Planning the Ventilation of New or Reorganised Collieries

(SUPERSEDING INFORMATION BULLETINS MP(51)14 & 53/93)

INTRODUCTION

The volumes of air which will be required for the efficient and safe ventilation of a new or re-organised colliery at various stages in its development, and the fan pressures necessary to circulate these quantities, must be estimated as closely as possible if the fan to be installed is to be suitable for the duties required of it and if the mine is to be ventilated adequately but economically.

This bulletin contains the data on airway resistances and air leakage quantities necessary for calculating the required capacities of the fans. The data given can also be used for planning the general layout of the underground airways to ensure that the ventilation system will be well balanced and efficient.

The appendix comprises a series of thirteen charts which enable the pressure losses in shafts, airways and faces of various sizes and having various types of lining to be estimated quickly without calculation.

VENTILATION PLANNING

In order to plan the ventilation of a new or reorganised mine, the first step to be taken is the preparation of plans showing the projected workings of the mine at each stage of its development. It is nearly always impossible to forecast precisely the manner in which a mine will be developed and worked, and the future positions of the workings, especially at the later stages, must necessarily be hypothetical. Nevertheless, such plans must be made, and they should be based on the best information available. They should show the estimated positions of all the faces and all underground roadways, drifts, headings, motor houses, pump houses, locomotive garages and water-lodges which will need to be ventilated. A scale of 1:5000 will usually be found suitable for this purpose and it will be necessary to prepare a set of plans for each seam to be worked.

The stages in the development of the colliery for which the plans are made should, as far as possible, be chosen so that they cover periods following major changes in the ventilation system, as for example, the connection of airways at a new horizon. If there are no natural stages in the development of the mine, suitable stages might be as follows:-

Stage (1) When production of coal is started.
" (2) 5 years after production of coal has started.
" (3) 15 " " " " " " "
" (4) 25 " " " " " " "

The quantity of air required for the ventilation of the colliery at each stage will usually be determined by the rate at which firedamp will be evolved and the maximum methane percentage which will be permissible on the faces or in the return airways, except that the quantity of air to be provided for in a mine producing little gas should not fall below a certain minimum and in the case of a hot deep mine it may be necessary to circulate additional air for cooling purposes.

Calculations of the amount of gas in each district which it will be necessary to dilute with air should be made by multiplying the output of coal to be obtained from the district by the amount of gas it is estimated will be discharged on to the faces or into the return airways for each ton of coal worked.

Having decided on the maximum permissible methane percentage in the air and on the gas emission figure for each face, the quantity of air required on each face should be calculated and marked on the plans. The quantity of air passing in each airway should then be shown, starting inbye and working towards the surface, allowances being made for leakages between the intake and return airways and for the air required to ventilate motorhouses, etc., and districts which are standing or being drawn off.

Calculations should then be made of the ventilating pressure drops across each face and in the airways and shafts. For this purpose the airways should be divided into lengths according to their planned cross-sectional area and the quantity of air they are to carry. The calculated pressure drops should be marked on the plans and also recorded in tabular form. The pressure drops can best be obtained directly from the charts given in the appendix.

At this stage, calculations should be made of the resulting air velocities (a) on each face, (b) in the main trunk airways, with special attention to the pit-bottom area, and (c) in the shafts, to ensure that there is none which is undesirably high. If high velocities are found, it may be necessary to allow for shorter faces and larger or duplicated airways.

Sharp elbow bends in main airways cause high pressure losses: the main airways should therefore be as straight as possible. If a bend is unavoidable, the airway should, if possible, be curved so that the radius of the centre-line of the bend is not less than twice the width of the road. Main airways at junctions should also be curved in a similar manner, whenever practicable.

The whole ventilation system should then be reviewed to ascertain whether it would be economical to reduce any high pressure losses in the main airways (a) by allowing for airways or larger cross-sectional area, or (b) allowing for airways having a smooth lining, or (c) for duplication of the airways. For example, it may prove to be an advantage to make certain roadways or sink staple shafts and to use them for ventilation purposes before they are needed for transport. The system should also be examined to see whether it is well-balanced and, if not, to see whether it would pay to reduce the pressures required to ventilate those districts requiring the highest water gauges and so to avoid, or minimise, the use of regulators. Regulators should be avoided as far as possible as they decrease the efficiency of the system and increase the power required to ventilate the mine. If, in exceptional circumstances, it is found to be uneconomical to avoid having one or two districts which absorb a considerably higher water gauge than the remainder, the use of underground booster fans should be considered.

