways in another seam of virgin coal lying above or below the seam being worked, and to connect the district returns with it or them.

The Disposition of the Air in Different Systems of Working.—There are a number of

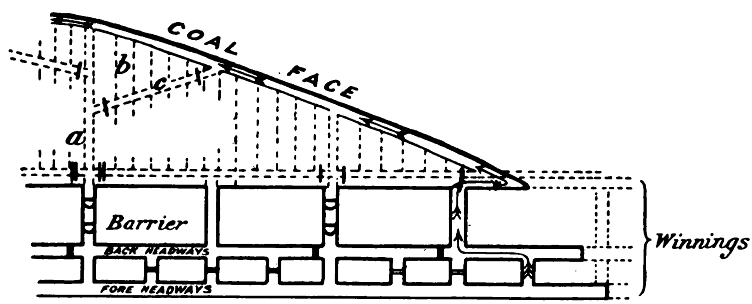
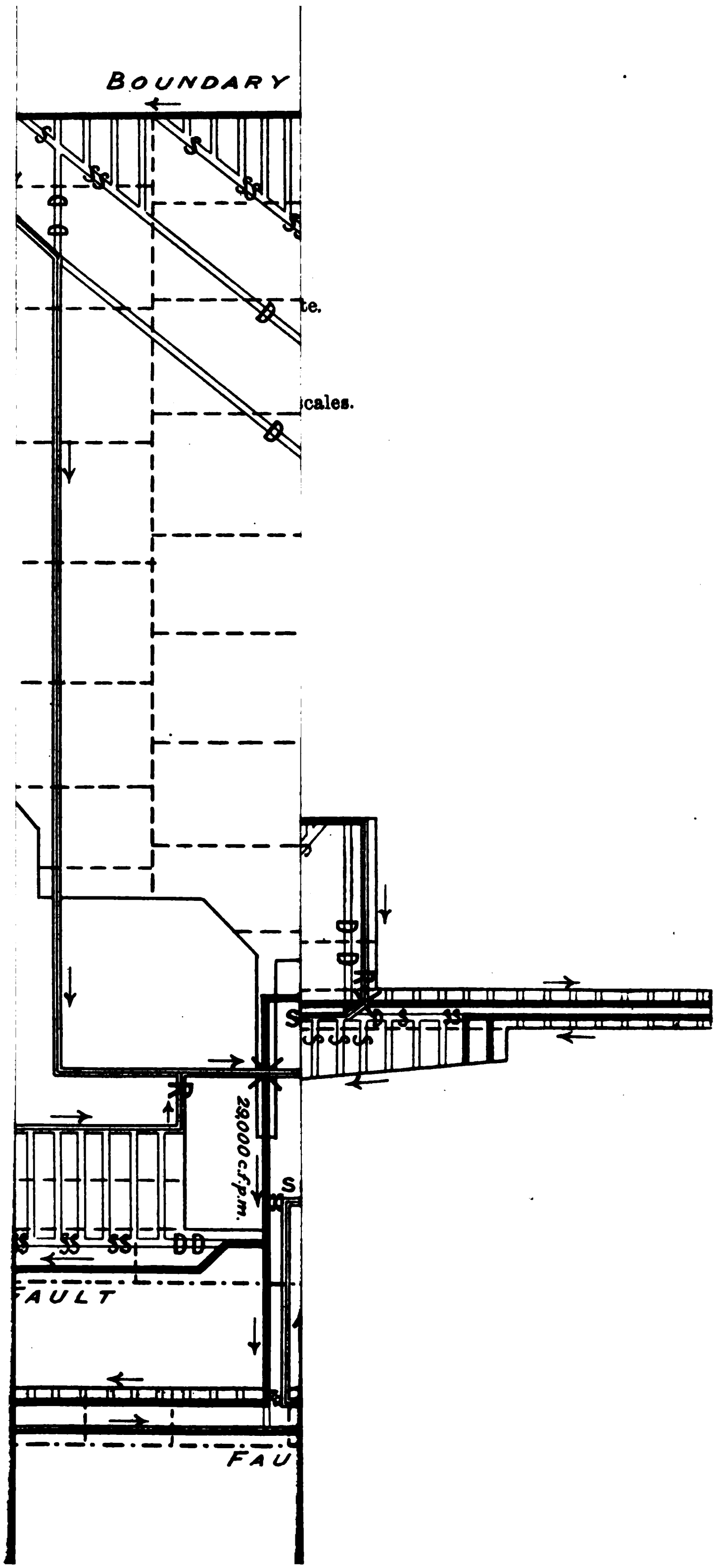


FIG. 99.—Longwall Worked from Barrier Place.
(*a*) is the mother gate; (*b*) gateway; (*c*) cross-heading.

methods of working coal and other stratified minerals (see *Methods of Working*, vol. iii.), but, broadly speaking, all of these are modifications of one or other of the principal systems, bord-and-pillar (stoop-and-room) and longwall.

