VENTILATION SURVEY PROCEDURES AND DATABASE FOR USE WITH COMPUTER SIMULATION TECHNIQUES

by

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ABSTRACT

The Australian Coal Industry Research Laboratories Ltd. (A.C.I.R.L.) MINVENT computer program can be used to simulate existing and projected colliery ventilation requirements. New collieries can be planned using a database of ventilation parameters which were ascertained by A.C.I.R.L. surveys at selected collieries in N.S.W. and Queensland.

Several techniques are used in both the formulation of the database and in the evaluation of a ventilation system, prior to analysis by the MINVENT program.

The analysis of a ventilation system requires the measurement of air quantities, pressures and hence airway resistances. All resistances are calculated at standard air density of 1.205 kg/m^3. Air densities are derived from barometric pressure and wet and dry bulb temperature readings.

Airway frictional and shock losses, leakage through stoppings and overcasts, flow of air through regulators, face resistance and resistance of anomalous ventilation features are related to the typical Australian measured data presented.

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INTRODUCTION

Mine ventilation deals with the mine atmosphere environment. The principal objectives are to supply adequate fresh air to the working places and to dilute and carry away dust, noxious and toxic gases. Efficient ventilation is essential for mine operation.

In evaluation of a ventilation system the A.C.I.R.L. MINVENT program is used to simulate existing and projected ventilation requirements at various stages of mine development. Ventilation analysis is instrumental in planning an approach to the development of underground mines. The number and type of roadways to be driven, the size of the roadways, the amount of air to be supplied at given locations in the mine, the location and size of additional ventilation shafts or drifts and the number and size of fans required to service a given mine can be evaluated using this program.

VENTILATION SURVEY TECHNIQUES

AIR QUANTITY MEASUREMENTS

A ventilation survey is conducted with the objective of measuring the air density, quantity of air flowing and static pressure differences at selected locations in a mine ventilation system. The selection of suitable locations for the measurement of air quantities is of utmost importance. Locations should be chosen so that possible errors due to disturbed airflow conditions are decreased and an accurate determination of the roadway area can be achieved.

To obtain this a site should be chosen such that the airway is straight, has uniform size and shape, is smooth walled and free of obstructions.

Vance (1973) suggests that an air measurement station should be not less than three roadway diameters upwind of an obstruction and six roadway diameters downwind of an obstruction. Ideal sites are not common underground and a compromise of the best ideal site and those available underground is required so that a reliable determination of the airflow quantity can be achieved.

The most commonly used and recommended method for measuring air velocities is the traversing method. The anemometer is slowly moved across the airway or a series of imaginary vertical lines in such a way that equal areas are traversed in equal times. All measurements should be checked by repeated measurements until a satisfactory comparison of velocities is obtained. Precise traversing methods with an anemometer are only justified in major air currents.

At each location where the air velocity is measured the roadway dimensions, wet and dry bulb temperatures and barometric pressure are recorded. A standardised approach has been used for the measurement of roadway cross sectional area and perimeter. For belt roads the area of the belt is subtracted from the roadway area and the perimeter is calculated as the effective rubbing surface of the roadway (see Figure 1).

In most roadways unobstructed by belts two heights are sufficient to characterise the roadway, one at one third the roadway width and one at two thirds the roadway width. Similarly the roadway widths are measured at one third and two thirds the roadway height.

Once the air velocity and roadway area have been measured at a given location the calibrated corrected air quantity can be calculated by converting the measured air velocity to a standard air density and adjusting for any instrument calibration factor (assuming these are given at standard air density). That is

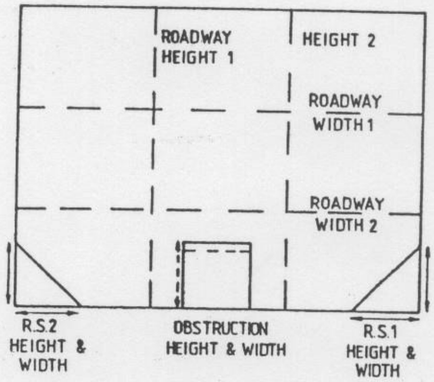

Fig. 1. Cross-sectional area of mine Roadway

$$V_s = V_m \sqrt{W_m/W_s} \qquad (1)$$

where V_s = air velocity at standard air density m/s
V_m = measured air velocity m/s
W_m = measured air density kg/m^3
W_s = standard air density kg/m^3

and

$$V_A = \alpha + \beta V_s \qquad (2)$$

V_A = calibrated corrected velocity m/s
α & β = least squares regression coefficients for the line of best fit to the calibration curve for the anemometer.