At some collieries where the seams are liable to spontaneous combustion, it has been found by experience that the ventilating pressures across the faces or between intake and return roads in the seam must not exceed certain values. If the seams at the colliery being planned are expected to be liable to spontaneous combustion, the ventilation system should be examined to see that no pressure difference exists which is higher than that which experience has shown should not be exceeded.

The quantity of air to be handled by the fan at any time (measured in cu. ft./min.) will be the estimated quantity of air at the bottom of the upcast shaft plus an allowance for its expansion while passing up the shaft, together with the air used for ventilating any mid-shaft insets and an allowance for air leakage at the surface through the upcast shaft casing, air-lock doors, fan-drift walls, etc.

The total pressure drop from the top of the downcast shaft inbye to the working face and back to the fan should be determined in respect of each ventilation district, and the greatest of these pressures will be the total ventilation pressure required for the mine.

There is usually a tendency to over-estimate the air quantities required and to under-estimate the pressure losses; this must be guarded against. If it is possible to measure the actual pressure drops in existing shafts and roadways or across existing faces at the colliery which is being re-organised, such measurements may be made and the data so obtained used in place of the data given in this bulletin.

NATURAL VENTILATION PRESSURE

The 'useful' pressure which a mine fan develops is equal to the total ventilating pressure required to circulate the air through the mine less the natural ventilation pressure generated in the shafts and workings.

The value of the N.V.P. at a colliery is subject to wide seasonal and diurnal variations, but Table I* gives approximate average values which have been calculated for a surface temperature of 50 deg. F ; the appropriate value should be deducted from the estimated total ventilating pressure in order to obtain the fan static pressure (sometimes termed the fan useful pressure).

CHOICE OF FAN SIZE

The correct size of fan to choose will be determined by the maximum quantity of air it is likely to be required to handle and the equivalent orifice (or resistance) of the mine.

It should nearly always be possible to make a fairly close estimate of the maximum quantity of ventilating air which is likely to be required, but in order to provide a margin of capacity to cover unexpected conditions, the fan chosen should, if possible, be so designed and constructed that it could be safely run at a speed 15 per cent above its speed at its highest estimated duty, if ever this was found to be desirable. It is not recommended that a similar margin should be allowed for the motor and driving gear.

*For convenience of use the tables have been grouped together at the end of the bulletin with a summary of other data.

If a rough estimate has to be made of the quantity of ventilating air required at a mine, or if an estimate has to be made for a mine at which the future underground conditions are largely unknown, the figures in Table II can be used. These figures are based on maximum and minimum face air velocities of 500 ft/min and 200 ft/min respectively, on a daily face advance of 4 ft 6 in and on the assumption that 50 per cent of the air in the fan drift reaches the working coal faces. For the gassiest seams the air quantities quoted would dilute the methane to 0.7 per cent at the return end of a face 100 yd long. For moderately gassy seams the methane would be diluted to 0.5 per cent at the return end of a face 100 yd long. The air quantity recommended for a seam in which the gas emission is 200 cu.ft/ton would dilute the methane to 0.4 per cent at the return end of a face 150 yd long.

For deep mines having workings at a depth of more than about 900 yd below the surface, at least 55 cu. ft/min per ton should be provided for on the faces, equivalent to about 3000 cu. ft/min per foot height at the face.

Care must be taken to obtain as close an estimate as possible of the equivalent orifice of the mine at the various stages, because the efficiency of a fixed-blade fan is determined by the resistance against which it operates and not by the quantity of air it passes.

It is most important that the cross-sectional areas of the airways on which the mine resistance calculations are based should not be overestimated, since a slight overestimate of the areas may result in a serious underestimate of the fan pressures required. Realistic allowances, based on a knowledge of local conditions, must be made for crush on the airways.

In most cases the new fan will have to work on a wide range of equivalent orifices and it will not be possible for it to maintain its highest efficiency over the whole range. The size of fan chosen should usually be such that the fan is working at its highest efficiency during the period for which the figure for H P x Years is greatest. It is recommended that the final duty of the fan should normally be its estimated duty 25 years after its installation. It will not usually be possible to forecast with any degree of accuracy the air quantities and ventilating pressures which will be required beyond this point.