The coloured folding plate (p. 184) represents the ventilation plan of a seam which is worked on the bord-and-pillar system, the blue lines indicating the intake air, and the red lines the return air. The reader's attention is directed to the important fact that the main intakes (which are also the haulage roads) are skirted on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either side by return air-ways, one pillar distant, so that any gas issuing from the old workings will pass into the returns and be swept away, and there is no risk of its leaking on to the haulage roads, as would be the case were there a return on one side only of the intake. The plan is not that of an imaginary instance, but of an

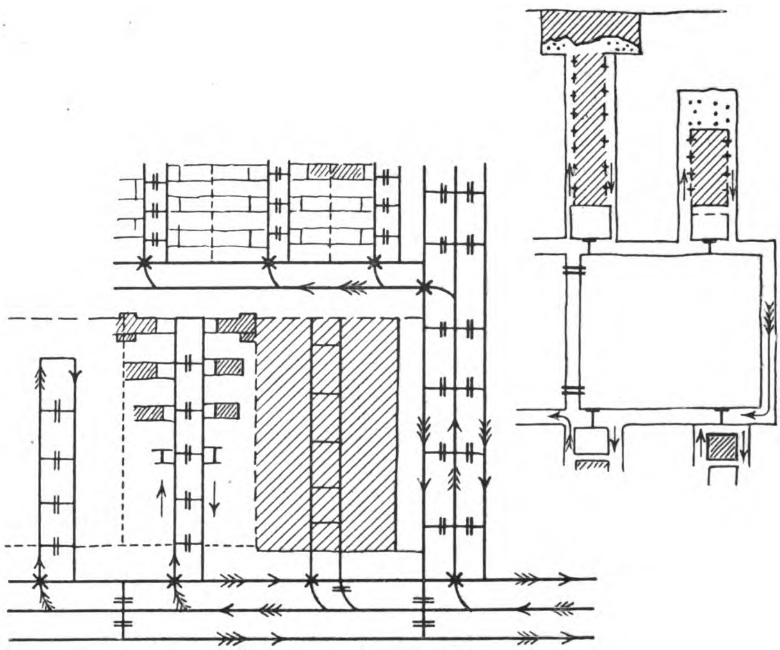


FIG. 101.—Plan showing Ventilation of Double-stall Workings.

actual working colliery, and may be taken as representative of good mining practice. The position of the stoppings, doors, crossings, and regulators should be noted by the young mining student.

The folding plate (Fig. 100) represents the ventilation of a seam worked on the longwall system. For this plan I am indebted to Mr. W. E. Garforth, the Chairman of Messrs. Pope and Pearson's Collieries in Yorkshire.

This, again, is illustrative of actual conditions, and may be taken as a model in respect both of mode of working and ventilation of a seam of moderate inclination.

Fig. 101 indicates the disposition of the air-current in the case of a seam worked by the double-stall system, which is a modification of the bord-and-pillar system.

The Relative Advantages, considered from the Point of View of Ventilation only, of Working by the Bord-and-Pillar and Longwall Systems.—For a given output of coal from a given seam the ventilating current, in a mine worked by the bord-and-pillar system, will have to travel a much greater distance, and hence have to contend with a much greater frictional resistance than in the case of a mine worked on the longwall system. As it will have to travel in and out of the "whole" places or "broken" workings instead of traversing a continuous, and more or less straight, face, as would be the case were the seam worked by longwall, the leakage also at the numerous bratticings and canvas flaps will be great, so that only a tithe of the current will reach the face which is carried into each working face. It might be argued that considerable leakage takes place under the longwall system from the air-current traversing the edge of the waste, but the answer to this argument is that the road along the face in longwall working usually presents the line of least resistance, and the most direct route to the return air-way. On the other hand, however, if certain parts of the seam are liable to greater efflux of firedamp, or in the event of an underground fire, it is easier to separate such portions from the rest of the workings, where the mine is laid out on the bord-and-pillar system. But, as the author has stated elsewhere, "it is seldom that a single fact determines the manner of working, but rather a series of general conditions point to the right conclusion."[1]

The Ventilation of Panel Workings.—Before leaving the subject of the disposition of the air-current in mine workings, there are a few problems, relating more particularly perhaps to the case of the ventilation of panel workings, which may be considered. These have reference to the manner of directing the air-current where (*a*) the coal gives off much gas; (*b*) where the barrier between the panels is being worked off.

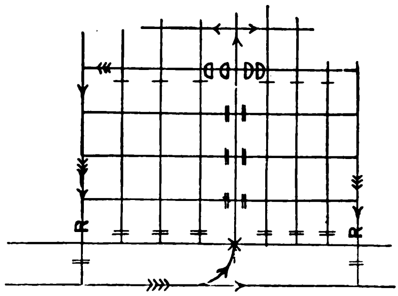


FIG. 102.—Plan of Ventilation of a Panel (Non-gassy). (From Bulman and Redmayne's *Colliery Working and Management.*)

(*a*) When a panel is being "worked in the whole," and the workings are not subject to much gas, the air-current may be directed, as shown in Fig. 102, but where the coal gives off gas to an appreciable extent, that is, in those cases where it is frequently detected by a visible cap on the flame of a safety lamp, and places have in consequence to be stopped from

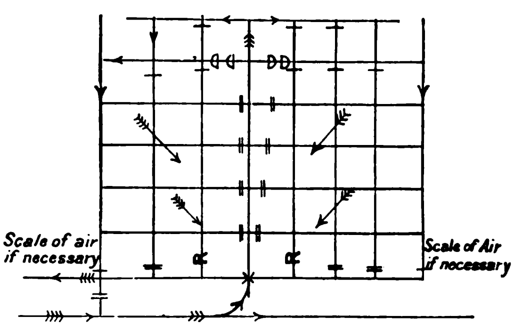


FIG. 103.—Plan of Ventilation of a Panel (Gassy). (From Bulman and Redmayne's *Colliery Working and Management.*)

[1] *Colliery Working and Management*, by H. F. Bulman and R. A. S. Redmayne, 2nd ed., p. 139.