The measured air density can be calculated from

$$W_m = \frac{(P - e)\ (1 + x)}{R_a\ (273.15 + t_d)}\ \text{kg/m}^3 \qquad (3)$$

where P = barometric pressure (Pa)
e = non-saturated vapour pressure (Pa)
x = moisture content
R_a = gas constant for dry air Nm/kg^{o}K
t_d = dry bulb temperature (^{o}C)

PRESSURE MEASUREMENTS

Pressure differences are necessary to cause airflow between nodes in an air circuit or mine. Fans provide mechanically produced pressure between nodes in a mine ventilation circuit. In addition to mechanical pressure, barometric pressure influences airflow in a mine. The sum of mechanical pressure and barometric pressure (with the fan turned off) is the mine ventilation pressure.

Static pressures indicate the potential for air movement and can be measured by the trailing hose method. This involves the use of either a magnehelic pressure gauge or inclined manometer connected to lengths of tube. Care should be taken so as to avoid using excessive lengths of tubes since with tube lengths in excess of 75 m the internal resistance of the tubes may significantly affect the pressure readings.

By the application of Kirchoffs second law the sum of pressure drops around a closed loop in a mine ventilation circuit must equal zero. Therefore the trailing hose method in conjunction with short circuit pressure drop readings from intake to return roadways can be used to ascertain the pressure drop over hazardous or inaccessible areas in a mine ventilation system.

ENERGY LOSSES IN A MINE VENTILATION SYSTEM

The pressure head developed by a fan is expended in overcoming the resistance of the mine to airflow. The pressure losses in the system can be categorised into frictional and shock losses.

FRICTIONAL PRESSURE LOSSES

Frictional pressure losses are caused by resistance to the air stream and depend on parameters such as:

(i) type of surface, seam type,
(ii) percent of the roadway obstructed,
(iii) peripheral obstructions, condition of the ribs,
(iv) irregularities, and
(v) geometry, roadway shape.

Frictional pressure losses in straight sections of roadway account for the major pressure losses in a mine ventilation system. All formulae for frictional pressure losses include an empirical factor determined by actual experiment. This factor is commonly referred to as the coefficient of friction or K factor. Atkinson formula (equation (4)) for pressure drop gives the basic relationship between the quantity of air flowing in a roadway, its dimensions, frictional characteristics and head loss.

$$P = K\,\frac{CLV^2}{A} \cdot \frac{W}{W_s} \qquad (4)$$

where P = static pressure (Pa)
C = perimeter of airway (m)
A = area of airway (m^2)
L = length of airway (m)
V = velocity (m/s)
K = frictional factor
W = actual air density (kg/m^3)
W_s = standard air density (kg/m^3)

The roadway geometry effects the frictional resistance. The roadway shape factor can be used to compare roadways of different shapes provided the roadway rubbing surface, spacing and size of props and bars and other irregularities remain constant. McElroy (1935) derived a roadway shape factor (F), which is proportional to the roadway perimeter divided by the square roof of the roadway area.

$$F = \frac{C}{3.545\sqrt{A}} \qquad (5)$$

The constant of proportionality (i.e., 3.545) in the above equation is derived by assuming the relative shape factor of a circular airway to be unity. All other airway shapes can be compared to a circular airway.

The hydraulic radius of an airway can be defined by

$$Rh = A/C \text{ (m)} \qquad (6)$$

From Atkinson equation it can be seen that

$$P \propto \frac{1}{R_h} \qquad (7)$$

McElroy (1935) observed that the coefficient of friction was maximum when the timber sets were 1.5 m apart; its value falling off for both increasing and decreasing spacing. This phenomenon is understandable if the timber sets are considered to offer shock resistance in addition to frictional resistance.

Table 1 is a list of roadway resistances and K factors relative to particular coal seams.

SHOCK LOSSES

These are caused by changes in airflow direction and/or roadway cross sectional area.

