

four methods of obtaining Q_1 :-

- i) Providing Q_1 is not more than 15 - 20% of the total desorbable gas content ' Q ' then:-

$$Q_1 = k t^n$$

k - a constant

n - 0,3 - 0,5 depending on the type of coal, but usually 0,5 is taken.

- ii) Alternatively Q_1 can be determined graphically by extrapolating from time $t = 0$ when coring intersected the coal, until time t when the core was inserted in the cylinder (FIGURE 19). Sometimes this method is quite rough (13).

- iii) To get Q_1 , normally a reading ' q ' is taken by desorbing the sample for the same amount of time t , that it took to extract the sample and place it in the canister (5).

Then for $n = 0,5$

$$Q_1 = q(2 + 2)$$

where $q = V_{\text{flask}} \times \frac{P}{P_f}$

P_f - atmospheric pressure
underground

P - pressure rise.

- iv) In Germany Q_1 is simply considered as 10% of $Q_2 + Q_3$.

- b) By allowing gas to desorb and measuring it as above (i - iii) until the rate of emission is below 0,05 cm^3/g for 5 days ' Q_2 ' is determined (53). $Q_1 + Q_2$ is the desorbable

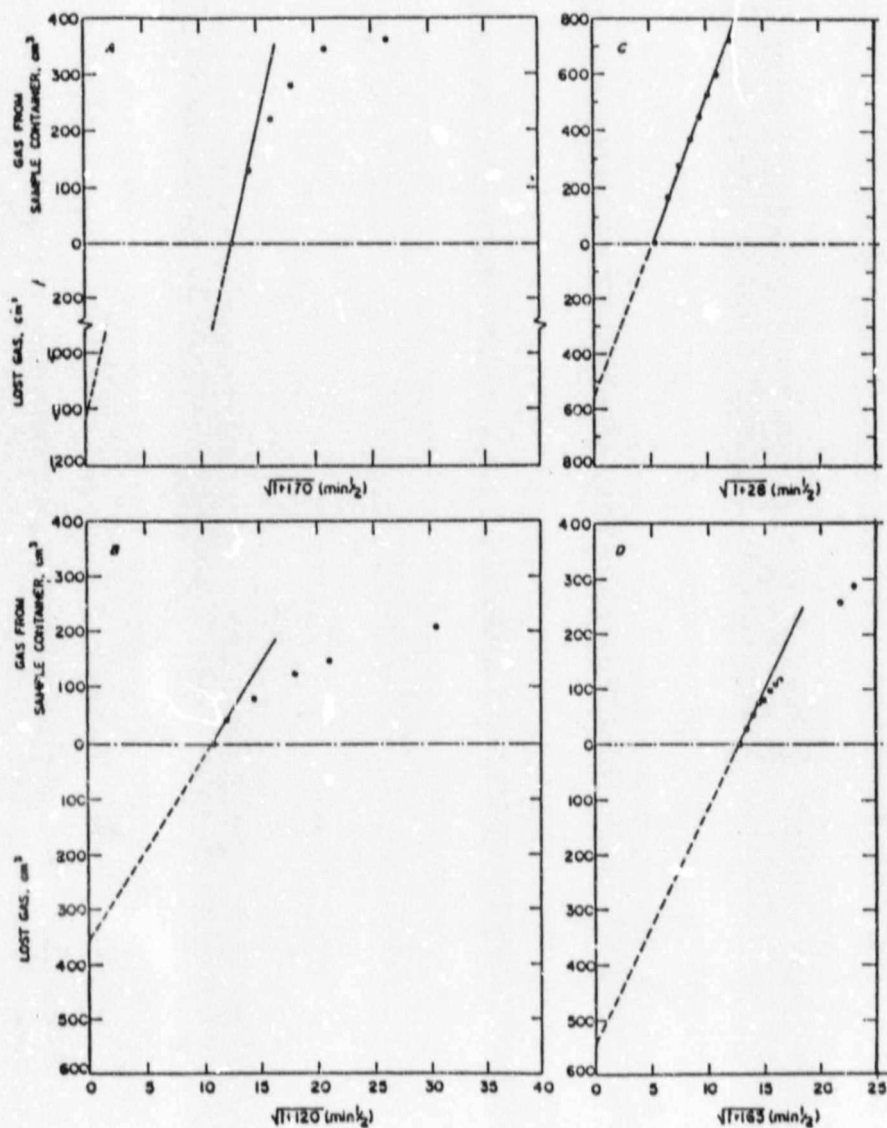


FIGURE 19 • Emission rate curves: A, Core from Inland site 1—Illinois No. 6 coalbed; B, core from Inland site 2—Illinois No. 5 coalbed; C, core from Kepler site—Pocahontas No. 3 coalbed; and D, core from Price site—Castlegate coalbed (subseam 3). (45)

gas content of coal which is considered relevant to the flow of methane into workings. In the literature this figure is always expressed with respect to dry, ash free coal given by:-

Mass of clean coal = $m (1 - 1,1c)$

m - mass of sample

c - ash content

- i) If the desorption meter method is used then:-

$$Q_2 = V_{\text{flask}} \frac{\bar{X} - X_0}{1 - X}$$

X_0 - initial methane content of the flask

X - final methane content of the flask before the sample is removed for determining Q_3 .

- ii) If the test tube and level meter method is used then

$$Q_2 = b_1 \left(\frac{P_J}{P_F} \times \frac{T_F}{T_J} \right) - h_0$$

P_J = atmospheric pressure on surface

P_F = atmospheric pressure underground

T_J = absolute temperature on surface

T_F = absolute temperature underground.

- c) The residual gas ' Q_3 ' is dependent on whether coal is blocky or friable and is found by one of the methods described below.

- i) crushing the sample to 200 mesh in a nitrogen atmosphere to prevent oxidation of the pulverised coal (16).

ii) Graphical means provided the coal can be classified as friable or blocky. For friable coals only 6% of their gas content Q_t is retained as residual gas but for blocky coals this figure can be 60%. Friability is related to fixed carbon FC (blocky <57% <friable) and Hardgrove Grindability Index (blocky <70 <friable). Other less certain but none the less correlating factors are depth (blocky <200m < friable) tectonic activity, cleat spacing and fusinite content. Results obtained by this method are shown in FIGURES 20 to 23 inclusive (53).

d) The total gas content ' Q_t ' is given by:-

$$Q_t = Q_1 + Q_2 + Q_3 \text{ m}^3/\text{t}$$

Often for comparisons this figure is converted to standard temperature and pressure (STP) (0°C, 101,325 kPa) by:

$$Q_t(\text{STP}) = Q_t \left(\frac{P_F}{101,325} \times \frac{273,3}{273,3 + T_F} \right)$$

or to atmospheric parameters for the underground site. It is considered that, with care, this method is accurate to within $\pm 0,5 \text{ m}^3/\text{t}$.

2.3.2 Indirect method

Coal samples from various areas of the coalbed are commonly used to determine adsorption isotherms using the indirect methods. Once these isotherms have been plotted, only pressure and temperature readings need be

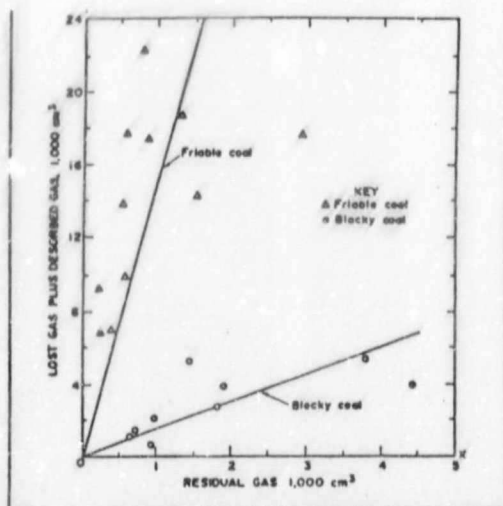


FIGURE 20 Graph for estimating remaining gas in a sample. (53)

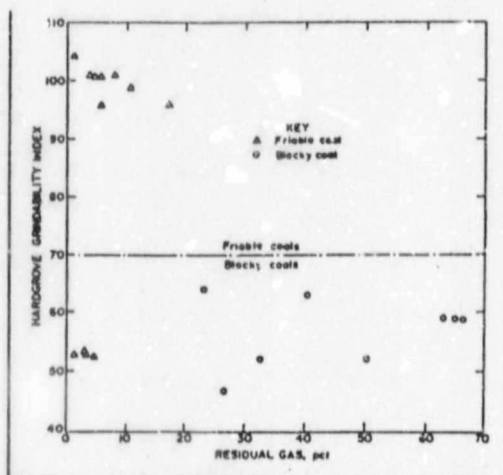


FIGURE 21 Residual gas vs Hardgrove Grindability Index. (53)

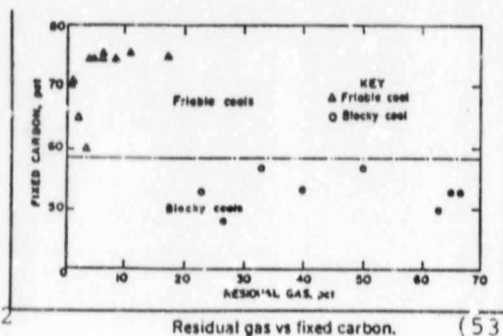


FIGURE 22 Residual gas vs fixed carbon. (53)

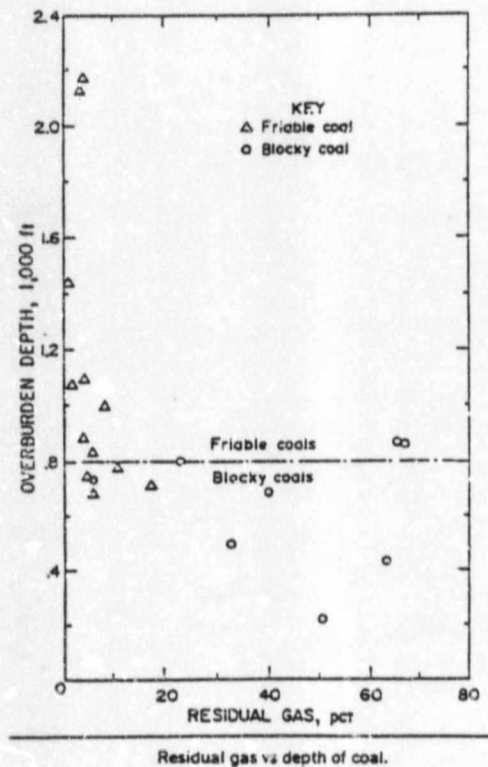


FIGURE 23 Residual gas vs depth of coal (53).