work until the gas is removed, the mode of ventilation shown in Fig. 103 is usually resorted to, the position of the regulator being such as to necessitate the air passing through all the old workings before reaching the same, and making its exit into the main return air-way. Where the mine or district of the mine is subject to "blowers" of gas, which may continue to give off large volumes of firedamp at a high pressure for a long period, the method known as "coursing the air" is sometimes adopted, viz. the air-

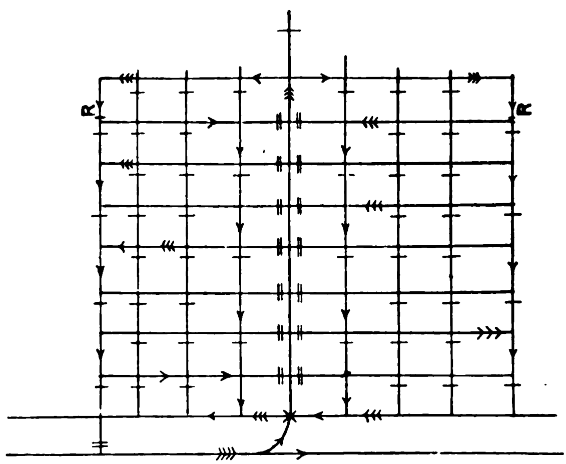


FIG. 104.—Plan of Ventilation of a Panel (Very Gassy).
(From Bulman and Redmayne's *Colliery Working and Management.*)

current is carried along the one end of a row of pillars and back along the other end, and so on along and back again the rows of pillars until it makes its exit into the main return air-way. Sometimes each row of pillars is "coursed," and sometimes a row is missed and the current brought back along the second row, and so on, the air-current being spoken of as being "coursed one and one," or "two and two," as the case may be. Where the seam being worked is exceptionally gassy, it may be necessary to course "one and one," but usually it suffices to course every three or four rows of pillars (see Figs. 104 and 105).

Fig. 106 shows the details of the ventilation of panels of bord-and-pillar work. Two of the panels are being worked back "in the broken." The diagram is taken from an actual case in the county of Durham. Concerning this plan the author has written elsewhere: "When working in the whole, the air was conducted along the face by means of wood brattice, which was fixed by the deputies to the props. Sometimes a row of props had to be set specially for the brattice. It ought to be kept within a few yards of the face, and the joints, in fiery pits, should be well pointed with lime, or 'cleaded' with strips of wood, as it is astonishing what a quantity of air may be lost and never reach the face when this is not carefully done. A place fouled with gas may sometimes be cleared by merely pointing the brattice and making it air-tight. Another matter requiring attention is the canvas doors at the place ends, which force the air up the brattice."[1] When the working back of the pillars has proceeded to some extent, it becomes necessary to work off the barrier dividing two of the panels. The question then arises: Which road will the return air of either district take? Will it pull from the first to the second district, or *vice versa?* "If no definite precautions are taken to keep it in one direction, it might, if there were heavy falls, or obstructions of any kind in the returns of No. 1 district, pull from this

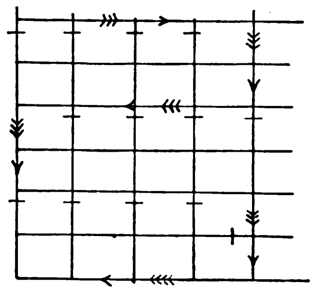


FIG. 105.—Plan of Ventilation of a Panel (Very Gassy: Coursing the Air Two-and-two). (From Bulman and Redmayne's *Colliery Working and Management.*)

[1] *Colliery Working and Management*, 2nd ed., pp. 170, 171.

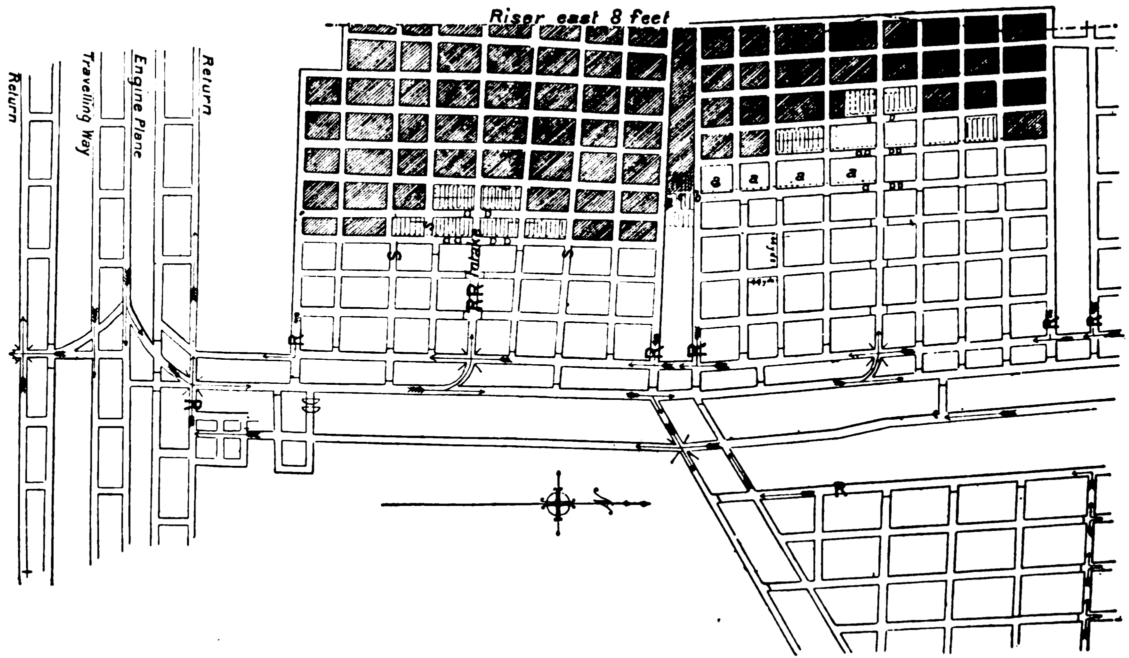


FIG. 106.—Plan of Panel Workings, Bord-and-Pillar System.