Changes in airflow direction

For rectangular roadways the effect of sinuosity can be accounted for by equation (8). This equation relates the shock loss factor to the roadway geometry and was derived from McElroy (1935).

$$X = \frac{0.6}{m\,\alpha^{0.5}}\left(\frac{\theta}{90^\circ}\right)^2 \qquad (8)$$

where X = shock loss factor (dimensionless)
m = Rh/width (dimensionless)
R_h = hydraulic radius (m)
C = perimeter of roadway (m)
A = area of roadway (m^2)
α = aspect ratio (H/W)
H = roadway height (m)
W = roadway width (m)
θ = angle of bend

Example: Shock loss for a 90° bend K factor $= 2.87 \times 10^{-2}$

$C = 15.4$
$A = 12.1$
$H = 2.2$
$W = 5.5$
$R = \frac{KCL}{A3} = 0.02494/100$ m

now $X = \frac{0.6}{m\,\alpha^{0.5}}$

$X = 6.645$

assume velocities of 1, 2 and 3 m/s

$$\therefore\; P_v = \frac{V^2 W}{2} \text{ Pa} \qquad (9)$$

$$H_v = X\,P_v \qquad (10)$$

P_v = 0.6025, 2.41 and 5.423 (Pa) respectively
H_v = 4.009, 16.01, 36.04 (Pa) respectively
Q = 12.1, 24.2, 36.3 m^3/s
R = 0.02735, 0.02734, 0.02735 N_s/m^8

Hartman (1961) derived a method for calculating shock losses. He assumed they are equivalent to an additional length of roadway,

TABLE 1

LIST OF ROADWAY RESISTANCES AND K FACTORS

RELATIVE TO PARTICULAR COAL SEAMS

SEAM	MINE	ROADWAY	AREA	PERIMETER	RESISTANCE	K FACTOR
1.	1.	1.	14.60	17.20	0.00909	0.01645
1.	1.	1.	15.10	17.90	0.00824	0.01585
1.	1.	1.	15.50	17.80	0.01140	0.02385
1.	1.	1.	13.90	16.90	0.00940	0.01494
1.	1.	1.	16.20	17.70	0.01010	0.02426
1.	1.	2.	12.50	18.40	0.01300	0.01380
1.	1.	2.	12.90	18.30	0.01290	0.01513
1.	1.	2.	16.60	20.80	0.01360	0.02991
1.	1.	3.	15.20	17.20	0.00750	0.01531
1.	1.	3.	17.20	18.90	0.00730	0.01965
1.	1.	3.	15.70	18.00	0.01100	0.02365
1.	2.	1.	15.40	18.30	0.00800	0.01597
1.	2.	1.	13.90	18.70	0.00610	0.00876
1.	2.	4.	16.05	18.30	0.00585	0.01322
1.	2.	4.	14.20	16.70	0.00907	0.01555
1.	2.	4.	15.50	17.50	0.00490	0.01043
1.	2.	3.	15.00	19.00	0.00913	0.01622
1.	2.	2.	15.80	19.20	0.01860	0.03821
1.	3.	1.	12.30	15.50	0.01480	0.01777
1.	3.	1.	11.10	14.80	0.01720	0.01589
1.	3.	1.	11.70	15.10	0.01730	0.01835
1.	3.	1.	13.70	14.90	0.01090	0.01881
1.	3.	2.	12.60	18.10	0.01780	0.01967
1.	3.	4.	12.20	15.30	0.02470	0.02931
1.	3.	4.	15.00	16.60	0.01490	0.03029
2.	4.	3.	11.46	13.79	0.02380	0.02598
2.	4.	3.	15.33	16.87	0.00779	0.01664
2.	4.	3.	17.10	17.40	0.00860	0.02471
2.	4.	3.	13.80	15.80	0.00670	0.01114
2.	4.	2.	12.50	17.90	0.01200	0.01309
2.	4.	4.	18.90	19.20	0.01690	0.05943
2.	5.	3.	12.96	14.89	0.00587	0.00858
2.	5.	1.	12.70	15.50	0.00381	0.00504
2.	5.	4.	11.60	13.90	0.01500	0.01684
3.	6.	3.	15.50	18.00	0.01420	0.02938
3.	6.	3.	14.40	16.90	0.01570	0.02774
3.	6.	3.	14.60	17.10	0.01220	0.02220
3.	6.	3.	14.60	17.10	0.01630	0.02967
3.	6.	4.	14.91	17.60	0.02630	0.04953
3.	6.	4.	13.60	17.30	0.02760	0.04013
3.	6.	2.	13.90	18.80	0.02830	0.04043
4.	7.	3.	17.90	18.82	0.01250	0.03809
4.	7.	1.	16.80	18.20	0.01380	0.03595
4.	7.	2.	12.70	18.70	0.02290	0.02508
4.	7.	4.	18.70	19.20	0.00380	0.01294
4.	7.	4.	17.10	17.70	0.00453	0.01280
5.	8.	1.	14.10	16.30	0.00313	0.00538
5.	8.	1.	14.60	17.00	0.00296	0.00542