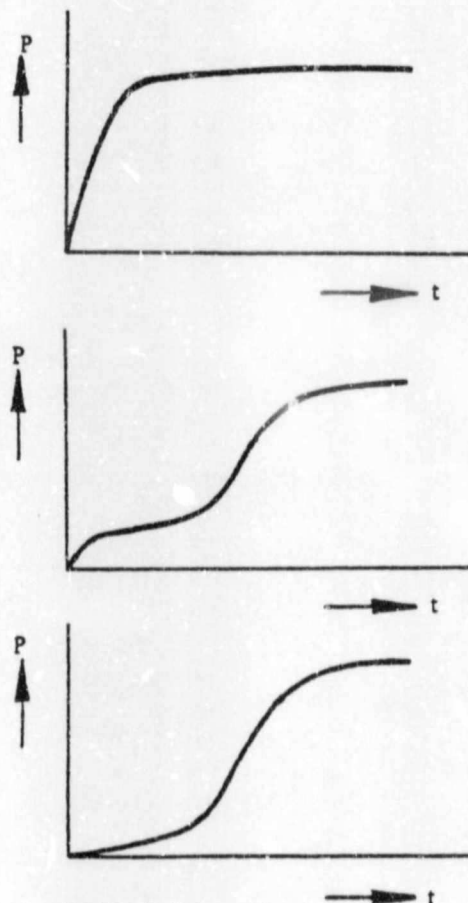


FIGURE 24 Typical forms of pressure - time curves in boreholes into coal seams (14).

measured in-situ. This is done by sealing the face area of the borehole either pneumatically by means of an expanded rubber collar, or mechanically. The gas pressure is taken as the in-seam gas pressure. Care must be taken if water is used for drilling as the hydrostatic head can lead to incorrect pressure readings. This can be avoided by recognising the shape of the pressure build-up curve as shown in FIGURE 24. However, if adsorption data is not available, predetermined relationships between gas content, coal rank (which is related to fixed carbon) and depth can be used (FIGURE 25). If the natural moisture content, percentage volatiles and ash content are known, they can also be used (41). In addition if the geothermal gradient and pressure gradient for the area are known, temperature and pressure measurements are not necessary.

The volume of gas adsorbed by coal 'Q' is given by

$$Q = k_0 P^{n_0 - bT}$$

where $b = 0,14 \text{ cm}^3/\text{g}^\circ\text{C}$

$$k_0 = 0,8 \frac{FC}{VM} + 5,6 \quad (\text{FIGURE 26})$$

$$\begin{aligned} n_0 &= 0,315 - 0,01 \frac{FC}{VM} \\ &= 0,39 - 0,13 k_0 \end{aligned}$$

k_0 - constant ($\text{cm}^3/\text{g atm}$) related to rank

n_0 - constant 0,1 to 0,38 related to rank

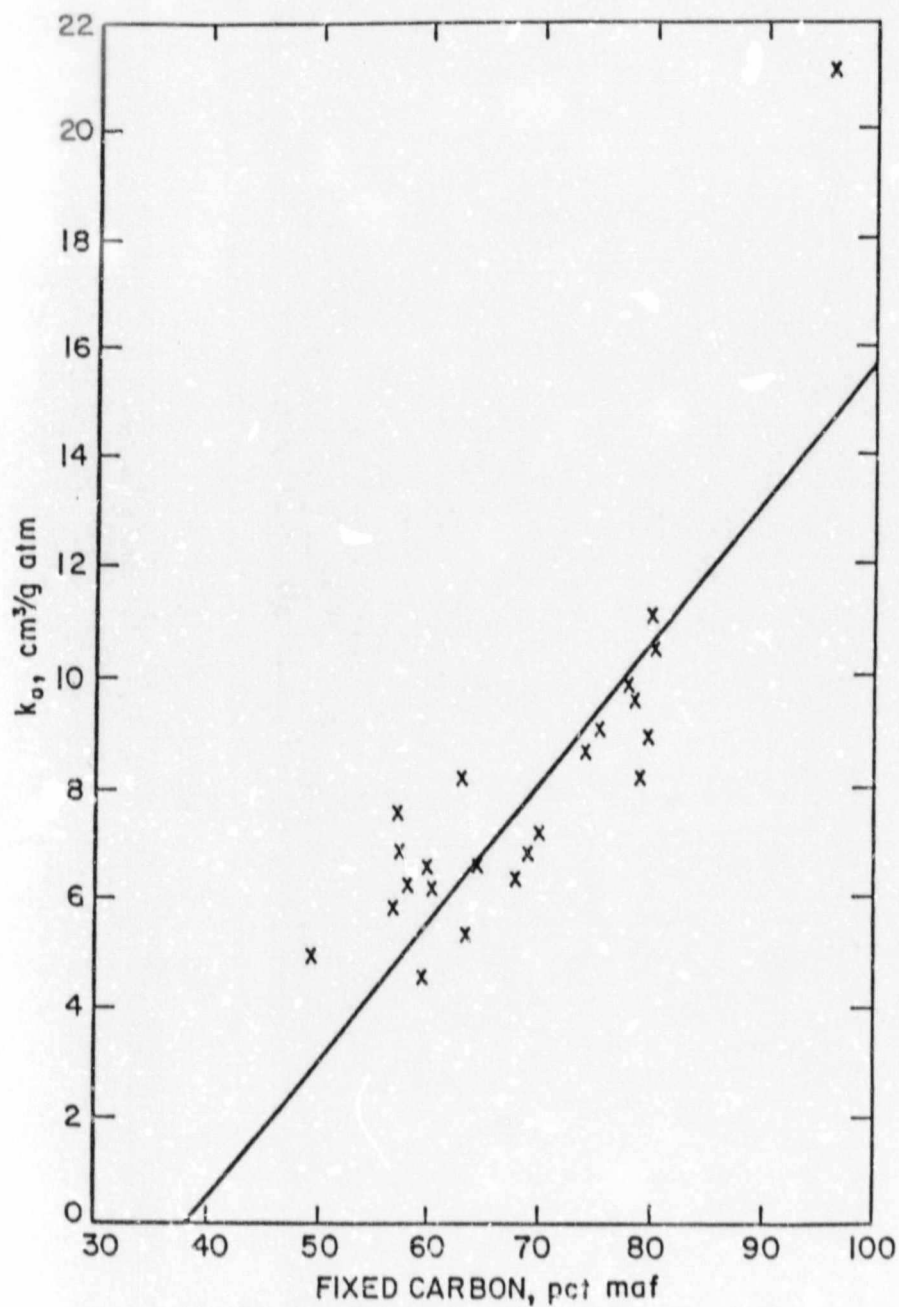


FIGURE 25. - Relationship of k_a to fixed carbon (41).

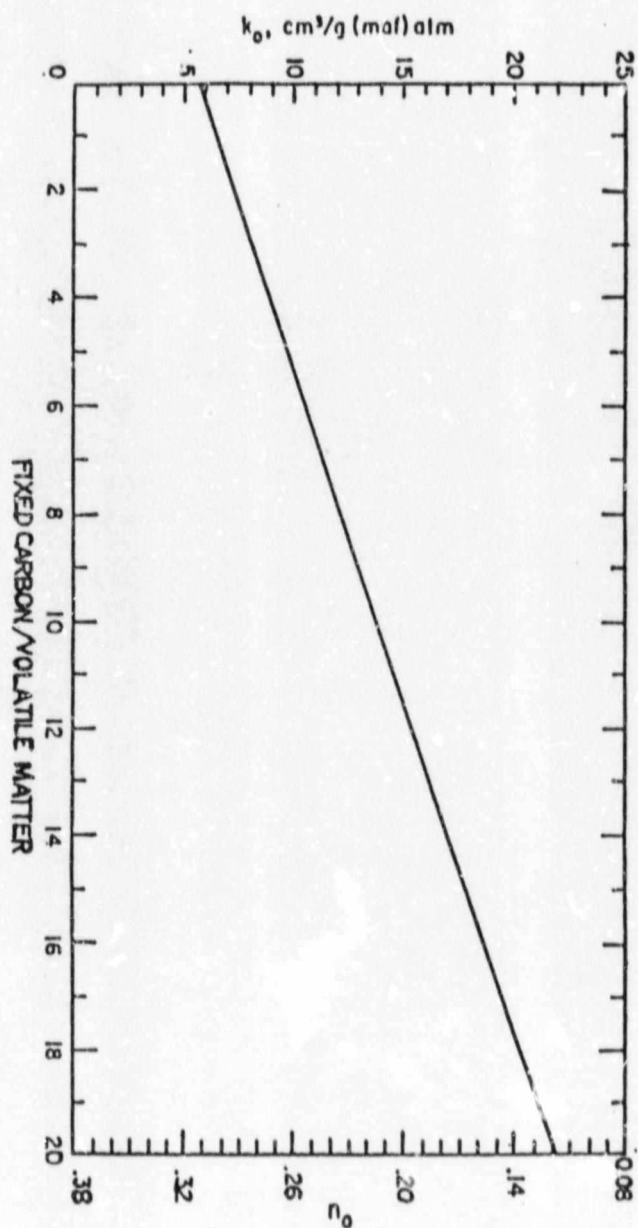


FIGURE 26 Value of adsorption constants k_0 and n_0 versus the ratio of fixed carbon to volatile matter (40).

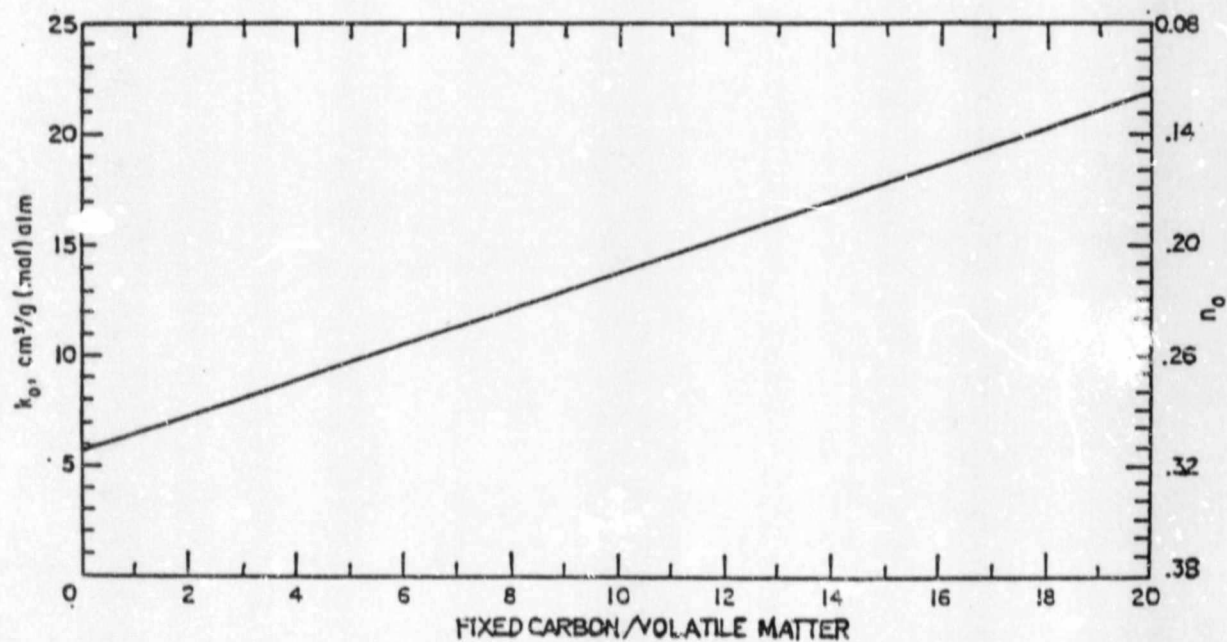


FIGURE 26 Value of adsorption constants k_0 and n_0 versus the ratio of fixed carbon to volatile matter (40).

and pressure 'P'(atm) and temperature 'T' (°C) are functions of depth 'h' (metres).

$$\text{Phydrostatic} = 0,096h$$

Usually seam pressure is less than hydrostatic pressure and Phydrostatic is the upper limiting case. In the U.S. the function normally used is:

$$P = 0,063 \times h.$$

$$T = 1,8 \frac{h}{100} + 11$$

1,8 - average geothermal gradient in the U.S.