Reference.—Shaded part, Goaf; Air-crossings shown thus, ×; Regulators, R; Doors, D; Stoppings—Air Scales, S; Arrows denote direction of air-currents.

(From Bulman and Redmayne's *Colliery Working and Management.*)

district to the No. 2 district, carrying the gas with it from the goaf edge, if there were a low barometer, and so foul all of the workings of No. 2 district which it passed."[1] In order to avoid this the following steps may be taken. Two regulator doors, R, R, should be placed in the intake of the No. 1 district, the area of the aperture in each of which may, for the sake of explanation, be supposed to be 7 square feet, whilst the area of the aperture in the regulator in the north side return was, say, 10 square feet, so as not to put much of a check upon the efflux of the return air. The regulator doors on the intake of No. 1 district act as a check on the air-current passing through the apertures at a high velocity and against a corresponding resistance, but in the north side return there is say less resistance and an easy road for the air. Now, as there is no obstruction offered to the intake air No. 2 district, a more considerable volume of air will pass into it than into No. 1 district, and the No. 1 district regulator will be capable of passing a much greater volume of air than is supplied to it from No. 1 alone, a portion of the air from No. 2 will therefore pass through the barrier and down the north side return of No. 1, through the regulator there, and into the main return, and can always be depended upon as keeping to this road. This arrangement is known as "regulating the intake," the position of the intake double regulator being shown in Fig. 107.

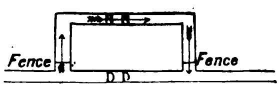


FIG. 107.—Mode of Regulating Intake Air. (From Bulman and Redmayne's *Colliery Working and Management.*)

VENTILATION OF METALLIFEROUS MINES

Ventilation of the Comstock Silver and Lead Mines, Nevada.—At the Comstock Mines in Nevada

[1] *Colliery Working and Management*, 2nd ed., pp. 173, 174.

there are six downcasts and three upcasts, and another shaft is divided into downcast and upcast. Some idea of the extent of the system is shown by the fact that the total volume of upcast air amounts to 235,835 cubic feet per minute, whilst the volume of the downcast air equals 216,687 cubic feet per minute. Leakage, rise in temperature, addition of water vapour and mine gas, variation in velocity at time of measurement, accounting for the discrepancy between the respective volumes of intake and upcast air.

Thirty-three years ago the minimum amount of upcast air at this mine was estimated at 288,630 cubic feet per minute, of an average temperature of 88·05° F. at the top of the upcast shaft, the outside temperature being 73° F.[1] The great effectiveness of the ventilation is most probably attributable to the direction of the underground air-currents, due to the liberal use of small fans placed underground (these have been found to be most effective), restricted workings, and fewer upcast shafts.

The Cost of the Ventilating System.[2]—The cost of this system has been estimated as follows:—

Supplementary ventilating plant—

Motors	£1,250
Fans	1,250
Air-pipes (15-inch pipes at 3s. 2½d. per foot and 11-inch and 20-inch proportionately) . .	2,083
	£4,583

N.B.—Cost of fan and pipe in Overman and Caledonia Mines was not included, nor cost of electrical conduits.

[1] *The Comstock Lode: Its Formation and History*, by Dr. J. A. Church, p. 18 (1879).

[2] "Ventilating System at the Comstock Mines, Nevada," by Prof. G. J. Young. *Bulletin of the American Inst. M.E.*, No. 35, p. 994.

Depreciation, maintenance, and operating expenses—

Depreciation—Maintenance assumed as 10 per cent. of plant cost or £458 per month	£38	3	4
Operating Cost—			
Power—165 horse-power at £1, 0s. 10d. per horse-power per month	171	17	0
1 electrician at 16s. 8d. per day	25	0	0
Materials	10	7	0
Total expenses	£245	7	4

Or a cost of 0·06864d. per 10,000 cubic feet of air circulated per minute, not including cost of repairs to upcast shafts.

Mr. Young remarks (p. 980 of the *Bulletin*): "Since the cooling of the upcast air directly affects the efficiency of the upcast shaft, the factors contributing to this are of some importance. I reached the following conclusions:—

"1. The greater the difference between the temperature of the upcast air and the rock temperatures, the greater the cooling effect of the shaft.

"2. The higher the temperature of the air entering the bottom of the upcast, the greater the cooling effect of the shaft.

"3. The lower the velocity, and consequently the smaller the volume of air in the upcast shaft, with given area and rubbing surface, the greater the cooling effect of the shaft.

"4. The greater the proportion of rubbing surface to shaft area, the greater the cooling effect.

"5. When incast and upcast air are conducted through the same shaft, the cooling effect of the incast upon the upcast air is very marked.