TABLE 1 (continued)

SEAM	MINE	ROADWAY	AREA	PERIMETER	RESISTANCE	K FACTOR
5.	8.	1.	12.90	17.80	0.00440	0.00531
5.	8.	2.	13.00	19.00	0.03500	0.04047
5.	8.	2.	12.60	18.50	0.03900	0.04217
5.	8.	2.	12.80	18.90	0.04160	0.04616
5.	8.	4.	12.70	15.60	0.00446	0.00586
5.	8.	4.	13.10	16.20	0.00389	0.00540
6.	9.	3.	14.00	15.50	0.01600	0.02833
6.	9.	3.	13.40	15.90	0.01510	0.02285
6.	9.	5.	12.40	13.40	0.02760	0.03927
6.	9.	5.	12.00	17.80	0.02230	0.02165
6.	9.	5.	14.40	18.50	0.02350	0.03793
6.	9.	6.	11.40	15.40	0.05020	0.04829
6.	9.	4.	13.30	19.00	0.02630	0.03257
6.	9.	4.	13.20	18.50	0.02070	0.02573
6.	9.	4.	14.40	18.20	0.01810	0.02970
7.	10.	3.	14.10	16.70	0.01350	0.02266
7.	10.	1.	17.60	19.00	0.01000	0.02869
7.	10.	2.	13.60	17.30	0.01870	0.02719
7.	10.	4.	14.20	16.50	0.01100	0.01909
7.	10.	4.	13.50	15.80	0.01310	0.02040
8.	11.	1.	32.40	24.50	0.00039	0.00541
8.	11.	1.	24.80	22.30	0.00219	0.01498
8.	11.	2.	31.00	24.90	0.00517	0.06186
8.	11.	1.	11.50	13.40	0.01686	0.01914
8.	11.	1.	10.90	12.60	0.01964	0.02019
8.	11.	1.	11.20	13.10	0.01509	0.01618
8.	11.	4.	29.40	23.00	0.00292	0.03226
8.	11.	4.	41.80	28.20	0.00054	0.01399
9.	12.	3.	17.40	20.30	0.00498	0.01292
9.	12.	3.	13.60	15.00	0.01250	0.02096
9.	12.	2.	14.40	20.50	0.02080	0.03030
9.	12.	2.	14.90	19.70	0.01440	0.02418
9.	12.	4.	17.10	19.80	0.00295	0.00745
9.	12.	4.	19.90	21.70	0.00432	0.01569
10.	13.	4.	25.90	21.50	0.00192	0.01552
10.	13.	4.	23.00	20.30	0.00197	0.01181
10.	13.	2.	22.80	22.80	0.01440	0.07486
10.	13.	2.	24.10	24.00	0.00880	0.05132
10.	13.	1.	24.50	20.80	0.00297	0.02100
10.	13.	1.	22.80	19.70	0.00297	0.01787
10.	13.	1.	27.30	22.20	0.00219	0.02007
10.	13.	1.	25.00	21.10	0.00220	0.01629
11.	14.	1.	16.90	18.70	0.00715	0.01846
11.	14.	1.	16.60	18.30	0.00766	0.01915
11.	14.	1.	21.50	22.80	0.00374	0.01630
11.	14.	1.	18.00	21.00	0.00762	0.02116
11.	14.	4.	16.30	17.30	0.01160	0.02904
11.	14.	4.	19.40	19.70	0.01810	0.06708
11.	14.	4.	18.30	18.70	0.01100	0.03605
11.	14.	2.	17.90	21.50	0.02110	0.05629

that is

$$R = \frac{KC\ (L + L_e)}{A^3} \qquad (11)$$

In the preceding example the shock resistance loss remained constant for varying velocity heads. Therefore the assumption of equivalent lengths of roadway to represent shock losses is valid.