11 = average surface ground temperature (°C) in the U.S.

Combining the above gives:-

$$Q = (100 - \% \text{ moisture} - \% \text{ ash}) \times \frac{Q_w}{Q_d}$$

$$\left[\frac{k_0 - b(1,8 \frac{h}{100} + 11)}{100} \right]$$

where the ratio of gas adsorbed for moist coal gas adsorbed for dry coal is given by:-

$$\frac{Q_w}{Q_d} = \frac{1}{(C_{oxm} + 1)}$$

$$= A_m + 1$$

C_0 - constant

m - moisture content (wt %)

A - constant = 0,23

The above ratio generally increases with rank from 0,55 - 0,85 with an average of 0,73 but an average of 0,75 is generally used. To obtain this ratio accurately, isotherms must be determined for various moisture contents.

When compared with results achieved in the U.S., Belgium, France and Germany, by the direct method, the indirect method yielded 3,5 % greater values. The two results were within $\pm 30\%$ but more usually within $\pm 20\%$ of each other.

Other indirect methods are:-

- i) Continuously logging of gas emissions during surface drilling operations and then plotting gas content at intervals throughout the geological sequence, but this is laborious and expensive.
- ii) Taking samples from the sidewalls and using a relationship between the gas content of these samples and in-seam gas content (FIGURE 27) (129).

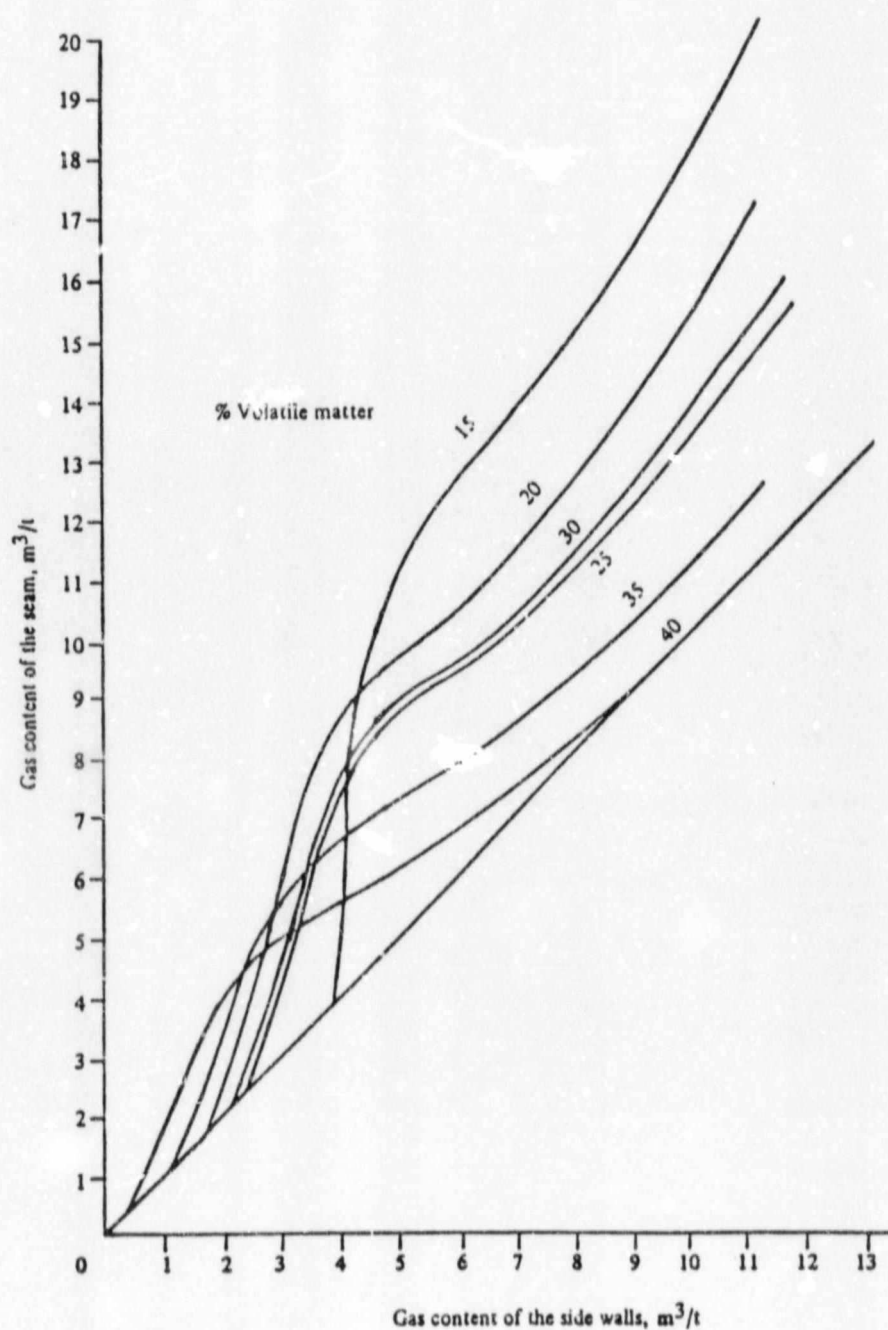


FIGURE 27 Correlation between gas content in the coal of side walls and in the virgin coal seams in Poland (14).

TABLE 7

ADVANTAGES	DISADVANTAGES
<u>DIRECT METHOD</u>	
(i) If it is done in conjunction with normal exploratory drilling it can be more convenient and less expensive than the indirect method, if this method has to take pressure and temperature measurements into account.	(i) The results are only accurate for that particular coal in that area and not for the coal bed in general, especially in areas of the seam exists at moderate differences in depth.
(ii) Where workings already exist results are more easily and rapidly obtained by the direct method.	(ii) It requires drilling of a hole to acquire a sample.

ADVANTAGES	DISADVANTAGES
<u>INDIRECT METHOD</u>	
(i) More precise results can be achieved for distant seams which can only be reached by a borehole (70).	(i) This method has only been proven for coals of rank exceeding sub-bituminous grading.
(ii) More general estimates can be made for the whole seam in an area, rather than relying on the precise result which only relates to that part of the seam tested.	(ii) It has not been conclusively proved that the laboratory derived equilibrium adsorption isotherms hold true for the coal underground.

2.4 Methane Release and Flow

The desorbable gas content ($Q_1 + Q_2$ using the direct method or Q using the indirect method) from a worked seam can be released into the mine ventilation by:

- i) Migrating through the seam to the coal face,
- ii) Migration through adjacent strata to the relaxed zone behind the advancing face,

iii) Desorbing from cut coal,

iv) Desorbing whilst being transported out of the workings.

There is a considerable pressure difference between the workings at atmospheric pressure and the higher gas pressure in the seam and adjacent strata. The adsorption equilibrium is disturbed and methane is released into pores and fissures. Some fissures are inherent in virgin strata, while others have resulted from fracturing of the strata due to abutment stresses and later relaxation of the strata (22). The methane desorption front moves slightly ahead of the face (10,43,54,58), but there can be a significant lag between mining operations and the resulting fall of pressure (69).

The mechanisms for this procedure is a two step process (64). The diffusion step is described by Fick's Law (64,125) which states:-

$$\frac{Q_e}{Q_\infty} = 1 - \frac{6}{\pi^2} \sum_{n=1}^{\infty} \frac{1}{n^2} e^{-D\pi^2 n^2 t/R^2}$$

$$= 1 - a e^{-\lambda t}$$

Q2 - quantity of gas emitted

Q - initial gas content = Q1 + Q2 + Q3

a - constant = 0,6

- constant = $D \pi^2 / R^2 = 0,006 - 0,010$

R - radius of sphere (particles considered as spheres)

D - coefficient of diffusion = $1,9 - 3,2 \times 10^6 \text{ cm}^2/\text{sec}$

For time 't' small -

$$\frac{Q_e}{Q_\infty} = \frac{6\sqrt{D}}{R^2} \left[\frac{1}{\sqrt{\pi}} \sum_{n=1}^{\infty} n \operatorname{erf} C \frac{1}{D t} \right] - \frac{3 D t}{R^2}$$

$$\begin{aligned} & 2 \frac{s}{v} \sqrt{\frac{D t}{\pi}} \\ & = G \sqrt{t} \end{aligned}$$

s - external surface area of particles

v - volume of particles

G - constant

For most coals studied the equation for 't' fairly large has been found to be more suitable. The larger the coal lumps the slower the diffusion process.

The second step is laminar flow within fissures in accordance with Darcy's flow equation (64) which states:-

$$\text{The rate of gas flow } \frac{dq}{dt} = \frac{K \Delta P}{\mu} \frac{A}{\Delta L}$$

K - permeability of the solid

μ - dynamic viscosity of the gas

ΔP - pressure differential

A - cross sectional area

ΔL - distance over which pressure drops

This equation requires the fluid flow to be viscous with the fluid adhering to the walls of the fissures. With gas flow this does not happen and slip occurs along the fissure walls giving rise to a greater flow and an apparent dependency of permeability on gas pressure. This is known as the Klinkenberg effect (58,74) which states that the relationship between

apparent permeability, where slip occurs (k_a) and Darcy's permeability is given as:-

$$k_a = k \left(1 + \frac{b}{p} \right)$$

$$= k \left(1 + \frac{4 c \gamma}{r} \right)$$

- b - constant
- c - constant
- r - pore radius
- γ - mean free path of gas

However for higher flowing pressures (ΔP) and the permeability (k) below 20 md, the Klinkenberg effect disappears.