GAS EMISSION RATES

It is most important that gas emission rates should be estimated as closely as possible, but unfortunately this is often very difficult owing to lack of information and to the wide variations which occur in the rates of emission for any seam within a small area.

Samples to determine the quantity of gas discharged into the ventilating air (a) in each return gate 10 yd from the face and (b) at the outbye end of each return gate, should be collected from all collieries within a fairly wide radius (say 5 miles) of the colliery being planned, and the results obtained should be marked on plans showing the outline workings in each seam. Good judgment is required when choosing the values to be used. Values in respect of faces from which the coal output is increasing or decreasing rapidly, or in respect of very short faces, or of faces advancing slowly, should be discarded, as they cannot be applied to normal faces advancing rapidly and regularly. Care should also be taken that the data collected refer to periods at which the gas-emission rates are highest. It is often found that the rate of gas emission from a face is at its maximum at the end of the working week, while in other faces there are considerable variations each day, peak gas emission rates occurring during coal-cutting or waste-drawing operations, or when the barometer is falling rapidly. It is therefore important that a sufficient number of air samples should be taken in each return airway, over a period, to detect and measure the peak rate of gas emission.

The gas emission figure used for the purpose of estimating the future air requirements of a face should be calculated from the estimated peak rate of emission (this may occur over periods of only a few hours, or of a shift, or a day) and the average daily output of the face to which it refers. The calculated gas emission per ton of coal worked should then be applied to the average planned daily output from the future face.

It is common experience that the quantity of gas produced when any seam is worked is dependent to a considerable extent on the number and distance away of any seams which have been previously worked above or below it, and that a seam in a virgin area usually produces most gas.

There are indications that a seam may be partially drained of its gas by working another seam within about 120 yd below it, and it is possible that it may also be partially drained by working another seam within about 50 yd above it. The degree to which the seam is so drained depends on the nature and thickness of the intermediate strata. The plans showing the gas emission figures should therefore also show, in respect of each face for which the figures are given, the distance above or below of any seam previously worked.

There is some evidence that the amount of gas produced when a seam is worked tends to increase with depth, but it is impossible with our present knowledge to give any reliable indication of the rate of increase. If the workings of the colliery being planned are to be appreciably deeper than those for which actual gas emission rates are known, it would be wise to allow for an increased rate of gas emission due to depth; a reasonable allowance might be a 10 per cent increase in the gas emission rate for every 100 yd increase in depth.

A careful study of the information given on the gas emission plan should enable a table to be drawn up of the gas emission rates to be expected at the colliery being planned. The table should be divided to show the estimated gas emission rates of each seam when worked in virgin ground and when over or undermined by previous workings; these rates should be applied, as appropriate, to the faces shown on the development plans in order to ascertain the quantity of air which would be required for their ventilation.

MAXIMUM PERMISSIBLE METHANE PERCENTAGES

It is usual for the quantity of methane in the return gate of a working face to be greater at the outbye end than at the inbye end. It is therefore necessary to decide whether the quantity of air required for each district is to be calculated with reference to the maximum methane percentage in the air in the return airways or to the percentage on the working face. This will largely depend on whether or not it is proposed to run electric or diesel locomotives, or to install electrical apparatus such as conveyors, tub-tipplers, haulage gear, etc. in the return airways. If such apparatus is to be used, the quantity of ventilating air required will be that necessary to dilute the amount of gas expected at the outbye end of each return gate to the standard fixed. If no electrical apparatus or diesel locomotives are to be used in the return airways, the amount of gas expected in the return gate at a point 10 yd from the face should be used to determine the quantity of air required. It can be assumed that any large emission of gas into the return gates can nearly always be prevented by firedamp drainage.

A reasonable methane percentage figure to take for the purpose of the calculations, both for return airways containing electrical apparatus and for the return end of the faces where no electrical apparatus or locomotives are to be used, is 0.5 per cent. This figure allows a small margin below the methane content of 0.6 per cent at which it becomes necessary to take weekly samples of the air for analysis under the Coal Mines (Ventilation) General Regulations, 1947.

If the seam to be worked is very gassy, it may not be possible to reduce the methane content of the air to 0.5 per cent without either having undesirably high air velocities on the faces or having the faces uneconomically short. This may necessitate the installation of compressed air operated machinery and haulages on the faces and in the return airways, in which case it would be reasonable to base the calculations on a methane content of 0.8 per cent at the return end of each face.