The equivalent length from the previous example is

$$L_e = \left(\frac{(0.2735 - 0.02494)}{0.02494} \times 100\right) + 100$$

$$= 109.6 \text{ m}$$
$$= 110 \text{ m}$$

In the Greta seam two ninety degree bends were surveyed at two different collieries. The roadway resistances per 100 m were 2.38 x 10^{-2} and 1.50 x 10^{-2} kg/m^7. The equivalent resistance of the bends in the roadway were 2.66 x 10^{-2} and 1.69 x 10^{-2} kg/m^7.

The equivalent roadway lengths were:

1. $$L_e = \left(\frac{(2.66 \times 10^{-2} - 2.38 \times 10^{-2})}{2.38 \times 10^{-2}} \times 100\right) + 100$$

$$= 112 \text{ m}$$

2. $$L_e = \left(\frac{(1.69 \times 10^{-2} - 1.50 \times 10^{-2})}{1.5 \times 10^{-2}} \times 100\right) + 100$$

$$= 113 \text{ m}$$

Sinuosity related shock losses are proportional to an equivalent length of roadway resistance and are dependent of the quantity of air flowing in that given roadway.

CHANGES IN ROADWAY CROSS-SECTIONAL AREA

In a mine ventilation circuit regulators are used to distribute the flow of air as required throughout the colliery. As air approaches a regulator the velocity of flow increases and continues to do so for a short distance beyond the regulator. This is unknown as a vena contracta (see Figure 2).

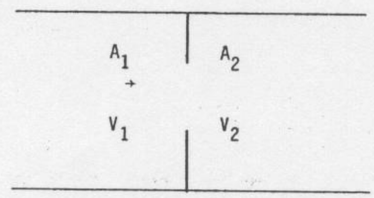

Fig. 2. Section of airway with a regulator

The Bernoulli equation is used for the basis of calculation of the velocity of air and hence airflow through the regulator (Mine Ventilation Society of South Africa, 1974). With the assumption that the roadway is horizontal and the air density remains constant the velocity flowing through the regulator (V_2) can be determined from

$$V_2 = \sqrt{\frac{2P}{W((1 - A_2^2/A_1^2))}} \qquad (12)$$

where V_2 = velocity of flow through the regulator (m/s)

V_1 = velocity upwind of regulator (m/s)

A_1 = roadway area upwind of regulator (m^2)

A_2 = area of regulator (m^2)

W = standard air density (kg/m^2)

$$\therefore \frac{A_1^2\ 1.66P}{V_2^2} = A_1^2 - A_2^2 \quad \text{(from (12))}$$

since $Q_2 = V_2 A_2$

then $$\frac{A_1^2\ A_2^2\ 1.66P}{Q_2^2} = A_1^2 - A_2^2$$

$$\therefore \quad A_2^2 = \frac{A_1^2 \quad Q_2^2}{A_2^2 + A_1^2\ 1.66P}$$

since $Q_2 = Q_1$

then

$$A_2 = A_1 Q_1 \sqrt{\frac{1}{Q_1^2 + A_1^2\ 1.66P}} \quad (m^2) \qquad (13)$$

Smith (1980) conducted a series of experiments to measure the flow through regulators. Using equation (13) and Smith's data, the calculated regulator areas were compared with the measured regulator areas. A T-value sum of the squares regression was computed. The goodness of fit was 99.5% (see Figure 3). Therefore equation (13) is better expressed as:

$$A_2 = C^* A_1 Q_1 \sqrt{\frac{1}{Q_1^2 + A_1^2\ 1.66P}} \quad (m^2) \qquad (14)$$

where $C^* = 1.383$

The Bernoulli equation does not take into consideration any frictional or shock losses across the regulator.

Equation (14) does take these into consideration.