The rate of emission is determined by either the first diffusion step or the second laminar flow-step, whichever provides the slower rate. That slower step is the rate determining step (RDS) (114). More methane is released per unit drop in pressure if the original virgin pressure was in the 0,1 - 4 MPa range than if the virgin pressure was above 4 MPa.

2.5 Porosity and Permeability

Porosity describes the storage capacity of methane in the free gas state in pores of coal. These pores take up 3-10% of the volume of coal (64) and can be sub-divided into two systems:-

- i) Micro-pores - capable of being penetrated by helium

- ii) Macro-pores - capable of being penetrated by mercury

Coal structure can be modelled as flat poly- condensed aromatic lamellae (33).

- i) Low-rank coals (< 85% carbon (C)), have randomly connected lamellae connected by cross-links. This coal is highly porous.
- ii) Bituminous coals (85-91% C) have some lamellae orientation but little cross-linking. Pores are practically absent and this coincides with a minimum porosity at 89% C.
- iii) High-rank coals (> 91% C) have no cross- links but orientation has increased and so porosity increases again.

For any type of coal porosity is relatively stable (112) and little affected by confining pressure.

Permeability describes the methane conductivity of coal. It is a principal factor relating to gas emission and again is subdivided into:-

- i) Micro-permeability - permeability of micro-pores. Methane flow is by diffusion.
- ii) Macro-permeability - permeability of fissures (also called macro-pores). This provides the main path for emission by laminar flow.

Permeability is mainly dependent on the number and width of fissures and their continuity in the direction of flow (74). The density of inherent fissures or cleats is dependent on coal rank (60) and overburden

pressure (14). These cleats are subdivided into:-

- i) Primary or face cleats. Coal is more permeable in the direction of primary cleats than secondary cleats.
- ii) Secondary or butt cleats. Typically these are roughly at 90° to the primary cleats.

High fusinite contents in coal cause the coal to be more friable and hence have more persistent bedding fractures (cleats) and therefore increased permeability (14). Coal full of connate water is virtually impermeable (132).

As gas is emitted, coal contracts 0,2 - 0,5%, and the number and width of fissures increases (64). Relaxation of strata has the same effect (104,43,75,113) but to a far larger degree (up to 10%) (89). Initially permeability decreases by more than 3 orders of magnitude (21) in the abutment zone since the increasing stress closes the pores, causing a gas barrier ahead of the coal face (80,89,108,121) (FIGURES 28,29). Further increasing stress propagates fractures and increases permeability, but this happens only after the gas barrier has formed in front of this zone. Permeability of coal shows a stress hysteresis effect (74) so even if recompression following fracturing again reduces permeability, it will still be greater than that of the virgin seam. Permeabilities range from 1 darcy to 1×10^{-9} darcy (21).

Methods of measuring permeability of coal are:-

- i) Placing a core in a triaxial loading cell and passing gas which is under pressure through it (FIGURE 30). Although accurate, results obtained

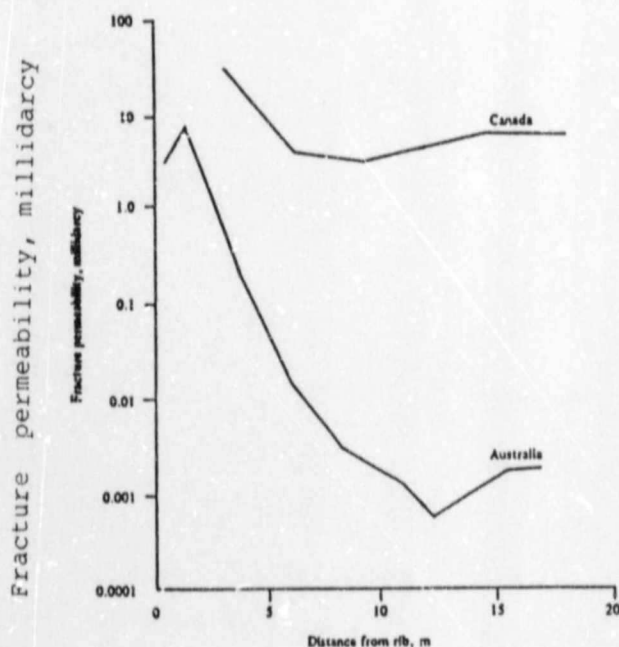


FIGURE 28 Example of curves of fracture permeability versus distance from the rib (14)

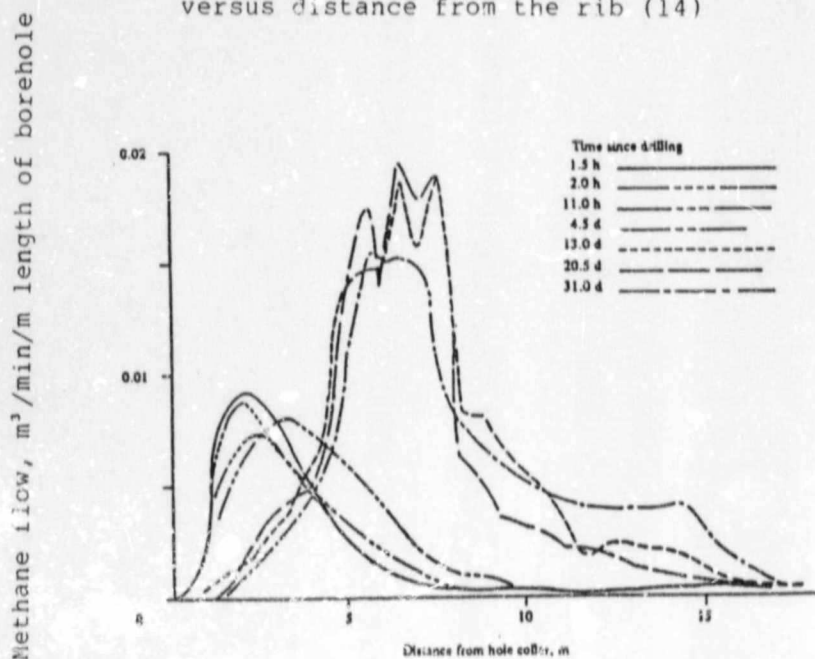
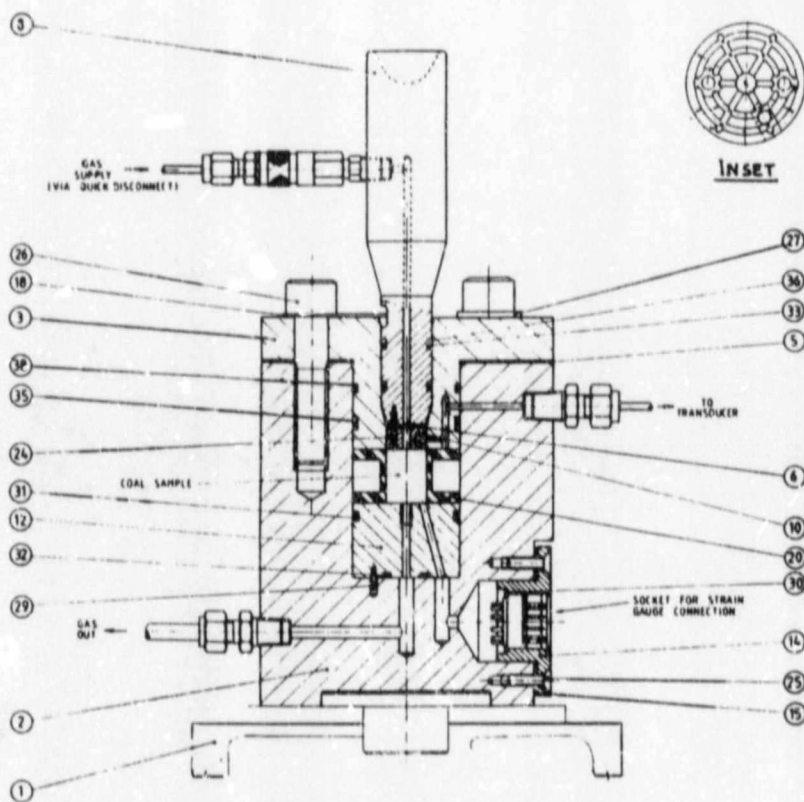


FIGURE 29 Example of incremental flow measurements to a 50mm borehole in a coal seam ahead of a coalface (51).



SECTIONAL VIEW SHOWING GAS LINE

TYPE 1 TEST CELL

Test cell design with (inset) details of gas distribution channels in upper and lower platens: 1, channel, main frame; 2, cell body; 3, main retaining flange; 5, 6, gaskets; 8, plunger; 10, upper gas distribution platen; 12, lower gas distribution platen; 14, 12-pin electrical contact port; 15, gasket; 18, keeper plate for plunger; 20, polyurethane rubber sheath; 24, platen retaining screws; 25, 12-pin port retaining screws; 26, main retaining flange bolts; 27, washers; 29, locating pin; 30, sealing ring; 31-38, O-ring seals with Parbaks

FIGURE 30 (48)

are suspect because core drilling induces fractures. Also the core's permeability may be orders of magnitude less than the permeability of the in-situ strata as a whole (48).

The cores must not be evacuated prior to testing as this causes a reduction in permeability (28).

- ii) Drilling two holes and injecting one hole with methane and a tracer gas (carbon 14 or sulphur hexafluoride) and receiving some of it at the other hole (57). This method is liable to errors caused by gas leakage (14).
- iii) Placing a camera in a borehole and photographing the hole walls. The photographs are then computer analysed for permeability.
- iv) Sonic velocity testing has been tried but attempts to correlate sonic behaviour with permeability have generally been unsuccessful (139).

2.6 Geological and Mining Factors Influencing Emission

2.6.1 Geological Factors

Large differences in gas content in the same seam have been measured on either side of a geological disturbance (72) especially large throw faults (more than 1,5 m). Where gas has to flow up dip through strata containing water, the water may also retard methane flow (94). Minor geological changes such as rolls, bad roof and a thinning in the coal seam generally increase gassiness. Big variations such as a

are suspect because core drilling induces fractures. Also the core's permeability may be orders of magnitude less than the permeability of the in-situ strata as a whole (48).

The cores must not be evacuated prior to testing as this causes a reduction in permeability (28).

- ii) Drilling two holes and injecting one hole with methane and a tracer gas (carbon 14 or sulphur hexafluoride) and receiving some of it at the other hole (57). This method is liable to errors caused by gas leakage (14).
- iii) Placing a camera in a borehole and photographing the hole walls. The photographs are then computer analysed for permeability.
- iv) Sonic velocity testing has been tried but attempts to correlate sonic behaviour with permeability have generally been unsuccessful (139).