AIR LEAKAGE QUANTITIES

Across Newly Formed Goaf

The quantity of air passing along each gate road should be assumed to exceed the quantity passing along the face by the amount of leakage given in Table III, later.

If solid stowing is practised no allowance need be made for leakage.

Separation Doors

Doors near the pit bottom will be subjected to a higher ventilation pressure than those inbye, but they will usually be in more settled ground and can be maintained in a more air-tight condition. The amount of air leaking through each set of doors can be assumed to be 3,000 cu. ft./min, irrespective of their position. Care should be taken to allow for separation doors in all the connections which are likely to be made between intake and return airways.

Air Crossings (not 'explosion-proof')

The following values may be assumed - 3,000 cu. ft./min plus 3,000 cu. ft./min, for each set of doors between the intake and return airways.

Surface Leakage

The amount of air leakage through the casing at the top of the upcast shaft, the air-lock doors and fan-drift walls will depend on the fan-drift water-gauge pressure and on the size, method of construction and condition of the pit-top structures, on the number and type of entrances provided and on whether the shaft is used frequently for winding.

For a new pit top, well constructed in concrete and laid out for the regular winding of coal or dirt, the values given in Table IV should be assumed for the surface air leakage. If the shaft is not to be used regularly for winding, the values given in Table IV may be reduced by 20 per cent. In the case of existing shaft tops, the actual leakage should, if possible, be measured. This can often be done most conveniently by taking a series of simultaneous air samples in the fan outlet and at the top of the upcast shaft and analysing them for their methane content. The air leakage at the shaft top is given by the formula:-

$$\text{Leakage in cu. ft./min} = \frac{Q\,(U.C. - F.O.)}{U.C.}$$

Where Q = quantity of air in cu. ft. per min. passing in the fan-drift
U.C. = average methane percentage in the air in the upcast shaft.
F.O. = " " " " " " " " " fan outlet

Badly constructed pit tops or pit tops in poor condition may cause an air leakage of two to three times the figures given in Table IV.

AIR REQUIRED FOR VENTILATION OF HEADINGS, MOTOR HOUSES, LOCOMOTIVE SHEDS, ETC.

Hard Headings or Stone Drifts

It is usually necessary to provide 25 cu. ft. of air per sq. ft. of excavated area at the face of drifts. Allowing for leakage into or from the air-duct, an average quantity of air for each drift would be 8,000 cu. ft./min.

Small Drifts and Development Headings in Coal

Allow 5 000 cu. ft./min.

Motor, Haulage and Pump-houses

Allow 5 000 cu. ft./min.

Battery-charging Stations

Allow 10 000 cu. ft./min.

Diesel Locomotive Garages and Repair-shops

The ventilation required will depend on the largest number of locomotives it may be necessary to have running simultaneously in the garage and also on their size; but locomotives will not be running on full load in the shed. 150 cu. ft. of air per min. per total rated b.h.p. of the locomotives which may be running would be an adequate allowance.

Diesel Locomotive Haulage Roads

Each diesel locomotive road should pass a minimum of 150 cu. ft. of air per min per b.h.p. of the locomotives which may work in the ventilation circuit at any one time.

PRESSURE LOSSES

The pressure drop along a length of airway or in a shaft can best be obtained directly from the charts given in the appendix. It can also be calculated from the formula -

$$P = \frac{RQ^2}{5.2}$$

where P = Pressure drop in inches w.g.
R = Resistance in atkinsons.
Q = Air quantity in thousands of cubic feet per second.

Suitable values of R for different types and sizes of airway are given in Table V.

If a number of lengths of airway having resistance R_1, R_2 and R_3, etc. respectively, are in series, the total resistance R of all the airways is given by:-

$$R = R_1 + R_2 + R_3 \quad \text{etc.}$$

If the airways are arranged in parallel:

$$\frac{1}{\sqrt{R}} = \frac{1}{\sqrt{R_1}} + \frac{1}{\sqrt{R_2}} + \frac{1}{\sqrt{R_3}} \quad \text{etc.}$$

or, in the case of two airways only, in parallel:

$$R = \frac{R_1 R_2}{R_1 + R_2 + 2\sqrt{R_1 R_2}}$$

Faces

The resistance of a face when the packs have been built and the coal cut and blown is usually about five times the resistance when the face has been cleared.