ENERGY LOSSES DUE TO LEAKAGE

Leakage remains as one of the greatest problems in mine ventilation analysis. The amount of air which escapes through stoppings, doors, air crossings and shafts is considerable. In operating mines today it is not common to lose 50% of the air before it reaches the working face.

High leakage values cause greater quantities of air to be handled by the fan at higher pressures to deliver a particular quantity of air to the working areas. This results in higher power costs and dust problems due to higher velocities within the mine ventilation system itself.

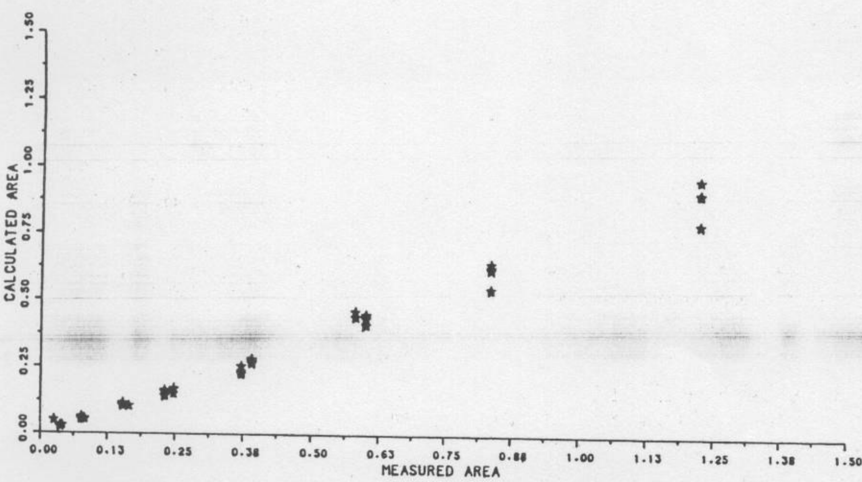

Fig. 3. Regulator Area Calculated vs Measured Area

NATURE OF AIRFLOW THROUGH STOPPINGS

Assume a quantity of 0.1 m^3/s of air flows through a permanent stopping at a pressure drop of 500 pascals and that the stopping has an area of 10 m^2 and a thickness of 0.15 m (h).

The total equivalent pore space can be calculated from equation (14).

$$A_2 = C^* h\, A_1 Q_1 \sqrt{\frac{1}{Q_1^2 + A_1^2 \; 1.66P}} \qquad (15)$$

$$= 0.0048\, h$$
$$= 0.0078\ m^3$$

where h = thickness of the stopping

The total equivalent pore space is the volume occupied by a finite number of pores which are present in the material which constitutes the stopping. If each pore is considered to be a smooth cylindrical pipe which traverses the medium then the critical value for Reynolds number will be 2,000. Therefore the critical pore diameter, which is the diameter below which the flow will be laminar, can be calculated from:

$$D_c = \frac{R_e \quad \mu}{V.W} = 0.0028\ (m)$$

where μ = 2.3 x 10^{-4} Pa s (for air at 20^oC)

R_e = Reynolds number (2000)

V = velocity of flow through each pore (m/s)

W = air density (kg/m^3)

It is unlikely that the pore diameters in a stopping are 2.8 mm or greater. Consequently the flow of air through a mine stopping can be assumed to be laminar. The porosity of the stopping in this example can be ascertained from the Carman-Konzeny (1979) equations:

$$e = \frac{V_v}{V_v + V_s} \qquad (16)$$
$$= 4.8 \times 10^{-4}$$

where e = porosity

V_v = volume of voids (m^3)

V_s = volume of solid (m^3)

If the porosity reflects a series of tortuous channels then

$$V = \frac{d_m \; \Delta P}{K \mu l} \qquad (17)$$

where V = mean velocity (m/s)

d_m = equivalent diameter of the pore channels (m)

K = dimensionless constant whose value depends on the structure of the bed

l = length of the channel (m)

μ = viscosity (Pa.s)

Konzeny further proposed that:

$$d_m = \frac{e}{s\ (1-e)} \qquad (18)$$

where s = total surface area of the voids (m^2)

From equation (18) Coulson (1968) developed equation (19):

$$V = \frac{1}{K} \quad \frac{e^3}{s^2\ (1-e)^2} \quad \frac{\Delta P}{\mu l} \qquad (19)$$

where $\frac{1}{K} \quad \frac{e^3}{s^2\ (1-e)^2}$

is the permeability coefficient.