2.6 Geological and Mining Factors Influencing Emission

2.6.1 Geological Factors

Large differences in gas content in the same seam have been measured on either side of a geological disturbance (72) especially large throw faults (more than 1,5 m). Where gas has to flow up dip through strata containing water, the water may also retard methane flow (94). Minor geological changes such as rolls, bad roof and a thinning in the coal seam generally increase gassiness. Big variations such as a

change in gradient can change the gas emission pattern. Gas is known to accumulate in anticlines and domes and their related synclines and valleys are less gassy. Pockets of gas occur where coal has been pulverised by tectonic activity. This can lead to outbursts where other conditions such as depth are conducive (57,118).

The porosity, permeability and strength of adjacent strata are important factors as a considerable amount of emission comes from adjacent strata. For instance a porous bed which is also permeable could release the gas it contains into the workings, whereas an impermeable strata in the roof or floor which is strong and does not readily fracture, forms a barrier to gas entering workings (108).

Geological anomalies can be determined before mining by the Radar Imaging Method (RIM) (96). These surveys can be done from surface by lowering a transmitter down one hole and a receiver down another, within 500 m of the first hole. By repeating this process at regular intervals a picture of conditions in the area can be built up.

An electromagnetic wave is passed through the coal and if some intrusion is encountered the receiver will pick up a weaker signal. The signals are analysed by computer and a graphical depiction of the seam is made. Once mining has started, the transmitter can be placed at the headgate and the receiver at the tailgate.

2.6.2 Mining Factors

The proximity of previous workings which by now have become partially drained, may reduce gas flow into new workings. In this way, planning of extraction of seams

can assist in methane control.

In gassy conditions, single heading development may be impractical because of excessive concentrations of methane in the ventilation. A twin entry system suffers from methane pickup from the intake ribside before the air reaches the face. If three or more headings are used, the inside entries are bordered by partially or completely predrained pillars. They can be used for the intake, so reducing methane levels at the face. The outside entries are then used for returns.

To provide increased volumes of air to the face, use Y, H or W rather than U or Z ventilation systems, (FIGURE 31) although the former three systems require at least one additional gateroad to the main intake or main return airways (62,117).

Another factor is the type of ventilation fan employed. Waste areas are pressurised on starting a force fan and become depressurised when it is stopped. Large volumes of accumulated waste gas then flood into the workings. Therefore no mine with a methane problem should be force ventilated although the exhaust fans should be equipped with reversal arrangements. In an emergency modern radial flow fans or axial flow fans can respectively produce 90% or 40% of normal mine ventilation when reversed (102).

1) Advancing Longwall Mining

Generally a little gas enters the ventilation along the intake ribside, more gas along the face and a little along the return ribside. The pressure drop from the intake to the return causes leakage across the goaf and this flushes the goaf. Flushing prevents methane accumulations in the

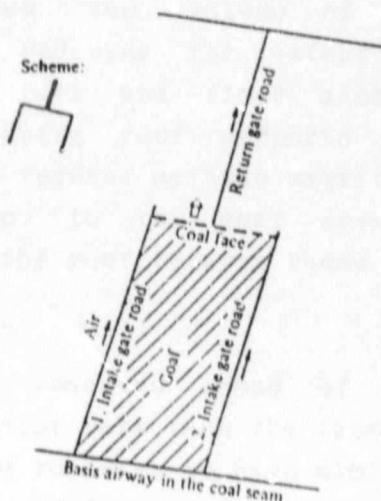
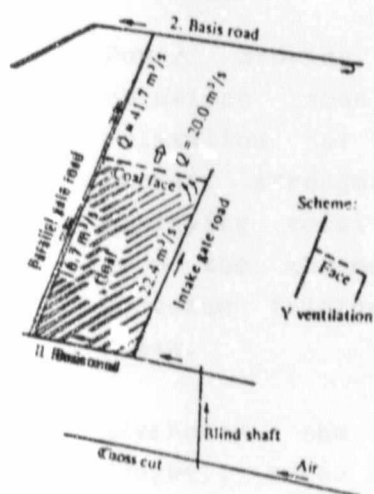
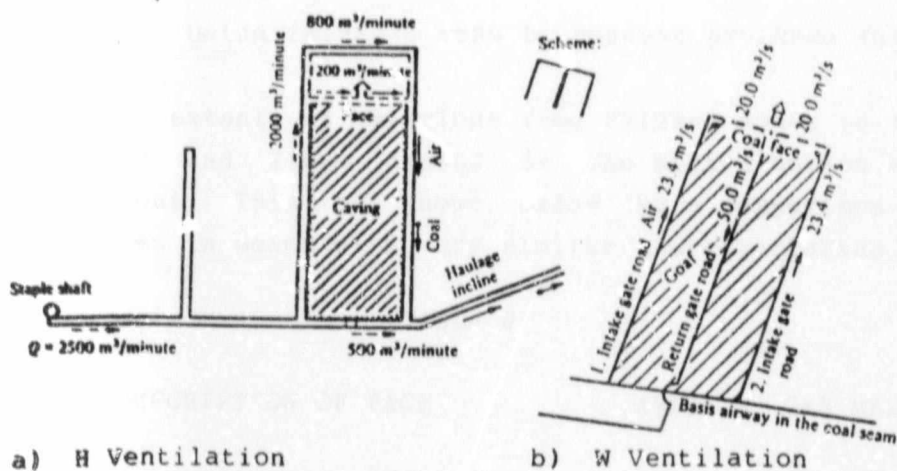


FIGURE 31 European Designation of Ventilation Systems (82)

goaf being released when barometric pressure falls (72).

The extent of emissions from relaxed zones in the roof and floor depend on the mining method and layout. This is shown below by a comparison of faces in seams with very similar characteristics.

TABLE 8

DESCRIPTION OF FACE	AVERAGE GAS MAKE
Caved face with little or no overworking	29,1 m ³ /t
Heavily overworked caved face	20,1 m ³ /t
Power stowed face with little or no overworking	12,4 m ³ /t

Power stowing reduces the volume of the emissions zone, by reducing the extent of relaxation of the roof and floor arches. Larger, stronger advancing roof supports and employing total caving reduces methane migration from the abutment zone to the goaf area by reducing fractures in the roof between these two areas.

Advancing the gate roadways ahead of the longwall faces at a faster rate than the face is advancing, causes these roadways to have similar characteristics and problems experienced in development roadways. Also if successive adjoining longwalls are formed in such a way that one gate road is against virgin coal and the other against an old goaf, emission control problems will increase.

ii) Retreating Longwall Mining

Conditions are similar as for (i) with the exception that the panel to be mined is partially predrained by the pre-developed gate roadways. However gas in the goaf can accumulate and migrate to the face because it is not flushed by leakage.

iii) Board and Pillar Mining

Tests in the US indicate that for longwalls, relaxation from differential subsidence following mining, extends up to 60 m in the roof. For bord and pillar mining the extent of relaxation is much less. High permeability coals (such as in the US) allow methane from ahead of the face to predrain into the workings and roadways. If large panels are delineated roadway developments the methane ahead will drain into these roadways before the main extraction of the coal takes place. This almost eliminates emissions into workings provided the bord and pillar method does not subsequently also allow caving. Caving would allow emission from adjacent strata which has not yet been disturbed, to desorb considerable quantities of gas into the workings.

iv) Pillar Extraction

The pillars are already predrained but most of the methane comes from caving in the goaf.

v) Shortwall Mining

Shortwall panels are partially predrained during development and again methane mainly

comes via the collapsed goaf.

Methane emission is related to production and advance rate. As the face advances, fracturing of the strata and gas flow through these fractures, takes place. Depending on the rock type, this fracturing and emissions may lag behind production. If there is no advance, the strata will still emit gas called the inert component (Y intercept). The energised component is the increase in emission per unit of production (slope)(FIGURES 32,33).

Specific emission (m^3/t) decreases as output increases because there is less time for the in-situ gas to desorb before it is mined. As output increases, a higher percentage of the desorbable methane ($Q_1 + Q_2$) is emitted when the coal is already cut (32). This only amounts to 8-15% of total emission. This increase is not particularly significant because the rest has already desorbed through fractures or is still retained in the coal at this stage.

For the same reason, although continuous miners produce more gas from broken coal than conventional mining, the increase is negligible. Notwithstanding, emission peaks from continuous miners are significant. By reducing the depth of web, these peaks are not as pronounced because the machine will be cutting for a longer period at a lower rate for similar total output.

It is not certain whether maximum output with minimum specific emission is best achieved by increasing face length or rate of advance. Local knowledge of a particular seam is necessary because in the UK short faces with quick advance are preferable (32) whereas in Germany emission rate (m^3/min) correlates linearly with production (t/day) and not very well with face

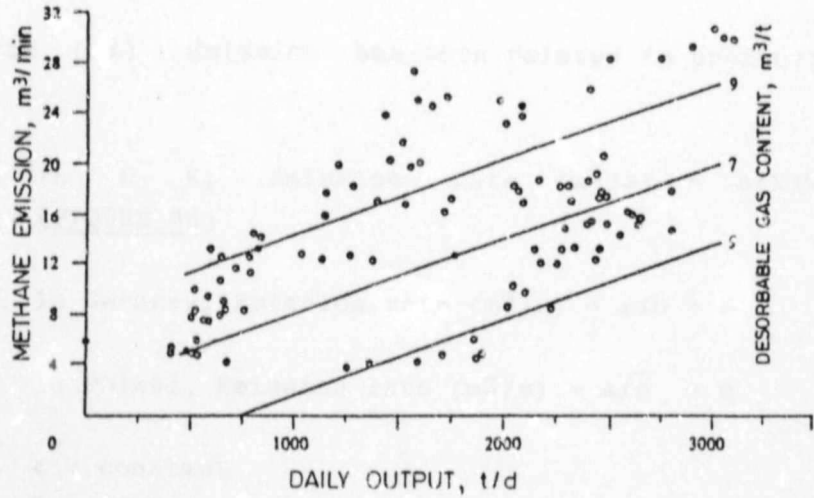


FIGURE 32 Total gas emission plotted against output and desorbable gas content. The straight lines denote weekly averages (11)

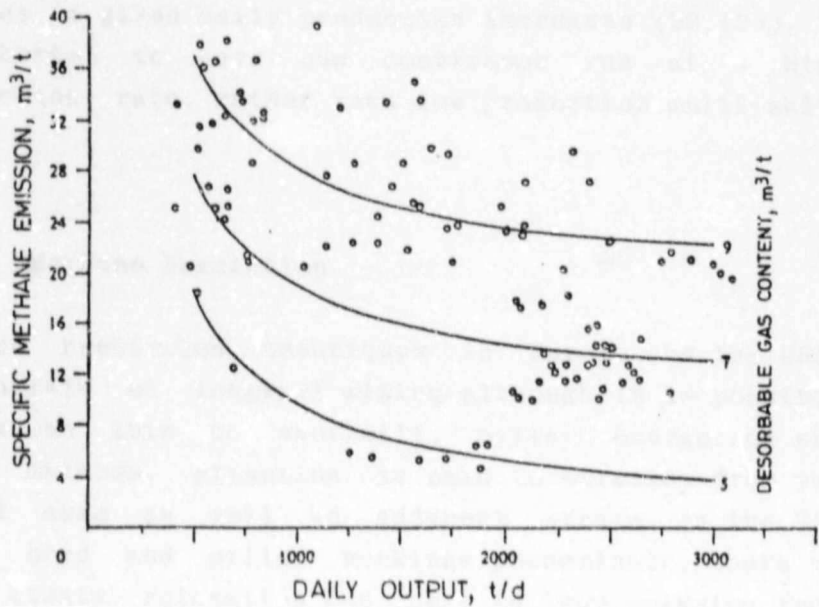


FIGURE 33 Specific gas emission plotted against output and desorbable gas content. The trend curves denote weekly averages (11)

advance (14). Emission has been related to production by:

(i) In U K, Emission rate (m^3/s) = $a(X)^{0,8}$
(FIGURE 33)

(ii) In Germany, Emission rate (m^3/s) = $axo + b$

(iii) In Poland, Emission rate (m^3/s) = $a\sqrt{o} + b$

a - constant

X - advance rate (m/d)

o - production (t/d)

b - constant

Gas make increases if the number of shifts required to produce a given daily production increases (32,123). It is better to have one continuous run at a high production rate, rather than low production multi-shift runs.