"Regarding the effect of humidity upon the cooling effect of the shaft, no definite conclusion was recorded."

The Ventilation of the East Rand Proprietary Mines, Transvaal.—A most interesting and valuable paper on the scheme of ventilation recently carried out at the East Rand Mines, near Johannesburg, appeared in the *Journal of the Chemical, Metallurgical, and Mining Society of South Africa*, contributed by M. S. Penlerick, the manager of those mines. The greatest credit is due to this enterprising mining engineer for carrying into effect what appears to be one of the most complete schemes of ventilation in operation in a metalliferous mine, and I am indebted to him and to the Council of the Society for the permission to reproduce the diagram illustrating the scheme, and to give particulars relating thereto.

The object of this great scheme of ventilation was largely to combat the three main factors which on the Witwatersrand combine to produce disease amongst the miners, viz. dust, infective processes, and fumes from explosives. The standard of ventilation recommended by the Transvaal Mining Regulations is the following:—

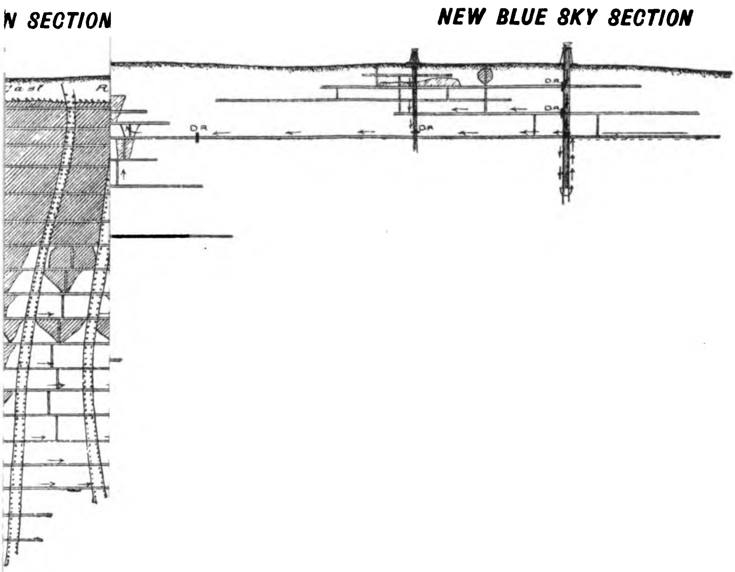
"1. In every metalliferous mine sufficient air shall be provided, and such other arrangements made, that in any sample of air taken under normal working conditions from any part of the mine, not less than one hour after blasting—

"(*a*) The proportion of carbon dioxide shall not, as regards any mine within the Witwatersrand area, exceed 20 volumes per 10,000 of air = 0·2 per cent.

"(*b*) The proportion of carbon monoxide shall not exceed 1 volume per 10,000 of air = 0·01 per cent.

"No practically determinable amount of the oxide of nitrogen (NO and NO_2) shall be present."

This, it will be observed, is a high and satisfactory



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R indicates Door with Regulator
,, Stopping
,, Subsidiary Ventilation Plan

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standard to assume, and it was on this assumption that the scheme under consideration was carried out.

The task, therefore, that the management had to meet was that inherent to all mine ventilating schemes, viz. (1) The provision of a sufficient quantity of fresh air; and (2) the efficient distribution of the same. In effecting these objects they decided to make all the main shafts downcast, and use the two prospecting shafts, which were connected to the old stopes, as upcasts, equipping them accordingly. The fans arranged at the top of the upcast shafts gave the following results:—

TABLE XXI.—*Results obtained with Ventilating Fans at the East Rand Mines.*

Shaft.	Fan.	Diameter.	Width.	Output Cubic Feet per Minute.	Water-gauge.
		Ft. In.	Ft. In.		Inches.
Angelo .	"Sirocco" double inlet .	10 6	8 6½	300,000	3
Cason .	"Barclay's" double inlet. Drum pattern	16 0	8 0	350,000	4

The accompanying ventilation chart shows the configuration of the mines in longitudinal section, and the direction of the air-currents.

When ventilating metalliferous mines one of the difficulties is the ventilation of the "dead ends" of drives; often the exhaust of compressed air drills is used for this purpose where such drills are in use, but such ventilation is intermittent and unsatisfactory. In this connection, therefore, it is interesting to note what Mr. Penlerick states regarding the ventilation of development drives: "Where a drive face is being carried far beyond the last connection with an adjacent level, it is necessary to provide for an intake and an outlet. This object is best obtained by using a galvanised iron pipe for the outlet; we prefer to use the pipe for the outlet, as the drive in which there is traffic is kept fresh throughout."

The effect of the scheme was remarkable. Before it was carried out the average analyses of the mine air gave—

$CO_2 = 0{\cdot}493$ per cent.
$CO = 0{\cdot}012$ „

But after the adoption of the scheme—

$CO_2 = 0{\cdot}127$ per cent.
$CO = 0{\cdot}005$ „

More recent samples taken from the air drift at the Angelo upcast "directly after blasting, when the vitiation of the air should be at its worst," gave the following results—

$CO_2 = 0{\cdot}182$ per cent.
$CO = 0{\cdot}001$ „

and the temperatures were reduced by 2° to 3° F.

The capital cost of the installation—fans, motors, erection of same, equipment of upcast shafts, bratticing, doors, pipes, &c.—was about £25,000.

Mr. Penlerick most forcibly sums up the result. He says: "The cost is insignificant compared to the valuable results obtained, and under the circumstances some system of artificial ventilation should be adopted in practically every mine on the Rand.