The figures given in Table VI are the approximate resistances of strip-packed conveyor faces when the packs have been built to within 8 ft of the face.

Conveyor Roads

The presence of a conveyor in a road will have a very variable effect on the resistance of the road. A reasonable allowance would be to assume that it decreases the effective area of the roadway by 10 sq. ft.

Haulage Roads

The presence of tubs in a haulage road causes an appreciable increase in the resistance of the road. The amount of the increase will depend on many factors, such as the relative areas of the roadway and the end of the tubs, the number and spacing of the tubs, etc. and the allowances suggested later are therefore necessarily approximate.

Where locomotive, main-rope or main-and-tail haulage is to be used and the haulage road is long or is occupied only intermittently by tubs, the haulage road can be treated as a main airway and its resistance calculated from the resistance figure given for airways.

The resistance of roads used for endless haulage in which the tubs are spaced at intervals and of roads used as standing room for tubs, such as at loading points and pit bottoms, should be calculated from the resistance figures given for airways increased by 75 per cent.

Airways

It is most important that the estimated resistance of an airway should be calculated from the average cross-sectional area at which it is expected the airway will be maintained and not from the cross sectional area of the airway when it is first made.

An allowance should be made for each right-angle bend in an airway by adding 20 times the width of the airway to the length of airway on which the resistance is calculated.

Leaking Airways

It is sometimes necessary to estimate the pressure loss in an airway from which or into which air is leaking throughout its length. Examples of such airways are often found at mines where the main intake and return are separated by only a short width of goaf, or have been driven as headings connected at short intervals by cut-throughs or thirlings.

Provided that the resistances of the leakage paths are fairly uniform, a close approximation of the pressure loss in it can be obtained by calculating the mean value of the airflows Q_0 and Q_1 at the outbye and inbye ends respectively of the airway, and applying it to the appropriate chart in the appendix. The value of $\sqrt{Q_0 \, Q_1}$ will give greater accuracy if the leakage is high.

Shafts and Staple-pits

The resistance of 100 yd of circular shaft or airway, in atkinsons, can be obtained from the formula:-

$$R = \frac{3.84 \times 10^6 \times k}{A^2 \times \sqrt{A}}$$

Where R = resistance in atkinsons
k = Atkinson's coefficient in lb/sq. ft.
A = area of shaft in sq. ft.

The values of k can be obtained from Table VII.

A close approximation of the value of k for a shaft fitted with 10 in x 6 in steel joist buntons at 10 ft spacing is given by the formula:-

$$k = 0.002 + 0.0002L$$

where L is the total length of one set of buntons in feet.

Reducing the bunton spacing to 8 ft will increase the value of k by approximately 10 per cent. Increasing the spacing to 12 ft will reduce the value of k by approximately 10 per cent.

An allowance should be made for the resistance of the shaft top and of the insets by adding 40 times the diameter of the shaft to the actual length of the shaft for the purpose of calculating the resistance.

The resistances of shafts of shapes other than circular can be obtained by finding the resistance of a circular shaft of the same cross sectional area and multiplying it by the appropriate Shape Factor S_f in Table VIII.

To the calculated resistance should be added the estimated resistance due to the cage or cages, which will vary, according to the sizes and relative areas of the shaft and the cages.

The approximate resistances of cages and skips in a 24 ft diameter shaft are given in Table IX.

To obtain the value of R_c for shafts of other diameters, it can be assumed that R_c varies inversely as the fourth power of the shaft diameter. For example, the resistance of two cages in a 16 ft diameter shaft would be approximately $0.01 \times \left(\frac{24}{16}\right)^4 =$ 0.05 atk.

AIR VELOCITIES

It is not possible to specify precise figures for the ranges within which the air velocities in the ventilation system of a mine should fall, since these will depend on local circumstances.

High air velocities result in high pressure losses and, therefore, in high ventilation power costs. On the other hand, in order to reduce the air velocities, it is necessary to increase the size, and possibly also the number of the airways; the cost of airway formation and maintenance will then be correspondingly higher. For each airway there will be an optimum size (and therefore air velocity) which will give a minimum total airway and ventilation cost. The optimum size will, of course, depend on the quantity of air to be carried.