2.7 Methane Prediction

Methane prediction techniques in Europe and the USSR concentrate on longwall mining although it is possible to extend this to shortwall, pillar extraction and other methods. Attention is paid to emission from the worked seam as well as adjacent strata. In the USA where bord and pillar workings predominate, there is less strata relaxation and there is less emission from adjacent strata. The main concern is with emission from the worked seam.

The parameters involved are:-

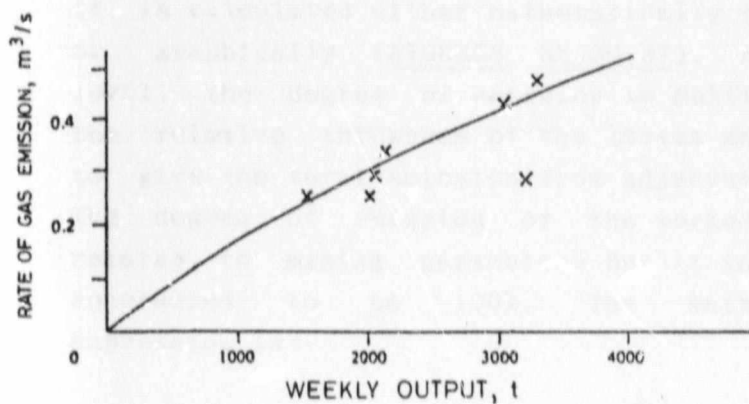
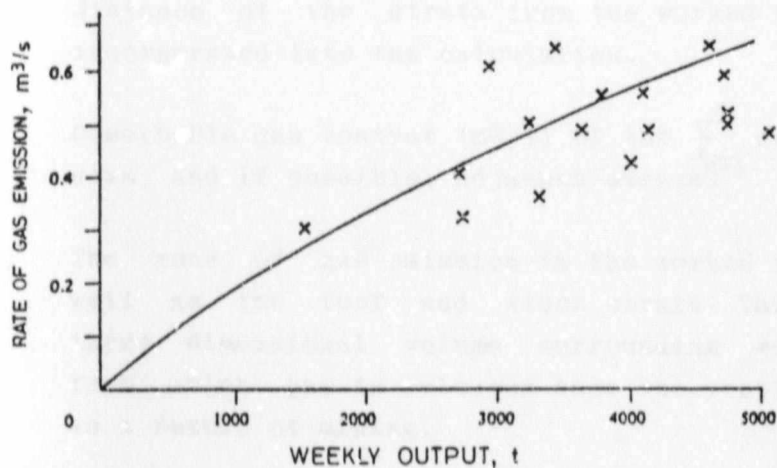


FIGURE 34 Gas emission from two UK longwall faces (1) compared with curves of the form:
Methane emission = constant $\times$ (advance rate)^{0.8}

- i) Stratigraphy above and below the worked seam. Shale and sandstone are considered to have a percentage of coal seam gas content. The distance of the strata from the worked seam is incorporated into the calculation.
- ii) Desorbable gas content (m^3/t) of the $\sum_{n=1}^n$ worked seam, and if possible, adjacent strata.
- iii) The zone of gas emission in the worked seam as well as the roof and floor strata. This is a three dimensional volume surrounding workings from which gas is released into the ventilation as a result of mining.
- iv) The degree of gas emission from the worked seam and adjacent strata. This is the percentage of gas present at a specific level which flows into the ventilation.

It is calculated either mathematically (TABLE 9) or graphically (FIGURES 35,36,37). For each level, the degree of emission is multiplied by the relative thickness of the strata and summed to give the total emission from adjacent strata. The degree of emission of the worked seam is related to mining parameters but it is usually considered to be 100%. The mathematical expression is:-

$$\text{Desorbable gas content } Q \text{ (m}^3\text{/t)} = \frac{\sum_{n=1}^n 100\% \times (Q_1 + Q_2) \text{worked seam} \times X_n \times Q_{\text{adjacent strata}}}{t \text{ worked seam}}$$

X_n = degree of gas emission (%)

t_n = thickness of the strata (m).

COUNTRY	METHOD	HEIGHT OF ZONE OF GAS EMISSION m	DEGREE OF GAS EMISSION %
BELGIUM	BAEX	$h_R = x + d$ $h_F = -50$	$A_R = 100\%$ to $h = x$, then linear decrease to 0% at $h = x + d$ $A_F = 100\%$ to $h = -20$, then linear decrease to 0% at $h = -50$
FRANCE	GRUTHIER, 1965 - 1977	$h_R = +100$ $h_F = -100$	$A_R = 100\%$ $A_F = 100\%$
FRANCE	JEGHER, 1978 on	$h_R = +170$ $h_F = -55$	$A_R = 90\%$ to $h = +30$, 80% from $h = +20$ to $+120$ i.e. linear decrease to 0% at $h = +170$ $A_F = 90\%$ to $h = -15$, then linear decrease to 0% at $h = -55$
FRG	FLÜGGE	$h_R = 0.5 l \tan \alpha$ $h_F = 0.5 l \tan \alpha$ ($\alpha \leq 25^\circ$ gon)	* $A_H = 100 - \frac{200h}{l}$ cut α $A_F = 100 + \frac{200h}{l}$ cut α ($\alpha \leq 25^\circ$ gon)
FRG	SCHULTZ	$h_R = 0.5 l$ $h_F = -20$	* $A_R = \frac{100}{\sqrt{1 + \frac{h^2}{l^2}}} = 4h^2$ $A_F = 100\%$ to $h = -5$, then linear decrease to 0% at $h = -20$
FRG	WINTER	h_R fixed by $A_R = 10\%$ h_F fixed by $A_F = 10\%$	$A_R = 100\%$ to $h = +20$, then $A_R = 100 \exp \{-\mu(h - 20)\}$ $A_F = 100\%$ to $h = -8$, then $A_F = 100 \exp \{-\mu(h + 8)\}$
FRG	KOPPE	$h_F = -50$ to -64 (depending on distance from face centre line, fig. 22)	$A_F = 142 + 4h + 0.027h^2$
POLAND	BARBARA MINE	$h_H = 120$ h/t $h_F = 120$ h/t	$A_R = 64.7 \exp \{-0.03957h/t\}$ $A_F = 54.1 \exp \{0.03721h/t\}$
UK	NCB/MRDE	unlimited in theory, in practice $h_R = 200$ $h_F = -100$	see figure 24
USSR	LIDIN	$h_R = H \cdot (1.2 + \cos \theta)$ $h_F = H \cdot (1.2 - \cos \theta)$	$A_R = 100 (1 - h/h_R)$ $A_F = 100 (1 - h/h_F)$

Table 9 Definitions of the zone and degree of gas emission for prediction methods.

Sign Convention: The working sram is taken as the zero datum of level.

Heights in the roof are positive (+) and depths in the floor negative (-).

* Assuming a rectangular prism as zone of gas emission.

Nomenclature for Table 3

All methods:—

h_R = maximum height of emission zone in the roof, m.

h_F = maximum depth of emission zone in the floor, m.

h = height or depth from worked seam, m.

Λ_R = degree of gas emission in the roof, %.

Λ_F = degree of gas emission in the floor, %.

Belgium:—

x = height in roof of 100% gas emission, m.

d = height in roof, above x , in which gas emission decreases to 0%, m.

FRG:—

l = face length, m.

α = angle of prism in Flügge's method, gon or degrees.

μ = attenuation co-efficient for Winter's method.

USSR & Poland:—

N = mining factor, e.g. $N = 60$ for full caving,
 $N = 45$ for stowing.

t = extracted seam thickness, m.

Θ = dip of strata, gon or degrees.