"Another important factor in securing the most satisfactory conditions of the mine air is the practice of confining all drilling and blasting operations to the day-shift. We are aware that many companies to-day are not in a position, for financial and other reasons, to adopt this practice, but where the adoption of it is possible, we cannot too strongly recommend it."

CHAPTER IX

COLLIERY EXPLOSIONS AND GOB FIRES

Colliery Explosions.—It is this form of colliery disaster that strikes the imagination most forcibly, so appalling in its suddenness, terrific in its force, and so disastrous and far reaching in its consequences. Yet the number of lives annually lost through falls of roof and sides exceeds the toll due to explosions of gas or coal dust. The figures for the last twenty-five years (1886–1910) show that 3135 lives were lost by explosions of firedamp and coal dust, 12,235 by falls of roofs and sides; whereas the miscellaneous accidents below ground (including shaft accidents) were 9052, or a total loss of 24,422 through accidents below ground.

It will be seen, therefore, how comparatively small is the loss due to explosions (about 13 per cent. of total underground fatalities), but these disasters constitute the worst form of accidents for all that. Think of the number of widows, and others dependent on the wage-earners, left destitute, and consider the economic effect of a pit laid idle, it may be, for months, owing to the sweeping devastation caused by the explosive force.

Three Eras in the History of Colliery Explosions.—The history of colliery explosions may be divided under three heads:—

(*a*) The era of small colliery explosions, the first account of a colliery explosion being found in a paper contributed by Roger Mostyn, in 1677, to the Royal

Society, and firedamp was first heard of in opening out a new colliery at Mostyn.

(*b*) The era of big explosions. Repeated disasters led to the formation at Sunderland, in the year 1813, of a Society for the Prevention of Accidents in Coal-mines.

(*c*) Coal dust. After Seaham explosion (county Durham) in 1880 (8th September), the coal-dust theory came into prominence—a theory which will be for ever associated with the names of Galloway[1] and the brothers Atkinson, for the former was the first to bring the theory to the notice of British mining engineers, and the latter, in their well-known and admirable work on colliery explosions,[2] demonstrated the truth of the theory from the facts revealed by actual colliery explosions.

The Chemistry and Physics of an Explosion of Gas.—An explosion of ordinary marsh gas may be chemically expressed thus—

$$CH_4 + 4\,O + 16\,N = CO_2 + 2\,H_2O + 16\,N.$$

It was in July 1831 that M. Berthelot and MM. Mallard and Le Chatelier wrote papers announcing the enormous velocity of explosions in gaseous mixtures.

Now a mixture of 9½ volumes of dry air to 1 volume of firedamp contains exactly the proper proportion of oxygen necessary to effect complete combustion of the firedamp, and ought, therefore, to be the most explosive mixture that can exist,—one through which the flame

[1] Professor Galloway had "made a special study of the subject of explosions, both in wet mines in Scotland, and in dry and dusty mines in South Wales, while acting in the capacity of an Assistant Inspector of Mines during the years 1873, 1874, and 1875, and during the course of these studies" he had observed "two remarkable facts, namely: (1) That a firedamp explosion in a wet mine never by any chance assumed the character and proportions of a great explosion; (2) that all great explosions took place in dry and dusty mines." Vide *Lectures on Mining—Colliery Explosions*, by W. Galloway, p. 16 *et seq.*

[2] *Investigations of Colliery Explosions*, by J. B. Atkinson and W. N. Atkinson.

would be expected to travel with the greatest rapidity. But that part of the mixture through which the flame has passed expands under the influence of the heat of combustion and drives the still unconsumed part in front of it; the velocity so acquired by the latter must be added to the normal velocity of the propagation of the flame.

The pressure due to the combustion of the above mixture, when ignited in a closed space, is equal to 102·6 lbs. per square inch, and the calculated temperature of combustion is 3902° F. The resulting damage from the explosion of say a few hundred cubic feet of such a mixture in a mine can therefore be well imagined.

Berthelot, associated with Vielle,[1] measured the rate of the explosive wave, and Le Chatelier and Mallard studied the preliminary phenomena which precede the formation of the waves.

Three important phenomena were demonstrated by the latter, by means of photography, on a revolving cylinder, viz.—

(1) That the flame (in a tube) travelled a certain distance at a uniform velocity.

(2) After a certain point, vibrations were set up which altered the character of the flame, and that these became more intense, swinging backwards and forwards.

(3) The flame either went out altogether, or the rest of the gas detonated with extreme velocity.

In these experiments either nitric oxide or carbon bisulphide were used, but similar results would be obtained with ordinary gas, except that firedamp would not detonate to the same extent as the above-mentioned gases. M. Berthelot made the important discovery that the rate

[1] The result of their investigations were made public in papers in the *Comptes Rendus* in 1881.

of explosion increased from the point of origin until it reached a maximum which remained constant, however long the column of gases might be.

Now these results may appear to be purely theoretical and of little practical use, but such is not the case, for, by applying them to actual colliery explosions, we find they are there borne out on a large scale, and frequently enable us to determine the originating point of the explosion, a matter of the utmost importance. For it has been very generally observed that—

(1) Little damage is done near the point of origin of the explosive flame, and for say 50 to 80 yards therefrom.

(2) Then great damage is evidenced.

(3) And after the cessation of the advancing explosion there is evidence of a "back-lash" or rebounding force.