It is now generally recognised, however, that the danger to health and safety due to dust is often greater than that due to firedamp. High air velocities increase the difficulty of preventing dust being raised into the air at the faces, at loading points and in conveyor, haulage and travelling roads, and they can therefore only be accepted where the dust problem does not exist or where it can be solved satisfactorily by special means.

High air velocities are also objectionable because they may chill and cause discomfort to men travelling and working in the roads, especially in the intake airways near the pit bottom.

Although the air velocities to be used in each case will depend on local conditions, the ventilation system of a mine should normally be designed so that they lie within the following limits:-

Faces

Experience has shown that if the air velocity on a face exceeds about 500 ft/min the working conditions for the men become uncomfortable and there is usually difficulty in keeping the dust concentration in the air current sufficiently low. The air velocity on a face should therefore not exceed 500 ft/min when the packs have been built. This limits the quantity of air which can be passed through a normal conveyor face to about 5,000 cu. ft /min for each foot of working height.

The air velocity should not be less than 150 ft/min opposite the packs when the face has been cleared, in order to ensure comfortable working conditions and to clear away dust and shotfiring fumes.

In mines where the wet-bulb temperature is likely to exceed 75°F, the air velocity on the face opposite the packs at the beginning of the coal-filling shift should, if possible, be about 400 ft/min.

Conveyor Roads, Loading Points and Transfer Points

Air velocities exceeding about 600 ft/min in conveyor roads and at loading and transfer points may cause difficulties due to an excessive amount of dust being raised into the air and should, therefore, be avoided.

Main Airways and Haulage Roads

Air velocities exceeding about 1,000 ft/min in haulage and travelling roads necessitate effective dust prevention measures in dusty mines, and they will cause high pressure losses per unit length of airway. The mine ventilation system should be designed, so far as is practicable, so that the air velocities in the main airways are kept below this figure, except that velocities of up to 1,500 ft/min may be permitted in smooth lined airways not used for haulage purposes.

Shafts and Staple Pits

If the shaft air velocity exceeds 1,500 ft/min the power and pressure losses in most shafts will be high. In some cases it may be necessary to accept higher velocities, but velocities exceeding 2,500 ft/min should be avoided. If the air velocity in the shaft greatly exceeds 1,500 ft/min rigid guides are desirable, unless there is a large clearance between each cage and between the cages and the shaft walls.

Diesel Locomotive Sheds and Battery-charging Stations

The air velocity should not be less than 150 ft/min.

Fan Drifts

The air velocity in a fan drift should not exceed 2500 ft/min; but there is little to be gained from a velocity of less than 2500 ft/min in a well-designed drift.

COMPRESSION AND EXPANSION OF AIR IN SHAFTS

The barometric pressure in a shaft increases by approximately 1 in. of mercury for each 1,000 ft of depth. The pressure at the bottom of a shaft 1,000 yd deep will therefore be about 3 in greater than that at the surface, i.e. about 10 per cent greater. The volume of the air, as measured by an anemometer, will therefore increase by about 10 per cent when passing up the shaft.

For the purpose of estimating the quantity of air to be handled by the fan, the quantity of air in the upcast pit bottom should therefore be increased by 1 per cent for each 100 yd depth of the shaft.

VOLUMETRIC EFFICIENCY OF THE VENTILATION SYSTEM

The volumetric efficiency (i.e. the ratio of the total quantity of air passing through the faces to the quantity passing in the fan drift) of a well laid-out mine in which the ventilation system is maintained to a reasonably high standard will usually vary between 45 and 55 per cent in the case of mines in which the 'in-the-seam' method of working is practised, and between 55 and 65 per cent for mines laid out for horizon mining. If the volumetric efficiency of a planned scheme is found to lie outside these ranges, the scheme should be examined to see whether reasonable allowances have been made for leakage, ventilation of motor rooms, air expansion, etc.