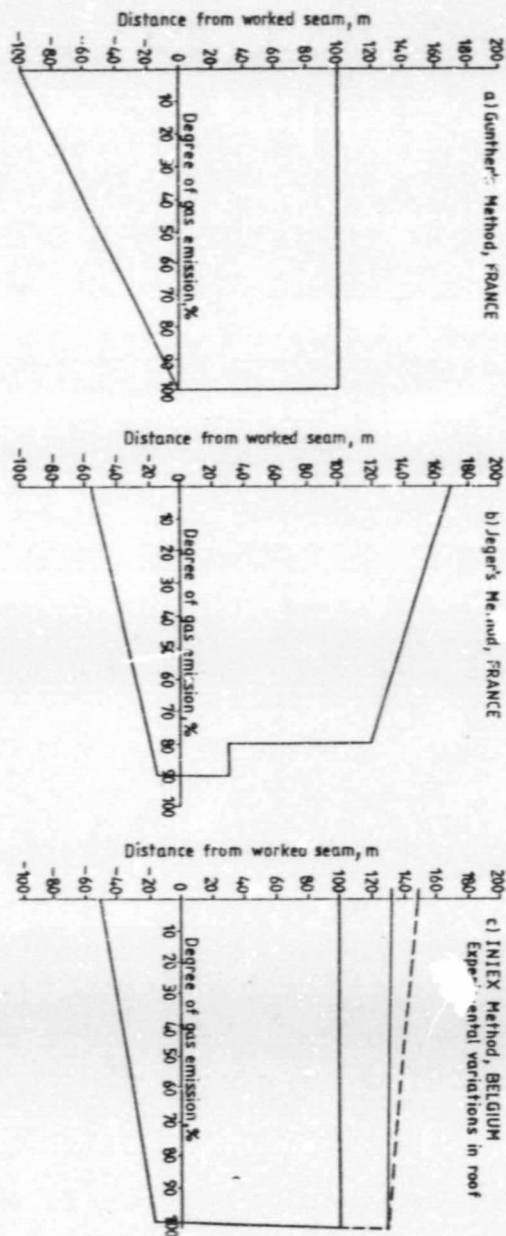


FIGURE 35

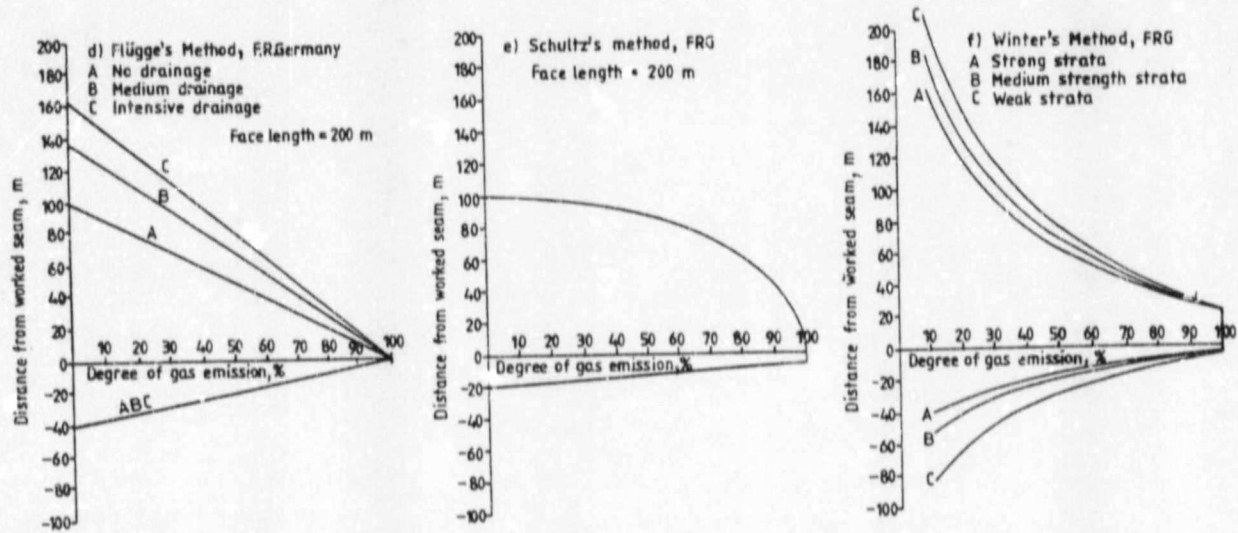


FIGURE 35

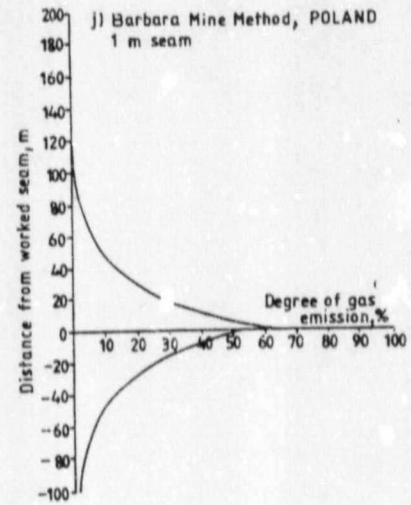
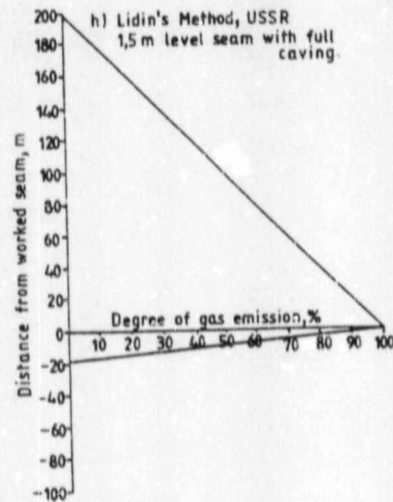
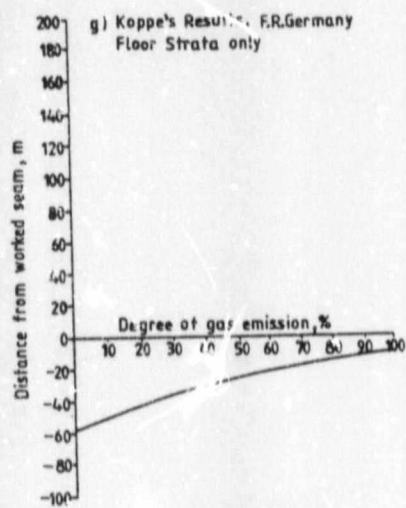


FIGURE 35

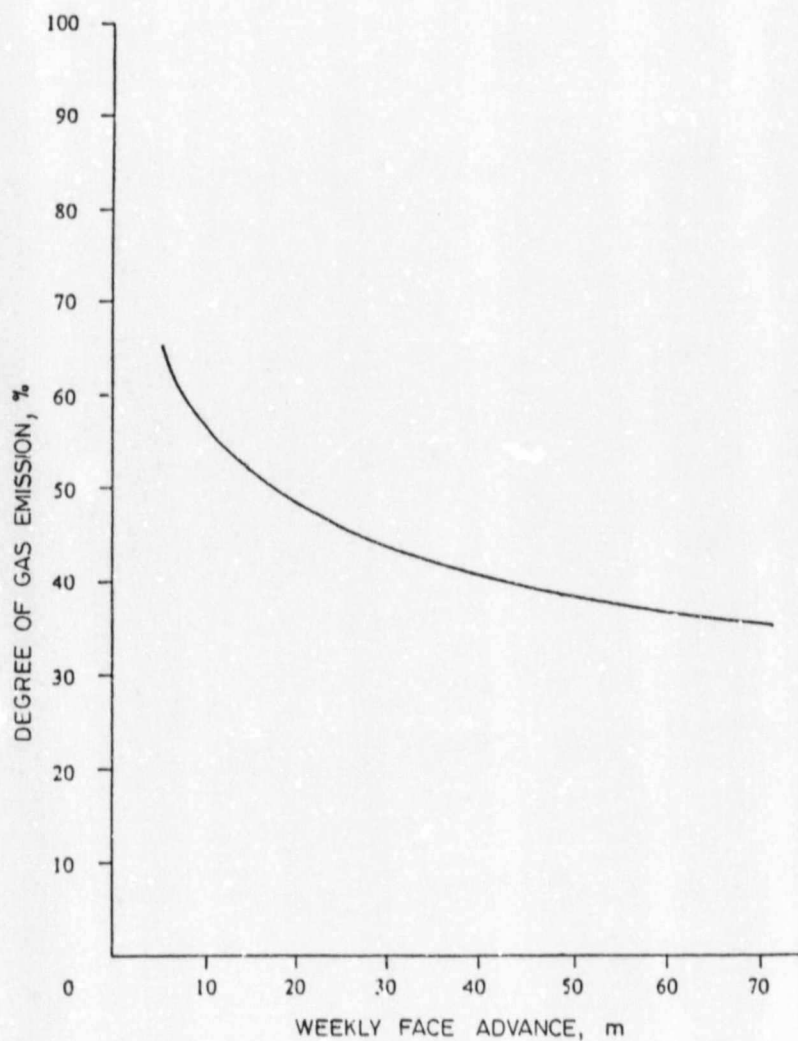
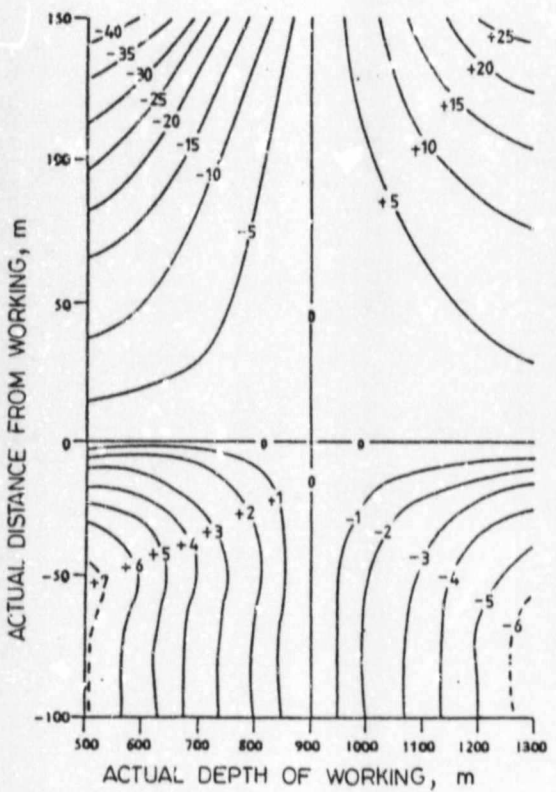
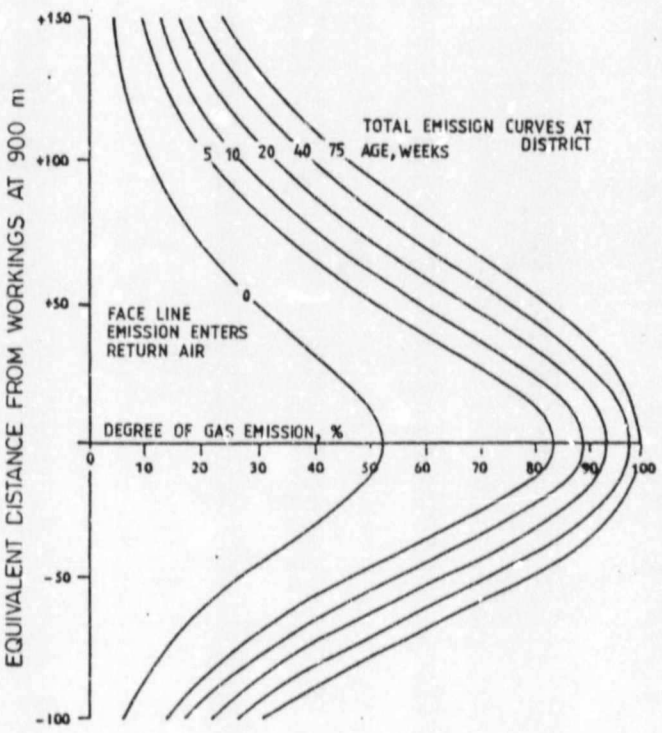


FIGURE 36 The MRDE curve for the degree of gas emission from the worked seam (14)

FIGURE 37
degree of gas emission curves for the MRDE
method (14)



a) Height correction before using 900 m curves



b) Degree of gas emission at 900 m

FIGURE 38 is a flow chart showing how these methods are applied in Europe. The USSR and Poland have, in addition, a second method using in-situ gas pressure. The USA is developing a computer prediction method based on principles of methane flow. Details of the methods developed in these countries are given below:-

2.7.1 France

CERCHAR previously used a simplified representation of the zone of emission. This was a parallelogram based on coal face length with the internal angles of the parallelogram determined by the horizontal inclination of the face. The degree of gas emission is shown in FIGURE 35a. 50% of the desorbable gas content was estimated to enter the ventilation (31,108).