Increased Loss of Life per Explosion.—Let us now consider a statistical point of importance. We have seen what is the loss of life due to explosions generally, but what does the loss of life *per fatal explosion* amount to, and what lessons can be drawn therefrom ?

Taking six decennial periods—

During the 10 years	ending	1860	the loss	of life	per explosion was	2·91
,,	,,	1870	,,	,,	,,	4·01
,,	,,	1880	,,	,,	,,	6·33
,,	,,	1890	,,	,,	,,	6·75
,,	,,	1900	,,	,,	,,	5·42
,,	,,	1910	,,	,,	,,	7·45

so that the average loss of life per explosion may be said to have steadily increased with the exception of the decennial period ending with 1900.

What is to be learned from this ?

(1) Coal dust is an element of danger in collieries as an augmenting influence in explosions, extending the area of devastation.

(2) That the coal-dust danger has increased since mines became deeper, drier, and warmer. Air takes up moisture in the main intakes, seeing it enters the mine in a comparatively dry condition, and by the time it reaches the face it has become laden with moisture,[1] and during its passage it has been raised to a temperature corresponding to that of the mine (when air is warmed its capacity for absorbing moisture is greater, and *vice versa*), and it absorbs moisture from the new-cut coal, so that its capacity for taking up further moisture is stopped; that in the returns, therefore, will not take up any more.

(3) That in 1887, and again in 1896, certain rules relating to shot-firing, where coal dust is present, were introduced into the Coal Mines Regulation Act, and in 1900 the first of a series of Orders in Council, cited as the explosives in Coal Mines Order, was issued by the Home Secretary, specifying what explosives shall be used in dry and dusty mines, which list is known as the permitted list, and is subject to revision from time to time, and that this *may*, perhaps, account for the abatement marked during the ten years ending 1900.

Conditions Necessary for the Production of a Coal-dust Explosion.—What are the conditions necessary to produce a coal-dust explosion?

Mr. W. N. Atkinson has stated these to be the existence of a cloud of dust in the air, and the production of flame at the same time.

The Royal Commission on Accidents in Mines (reported 1886) laid down three conditions as necessary:—

(*a*) Existence of coal dust in abundance.

(*b*) The dust must be very inflammable.

(*c*) The means of ignition must be a blown-out shot.

[1] This is *generally* true, but a case recently presented itself in which the air at the face of a coal mine was less humid than that of the outer atmosphere.

And the Royal Commission on Explosions from Coal Dust in Mines (Report issued in 1894) concluded that—

(*a*) The danger of explosion in a mine in which gas exists, even in very small quantities, is greatly increased by the presence of coal dust.

(*b*) A gas explosion in a fiery mine may be intensified, and carried on indefinitely, by coal dust raised by the explosion itself.

(*c*) Coal dust alone, without the presence of gas at all, may cause a dangerous explosion if ignited by a blown-out shot or other violent inflammation. To produce such a result, however, the conditions must be exceptional, and are only likely to occur on rare occasions.

(*d*) Different dusts are inflammable, and consequently dangerous, in varying degrees; but it cannot be said with absolute certainty that any dust is entirely free from risk.

(*e*) There appears no probability that a dangerous explosion of coal dust alone could ever be produced in a mine by a naked light or ordinary flame.

Professor Galloway, MM. Le Chatelier and Mallard, and the Prussian Commission, agreed that the danger of dusts, as regards their capability of producing an explosion in pure air, was in proportion to the amount of volatile hydrocarbons they contain, though the Prussian Commission somewhat qualified this statement by saying that, when the dust contains more than 27 per cent. of volatile hydrocarbons, it was less dangerous.

"Top" (that on the timbers supporting the roof and sides, and adhering to the sides of the roadway, in crevices, &c.) and "bottom" (on the floor) dust differ considerably in composition, the top dust constituting the most dangerous element; the bottom dust being diluted and rendered more or less harmless by the admixture of shale, horse-droppings, and other impurities.

Numerous experiments have shown that, though 2 per cent. of firedamp and 98 per cent. of air is by itself a harmless mixture, if with this is mixed a dense cloud of coal dust, and a pistol fired into it, a violent explosion will in all probability result.

The Genesis of a Coal-dust Explosion.—The present writer may be in error, but he is under the impression that the mining world is indebted to Mr. W. C. Blackett for first describing in detail what is rapidly coming to be regarded as the most probable theory of the genesis of what is generally called a coal-dust *explosion* in a coal-mine.

The origin of the explosion may be, say, a shot which has liberated and ignited gas or raised and ignited a cloud of coal dust, or it may be a light which has caused the ignition of the gas, the detonation of which has raised the cloud of dust, and the explosion once started is carried on by dust to the other parts of the workings. On the initiation of the explosion there is at the originating point a sudden expansion followed by a contraction of the air, so causing a wave of pressure and rarefaction to traverse the mine, which travels at a much greater speed than the following fire. This advance force, or, as Mr. Blackett terms it, "pioneering phenomenon," sweeps up the dust into the air, which, not having had time to resettle before the advent of the flame, is ignited, and so the explosion is continued without cessation, but with varying intensity, according to the state of the mine in respect of dust.