TABLES AND SUMMARY OF DATA

NATURAL VENTILATION PRESSURE

TABLE I

N.V.P. FOR VARIOUS DEPTHS OF WORKING

Depth of Working	N.V.P.
yards	inches water-gauge
600	½
700	¾
800	1
900	1¼
1000	1½

FAN CAPACITY

TABLE II

QUANTITY OF VENTILATING AIR REQUIRED

Gas emission measured at the return end of the face per ton of coal	Quantity of air on faces per ton of daily output	Quantity of air in fan drift per ton of daily output
cu. ft.	cu. ft/min	cu. ft/min
1000	100	200
800	95	190
700	90	180
600	85	170
500	70	140
400	55	110
300	45	90
200	35	70
Less than 100	30	60

Note: These figures are approximate and relate to the conditions described in the notes given earlier (page)

AIR LEAKAGE ALLOWANCES

TABLE III

LEAKAGE ACROSS NEWLY FORMED GOAF

Distance between Intake and Return Gates	Leakage across Goaf as a Percentage of the Air on the Face
yd	per cent
50	20
100	10
200	5

Separation Doors - 3000 cu. ft/min
Air Crossings - 3000 cu. ft/min plus 3000 cu. ft/min per set of doors

TABLE IV

SURFACE LEAKAGE

Fan-drift W.G.	Surface Leakage
in.	cu. ft/min
5	25 000
10	35 000
15	45 000
20	50 000
25	55 000

AIR REQUIRED FOR HEADINGS, MOTOR-HOUSES, ETC.

Stone Headings - 8000 cu. ft/min
Small Drifts and Coal Headings - 5000 cu. ft/min
Motor, Haulage and Pump Houses - 5000 cu. ft/min
Battery Charging Stations - 10 000 cu. ft/min
Diesel Loco. Garages - 150 cu. ft/min per b.h.p. running
Diesel Loco. Haulage Roads - 150 cu. ft/min per b.h.p. running

PRESSURE LOSSES

Airways - See charts in appendix
Conveyor Roads - Deduct 10 sq. ft from area
Endless Haulage Roads - Add 75 per cent
Standing-room for Tubs or Mine Cars - Add 75 per cent

TABLE VI

FACE RESISTANCES IN SEAMS OF VARIOUS THICKNESS

Thickness of Seam Worked		Resistance of 100 yd of Face
ft	in	atkinsons
1	9	420
2	0	270
2	3	180
2	6	130
2	9	95
3	0	72
3	3	55
3	6	45
3	9	37
4	0	30
4	3	25
4	6	22
4	9	19
5	0	16
5	6	12
6	0	10
6	6	8
7	0	7

Shafts $R(100\text{ yd}) = \dfrac{3.84 \times 10^6 \times k \times S_f}{A^2 \times \sqrt{A}}$ R = Resistance in atk.
A = Area in sq. ft.

TABLE VII

VALUE OF 'k' FOR SHAFTS

	lb/sq. ft.
1. Timber-lined shaft with a middle line of buntons	0.0120
2. Brick-lined shaft with two lines of side buntons and one tie girder to each bunton	0.0120
3. As No, 1, but with no middle buntons	0.0090
4. As No. 2, but with no tie girders	0.0095
5. Tubbing-lined shaft with no guides or cages	0.0075
6. Brick-lined shaft with rope guides and water or air ranges	0.0040
7. As No. 6, but smooth concrete lined.	0.0035
8. Brick-lined unobstructed shaft.	0.0020
9. Smooth-lined unobstructed shaft.	0.0016

Notes:-

(1) A close approximation of the value of 'k' for a circular shaft fitted with steel joist buntons at a spacing of $\frac{5D}{12}$ *is given by the formula* $k = 0.002 + \frac{0.005L}{D}$*, where L is the total length of one set of buntons in feet, and D is the shaft diameter in feet.*

If the bunton spacing is reduced to $\frac{D}{3}$*, the value of k is increased by 10 per cent.*

If the bunton spacing is increased to $\frac{D}{2}$*, the value of k is reduced by 10 per cent.*

(2) To allow for the resistance of the shaft top and insets, add 40 times the shaft diameter to the actual length of the shaft in calculating the resistance.

(3) The resistance of shafts of shapes other than circular can be calculated by finding the resistance of a circular shaft of the same cross sectional area and multiplying it by the appropriate shape factor S_f *in Table VIII.*

TABLE VIII

VALUES OF THE SHAPE FACTOR - S_f

Shape of Shaft or Airway	S_f
Circular	1.00
Semi-circular	1.07
Square	1.13
Rectangular with sides in ratio 1 : 1½	1.15
" " " " " 1 : 2	1.20
" " " " " 1 : 3	1.30

To calculate the resistance add the estimated resistance due to the cages, which will vary according to the size and relative areas of the shafts and cages. Approximate values for a 24 ft dia shaft are as shown in Table IX.