An improved version now uses a revised geometrical configuration for the variation in degree of emission with respect to height (see FIGURE 35b). Non-carbonaceous shale is considered to have 1% and carbonaceous shale and sandstone 10% of the methane content an equivalent thickness of the coal. The gas content of the coal bed nearest each horizon is used or if this is not known, then the gas content of the worked seam is used. The total emission for the worked seam is calculated to be the desorbable gas content less 2 m³/t.

To calculate the required ventilation flow, a co-efficient of irregularity or peak factor has been adopted to account for sudden peaks above the mean emission level. This factor is 2,0 for the return end of the face and 1,5 for the outbye end of the return airway. FIGURE 39 shows how this same concept is used in the UK.

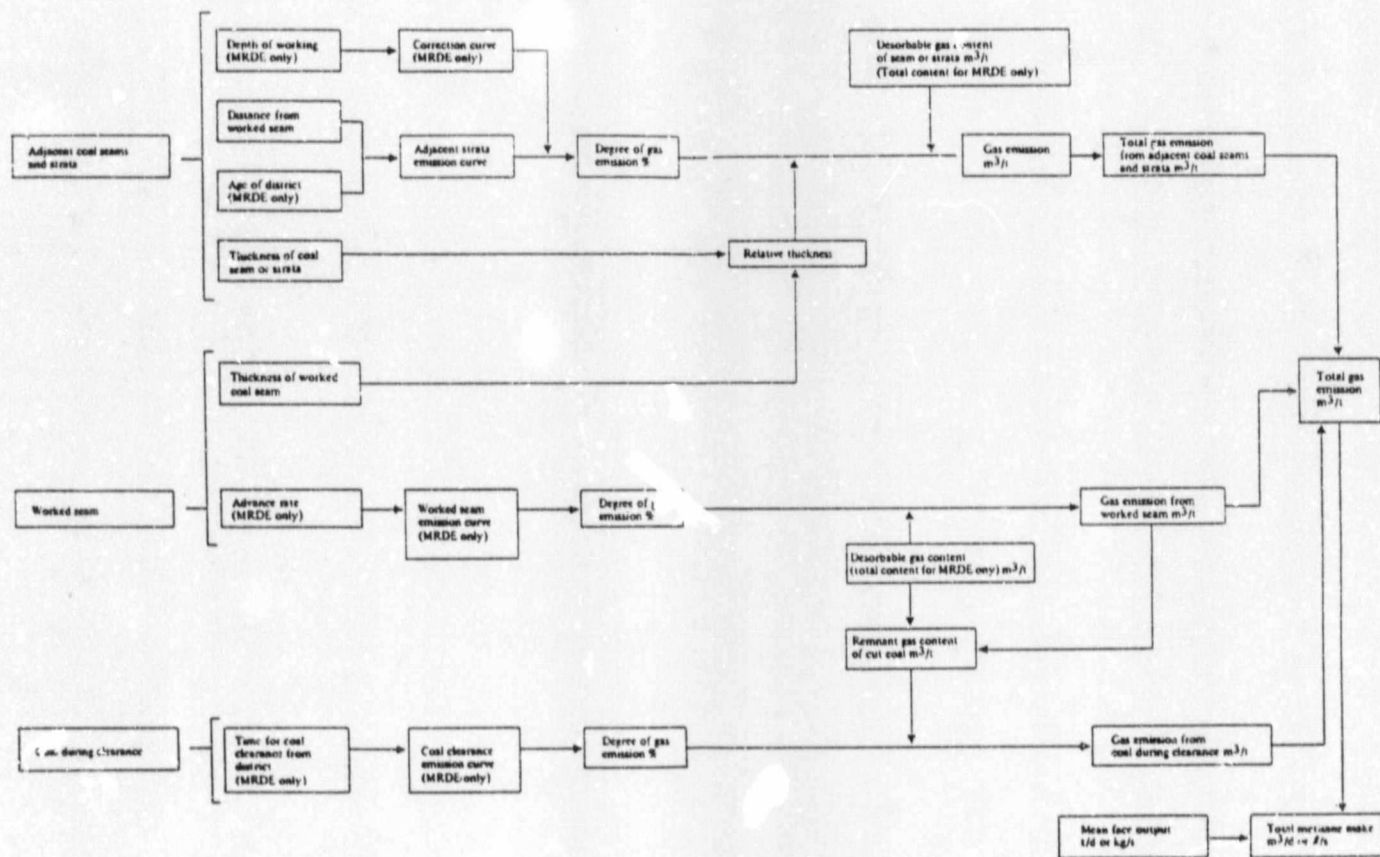


FIGURE 38 Flowchart of European Prediction Methods (14)

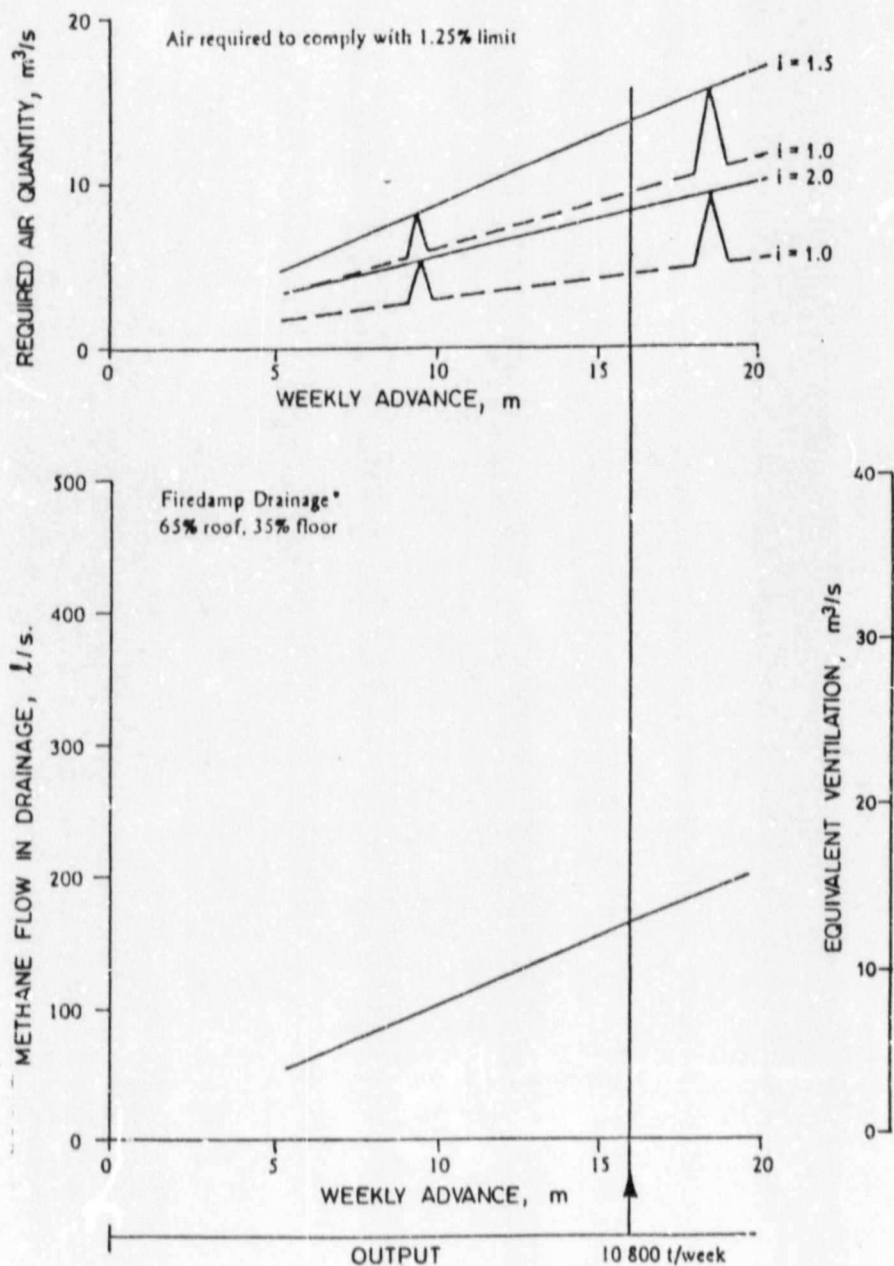


FIGURE 39 Example presentation of results from the MRDE prediction method (14)
 i = coefficient of irregularity
 * preferred term in the UK

Accuracy is estimated to be better than $\pm 20\%$.

2.7.2 Belgium

INIEX use a modified version of the CERCHAR method (8,135,136). Here methane content measurements are taken up to 200 m on plan from the workings to compensate for fluctuation with respect to area.

In the floor the degree of emission used is 100% for 0 - 20 m and 100% - 0% for 20 - 50 m below the worked seam. In the roof it is 100% to a height 'x' and 100% - 0% for a height 'd' ($0 < d < 25\text{m}$) (see FIGURE 35c).

Sandstone and shale contents are estimated in the same way as CERCHAR.

2.7.3 Federal Republic of Germany (FDR)

Three methods have been developed at Bergbau-Forschung GmbH by Flugge, Schultz and Winter. These methods consider perpendicular and not vertical distance between the worked seam and adjacent strata (except for method iv below) and are only accurate for gently sloping ($< 18^\circ$) seam.

- i) Flugge's method considers the zone of emission for longwall workings as triangular prisms superimposed on the workings (14). In the floor the internal angle is $22,5^\circ$ whereas for the roof this varies according to the intensity of methane drainage practiced (FIGURES 35d, 40a). The degree of emission is given by:-

Author Billenkamp Ernst Gottfried

Name of thesis Methane Desorption Characteristics Of Selected South African Coal Seams. 1988

PUBLISHER:

University of the Witwatersrand, Johannesburg

©2013

LEGAL NOTICES:

Copyright Notice: All materials on the University of the Witwatersrand, Johannesburg Library website are protected by South African copyright law and may not be distributed, transmitted, displayed, or otherwise published in any format, without the prior written permission of the copyright owner.

Disclaimer and Terms of Use: Provided that you maintain all copyright and other notices contained therein, you may download material (one machine readable copy and one print copy per page) for your personal and/or educational non-commercial use only.

The University of the Witwatersrand, Johannesburg, is not responsible for any errors or omissions and excludes any and all liability for any errors in or omissions from the information on the Library website.