If more information on the pore size in a material which will be used for a mine stopping is evaluated then the flow characteristics through the stopping will be better understood.

"When the mean free path of the molecules of a gas is a significant fraction of the capillary diameter, the flow rate at a given value of pressure gradient becomes greater than the predicted value. If the mean free path length exceeds the capillary diameter the flow becomes independent of the viscosity and the process is one of diffusion" (Coulson, 1968).

An alternative approach to estimate the equivalent leakage resistance offered by a stopping to airflow is to measure the quantity loss over a given number of stoppings and to calculate an equivalent leakage resistance for those stoppings. This approach for measuring stopping leakage values has the advantage of accounting for irregularities in the construction of individual stoppings. A larger number of stoppings can be surveyed using this approach (see Figures 4 and 5).

Four quantity stations can be chosen and the quantity loss over a given number of stoppings can be calculated. The use of four quantity stations is advisable so that losses in intake roadways can be compared with quantity gains in return roadways. The equivalent leakage resistance can be represented by a mid point leakage path. A rate leakage per mine development can be ascertained and used in the future modelling of colliery development. The validity of this technique depends on an individual stopping leakage response to varying pressure.

Generally leakage is most severe through old stoppings in the outbye portion of the circuit. They are subject to higher pressure than the newer inbye stoppings. Therefore the circuit air quantity decreases at a diminishing rate from outbye to inbye in a mine.

CONCLUSIONS

When individual airway resistances, and the quantity of flow in a given network have been defined the network can then be modelled using the A.C.I.R.L. MINVENT program. Normalised roadway resistances (for a given roadway geometry) individual stopping leakage resistances, overcast resistances (both leakage and the resistance offered to the flow across them) and working face resistances, measured at a given

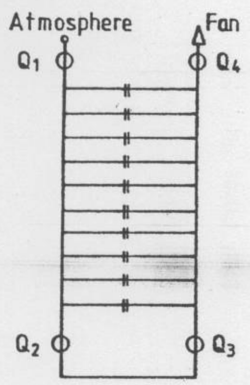

Fig. 4. Simplified Mine Plan

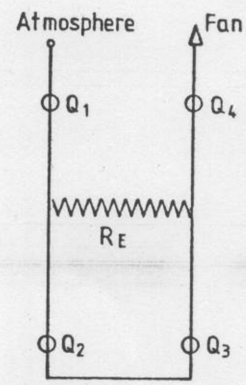

Fig. 5. Schematic Network Diagram of Mine Plan

location can be used to simulate future Colliery ventilation requirements. New collieries can be planned by referring to A.C.I.R.L.'s database of ventilation parameters.

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DISCUSSION

A. RALPH (A.C.I.R.L., North Ryde):
On page 7-9 of the paper equation (15) solves to 7.2×10^{-4} m^3 and the critical diameter is 18 mm (i.e., DC = 18 mm).

The nature of flow through a mine stopping becomes significant when the pressure is varied across the stopping. As an example consider five identical stoppings which at a pressure of 500 pascals leak 0.1 m^3/s of air each. That means that over 5 stoppings 0.5 m^3/s of air leak. Assuming that leakage obeys the square law (i.e., $P = RQ^2$) then the equivalent leakage resistance for all of the five stoppings would amount to a combined total of 2000 Ns^2/m^8. Assuming leakage to be laminar (i.e., P = RQ) then the equivalent leakage resistance for the same five stoppings would be 1000 Ns^2/m^8 . At this pressure each equivalent resistance will yield the correct leakage value. If for some reason the ventilation requirements increase such that the pressure differential across the same five stoppings is now 1000 pascals the leakage assuming the square law will be 0.7 m^3/s of air and assuming laminar flow it will be 1 m^3/s of air. This represents a difference in leakage estimation of 30 per cent over these five stoppings. For the reasons outlined in the paper the flow through mine stoppings is considered to be typically laminar.

Harvey (Paper No. 24) agreed with these findings, in that with changing pressures it was noticed that leakages did not appear to obey the square law.

I. MORRIS (National Coal Board, Doncaster): Do the leakage relationships include for flow around the periphery of the stoppings?