TABLE IX.

APPROXIMATE RESISTANCE OF CAGES AND SKIPS
IN A 24-FT SHAFT

Cages or Skips	Resistance - R_c
	atkinsons
One 28-ton skip, 17 ft 0 in x 6 ft 6 in and balance weight	0.008
Two 14-ton skips, 13 ft 6 in x 6 ft 0 in	0.008
Two cages 18 ft 9 in x 4 ft 6 in	0.010
Four cages 12 ft 9 in x 4 ft 3 in	0.025

To obtain the value R_c *for shafts of other diameters, it can be assumed that* R_c *varies inversely as the fourth power of the shaft diameter, provided that the shafts and cages or skips are geometrically similar - e.g. the resistance of two cages in a 16 ft dia shaft would be approximately* $0.01 \times \left(\frac{24}{16}\right)^4 = 0.05$ *atkinsons.*

AIR VELOCITIES

Faces Not more than 500 ft/min or less than 150 ft/min
Deep mines 300 - 400 ft/min

Conveyor Roads, Loading and Transfer Points - Not more than 600 ft/min

Main Haulage Roads Less than 1000 ft/min

Main Airways Not more than 1000 ft/min or more than 1500 ft/min if smooth lined

Diesel Loco Garages - Not less than 150 ft/min

Battery Charging Stations - Not less than 150 ft/min

Shafts - Not more than 2500 ft/min
Preferably not more than 1500 ft/min

Fan drifts Not more than 2500 ft/min

COMPRESSION AND EXPANSION OF AIR IN SHAFTS

1 per cent per 100 yd depth 3/4 % per 100 m

VOLUMETRIC EFFICIENCY

In-the-seam mining	-	45-55 per cent
Horizon mining	-	55-65 per cent

VB/CIRC/71(26)

M E M O R A N D U M

FROM: D.E.R. Lloyd, H.Q. Ventilation Engineer, The Lodge. P.16695/2

TO: ALL AREA VENTILATION ENGINEERS
MR. C. BROWN, KENT AREA 5th May, 1971.

SUBJECT: Metrication - Airway Resistance

In S.I. units the coefficient of friction K is given by the relation:-

$$RQ^2 = p = \frac{ksv^2}{A}$$

where

p	=	pressure absorbed	(N/m^2)
s	=	rubbing surface	(m^2)
v	=	air velocity	(m/s)
A	=	cross sectional area	(m^2)
R	=	resistance	(Ns^2/m^8, gaul)
Q	=	air flow	(m^3/s)

Hence $R = ks/A^3$

Values of the resistance per 100 ~~miles~~ metres for roads supported by steel arches are given below for values of k equivalent to those used in Table V of Information Bulletin No. 55/153.

If you have any comments on these values, please let me know. D.E.R. Lloyd

TABLE V
RESISTANCE IN GAULS PER 100M OF AIRWAY

Cross Sectional area of airway m^2	SUPPORTED BY STEEL ARCHES: Smooth Concrete all round	Concrete slabs or timber lagging between flanges all round	Concrete slabs, timber or bricks between flanges to spring	Lagging behind arches - good - straight airways	Rough conditions with irregular roof sides and floor
	k = 0.0037	k = 0.0074	k = 0.0093	k = 0.0121	k = 0.0158
3	0.090	0.180	0.226	0.294	0.384
4	0.044	0.088	0.111	0.144	0.188
5	0.0253	0.051	0.064	0.083	0.108
6	0.0162	0.0324	0.041	0.053	0.069
7	0.0109	0.0218	0.0274	0.0357	0.0466
8	0.0078	0.0155	0.0195	0.0253	0.0331
9	0.0059	0.0118	0.0148	0.0193	0.0252
10	0.0045	0.0090	0.0113	0.0148	0.0193
11	0.0035	0.0071	0.0089	0.0116	0.0151
12	0.0028	0.0057	0.0071	0.0093	0.0121
13	0.0023	0.0047	0.0059	0.0076	0.0100
14	0.0019	0.0037	0.0047	0.0061	0.0080
15	0.0016	0.0033	0.0041	0.0054	0.0070
16	0.0014	0.0028	0.0035	0.0045	0.0059