"The increasing commotion is stirring up greater clouds, and the increasing pressure is serving to intimately mix therewith more abundant oxygen, and pursuing this increasingly circulated explosive mixture is the flame. Air is, however, so elastic that this accelerated movement will, in the first instance, have been quite gentle, and in its flow it would naturally follow those paths most easy for it,—those, indeed, that it would have followed had it been guided by the ordinary laws of ventilation. Outbye it takes the nearest open way to the surface, and inbye to the expanding area of workings. This gentle flow prepares the way for the rush: the rush stirs up the dust and pioneers the path for the incoming explosion. There is no turning to the right or left, and no going through doors or stoppings into the returns; but as the explosion passes, down go these obstacles, beyond which, however, the road is not prepared, and therefore the explosion does not extend therein. The shaft is soon reached, the pressure relieved, and the explosion ceases, or perhaps up the shaft the velocity and counterweight are so great that the pressure can be actually higher than the resistance of a road beyond. In this case the pioneering phenomena are continued, and the explosion progresses.

"In the direction of the face the explosion has proceeded until the greater number of open galleries has reduced the speed of the rushing air, so that it will no longer produce a thick cloud of dust, and it dies away. It has perhaps, indeed, partly provided its own safety valve, for the stoppings and doors into the returns have been blown down by the passing explosion.

"All the available oxygen has been consumed; indeed, there has not always been sufficient for complete combustion, and instead of carbon dioxide alone, the deadly carbon monoxide is also found.

"If a damp part of the road, sufficiently long and free from dust, is encountered, of course the pioneering work is incomplete, and the explosion ceases. Similarly, if the area of the road were sufficiently increased it would probably cease."[1]

In these words it seems to the author that Mr. Blackett visualises with singular accuracy the genesis of a colliery explosion, and explains the otherwise inexplicable effects observable after an extensive colliery explosion.

Present State of our Knowledge respecting the Influence of Coal Dust in Colliery Explosions.—Since the experiments carried out by the Prussian and French Commissions, and those by the North of England Institute of Mining Engineers at Hebburn-on-Tyne, experimental work in respect of the influence of coal dust in colliery explosions has of late years been conducted, and is still being conducted, on a large scale at Altofts in Yorkshire, Gelsenkirchen in Austria, Frameries in Belgium, and Liévin in France, in long experimental galleries.

The experiments, so far as these stations are concerned, seem to show that the most important factors in the dust problem are—(1) The proportion of volatile hydrocarbons contained in the dust; (2) the purity of the dust; (3) fineness; (4) quantity; (5) dimensions of the road or gallery in which the ignition takes place; (6) the chemical and physical state of the air; (7) the humidity of the dust, and walls of the gallery; (8) the character of the initial inflammation.

So far, also, as investigations have proceeded it does not appear that the *quantity* of coal dust present is of much importance. In the recent French experiments an explosion was obtained when coal dust was present only to the extent of 56 grammes (about 2 oz.) per square metre (about $10\frac{3}{4}$ square feet).

[1] "The Combustion of Oxygen and Coal Dust in Mines," by W. C. Blackett, *Trans. Inst. M.E.*, vol. vii. pp. 54–63.

The higher the percentage of volatile constituents, up to a certain point, other things being equal, the greater the inflammation caused by the ignition of the dust. Also the finer the dust, and the lower the percentage of ash, the more dangerous it is. Humidity, whether of the air or of the dust, as might well be supposed, adversely affects the initial inflammability of the dust; and if inflamed, the duration and extent of the ignition or explosion. Temperature and barometric pressure do not appear to have much, if any, effect. It would appear, so far as experimental research has yet proceeded, that suspension of the dust in the air prior to or at the moment of the emission of flame is necessary to cause an explosion. There must be some disturbing influence to bring it into suspension. Such influences might be, a shot, an ignition of firedamp, a fall of roof or side, a set of tubs getting off the rails, or loaded tubs travelling outbye against a strong air-current, so causing the dust to be blown off the tubs.

In effect, not less terrible than the explosion, is the after-damp, small quantities of which render the air so fatal to those who may have survived the mechanical effects of the explosion.

Remedial Measures.—Many remedies have been proposed to render the galleries of a coal-mine safe from coal-dust explosions, the chief of which are :—

(1) Removal of the dust by shovelling.

(2) Watering by sprays, &c.

(3) The provision of "stone dust" or of "wet" zones on the main roads of the mine.

(4) Use of salt, calcium-chloride, or other inexpensive deliquescent compounds.

(5) Intermixture of stone or shale dust by sprinkling the roads with the same.

(6) Prohibition of dry tamping in shot firing.

(7) Prohibition of coal-dust tamping in any form.

The last precaution has been made compulsory in the United Kingdom, and the others are having the close consideration of mining men.

Gob Fires.—Akin to colliery explosions are gob fires, the origin of which has been the subject of much discussion. Some districts, notably Northern and Welsh mines, are almost free from this class of underground fire, and a comparative microscopic examination of the coals from these and from those districts prone to gob fires points to the influence of texture on the property of spontaneous combustion. Rapid absorption of oxygen by finely divided particles of coal causes evolution of heat, and as the heat increases, the absorption of oxygen increases until the coal bursts into flame. Iron pyrites may be to some extent the exciting cause, but it is not the supreme cause.

Remedial Measures.—Up to the point when what is called "green" smoke (commencement of incandescence) appears, it is well to keep as much air as possible on the site of the fire, but when actual fire is observed the air must be cut off entirely. To prevent spontaneous combustion in the "thick" coal, pillars must be left as large as possible so as to resist crush, which is accompanied by friction, and therefore heat, and which also makes much small coal. When working thinner seams, all the coal should be extracted, the faces advanced as quickly as possible, and the goaves packed tightly with stone, with as little admixture of coal as possible.

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END OF VOL. IV.

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