A. RALPH: It would be necessary to characterise the individual leakage path. If the flow around a stopping was, in terms of a parallel air passage and that passage had a diameter greater than 18 mm and all the criterions were similar to the worked out example it would be turbulent flow and just simply a branch could be put in where $P = RQ^2$. If those criteria were not met then the branch could be simulated with P = RQ. It would depend on the individual branch in question.

I. MORRIS: Yes, but during most site examinations it has been observed that the largest air flow associated with a stopping occurs around and not through the face.

A. RALPH: It is difficult to quantify whether it's going around or through, it would depend on how competent the stopping was, how fractured the coal was. Two different techniques have been used to measure the amount of air that flows through a stopping. The amount of air that either flows through a certain number of stoppings or across an individual stopping has been measured. The value that is of interest is perhaps a repetitive value, looking at future developmental work, a value which is characteristic of that unique feature underground and a value which responds to increasing pressures to give the appropriate quantity that should flow through it at a large pressure.

F. KISSELL (Bureau of Mines, Pittsburgh): Bureau of Mines experience is that most of the air flow is by cracks on the edge. That can be tested rather easily by going to a stopping or series of stoppings and measuring a leakage through them, then, painting the stopping but not the edge. If the leakage changes, it is coming through the edge. It will be found that most of the time leakage is through cracks at the edge. Also it will be found that it follows a pressure-quantity square law.

A. RALPH: It would be a function of the permeability of the coal and the coal may be more permeable than the bricks that are used for the individual stopping. It is unlikely that the coal around a stopping has air passages in the order of 18 mm in diameter.

G. CROFT (Gunnedah Colliery): What repeatability on the air-flow readings is achievable? These readings are significant in determining the leakages through stoppings.

A. RALPH: Repeatability of the ratings within 10% should be achievable - if not then there is something fundamentally wrong with the technique. Choosing the station is of utmost importance, if a station is chosen where there is streaming across the roadway problems will be encountered. One of the problems found with a streamed airflow, is that the anemometer is a device which responds quite rapidly to changes in airflow speed only going from low to high speed. So that in one direction across the roadway the result is an anomalously high value, depending which way traverse is done. So provided the station is chosen carefully, 10% is a reasonably good value, with practise it could get better than that.

G. CROFT: There are problems still with determining leakage through various areas of the mine if 10% is the case.

Management has asked the question - what improvement can be expected to the airflow if one or other of the commercial treatments is applied to stoppings? To answer this question it is necessary to have written data on the effect of the applications. Is there any such information?

A. RALPH: To evaluate this process the techniques available to measure airflow through stoppings were:

1. The Brattice Window technique
2. Using air quantity stations over a given number of stoppings to measure total drop in air and to assume that that air has obviously gone through a number of stoppings and to ignore the inhomogeneity of each individual stopping.

Then an average or a representative value of a particular leakage path is derived. If it was desired to look at an individual material and its effect the porosity and permeability of that particular material could be evaluated to get an indication of how that material will respond to the flow of air through it. In doing material evaluations it would be necessary to adopt this approach.

G. CROFT: On page 7-3 of the paper static pressure has been quoted as being measured in the surveys, it would appear that total pressure is what should be looked at for in modelling ventilation circuits.

A. RALPH: Fan curves are supplied with both static pressure versus air quantity and total pressure versus air quantity. Knowing the potential energy (static pressure) which needs to be overcome to service a particular ventilation system then a fan can be chosen accordingly. Knowing the static pressure in a given airway and the velocity flowing in that airway the total pressure can be computed. Particular attention is required to the sign of the static pressure measurements. The total pressure readings can be lower than the static pressure reading if the static pressure is negative. This is improtant since most pressure readings underground are small and present measurement difficulties.

G. CROFT: Please comment on the statement by Kissell concerning measurement of air flow through stoppings - how can 0.1 m^3/s be measured in the heading that's 5.5 metres wide and 2.5

metres high?

A. RALPH: There is a technique the U.S.B.M. uses called the Brattice Window technique. A stopping is erected behind a permanent stopping, then holes are cut in that stopping which reduces the effective cross-sectional area of the road. The velocity then increases. The problem with this technique is that there is leakage around the temporary stopping which is erected to measure the leakage around the first stopping.