METHANE DRAINAGE INVESTIGATIONS AS A METHOD OF CONTROL OF OUTBURSTS AT WEST CLIFF COLLIERY

By

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ABSTRACT

This paper describes investigations into drainage of gas from the solid to control outbursts. Initial studies conducted on drainage of shear zones showed that, given enough time, these can be successfully drained. Once drained, they can be intersected and driven through without any activity of outbursts. The results of programme of investigations to determine parameters for drainage from the solid show that gas flow rates from the Bulli Seam on the average are about 2 litres/min/metre length of borehole. But when shear zones are intersected, the flow rates could be as high as 8 litres/min/ metre length of hole. Shear zones can be picked up from pressure measurements 50 - 60 m away and this could give sufficient time to drill through these zones and drain them in advance. Systematic drainage of Bulli Seam at West Cliff Colliery could eliminate or reduce frequency of outbursts considerably. At present drainage holes are being drilled systematically in the seam, with drainage installations on the surface. Gas purity of 80% methane is being delivered at the surface plant.

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INTRODUCTION

West Cliff Colliery mines high quality coking coal from the Bulli Seam in the Southern Coal Field of N.S.W., at depths approaching 500 m. The principal feature of the mine is the presence of a large number of shear zones, which are centres of outbursts. Over 107 outbursts have been experienced at the Colliery, 101 of which have been associated with these shear zones. A general description of the phenomenon and associated studies have been presented in this Symposium by the authors in another paper (Marshall, Griffiths and Lama, 1980). The conclusions drawn from these studies are as follows :

1. Gas present in the coal seam is predominantly methane. Though carbon-dioxide occurs in the northern sections, the severity of the outbursts does not seem to be related to its presence.
2. All outbursts have occurred during development. Mining of pillars using the Wongawilli split and fender retreating system has resulted in no outbursts, though a number of pillars extracted were transgressed by shear zones.
3. At all places where shear zones are present, deterioration of roof occurs, requiring higher support density.
4. On approaching a shear zone or an imminent outburst zone, increase in jointing, increase in gas emission at the face, and decrease in strength of coal has been noticed. Shear zones vary in width from a

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few centimetres to several centimetres. Completely pulverised coal has a silken touch.

Based upon these conclusions a number of techniques, both active and passive have been used to control outbursts. An important active technique is the removal of gas from the seam ahead of the advancing roadways using well known gas drainage systems. This concept is based upon the well tested assumption that gas plays a dominant role in the precipitation of outbursts. This feature has been established at West Cliff Colliery.

If this were not so, then outbursts must have occurred both in development and in depillaring areas. It was observed that while shear zones release a large amount of gas during development, the amount of gas released during the pillar extraction stage, when these shear zones were crossed, was not significantly higher than normal. This suggests that since most of the gas has been released from these zones, their activity has dropped. If stress was a dominant cause of outbursts, then the severity should be higher during pillar extraction, due to higher stress concentrations on the shear zones. However, the absence of outbursts in pillar extraction areas where shear zones have been intersected confirms the dominant role of gas in outbursts at West Cliff Colliery. It is therefore strongly felt that if shear zones can be drained the problem of outbursts can be greatly overcome, if not completely eliminated.

GAS PRESSURE CLOSE TO SHEAR ZONES

Gas pressure measurements have been conducted at a number of points at West Cliff Colliery using a multiple packer system. The principle of operation is shown in Fig. 1. The multiple packer system consists of an inflatable hose with a stainless steel tube passing through it. One end of the hose is fixed to a screwed coupling which is mounted rigidly on the stainless steel tube passing through the hose. The other end of the hose is fixed to a sliding end moving on the steel tube with two O-rings

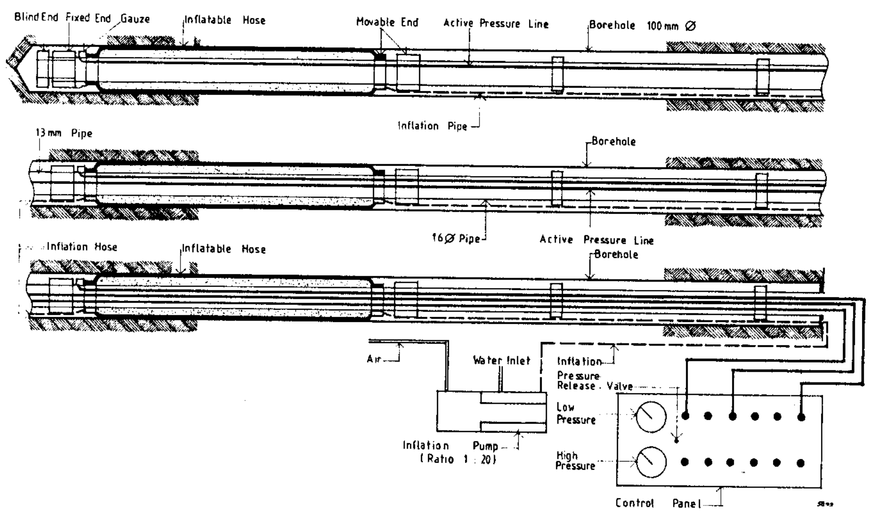

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Fig. 1.- Triple packer system to monitor gas pressure in boreholes.

serving as a seal for the inflatable hose. The internal diameter of the hose is 22 mm and the space between the stainless steel pipe and the hose is inflated using water pumped from an intensifier through a 3 mm diameter nylon hose into a nipple drilled through the movable coupling. Another nylon hose (active pressure line) is threaded through the inside of the steel tube and ends in a spigot opening to the outside. This open end is protected with a circular wire gauze of 25 mesh and a bronze ring with 5 mm diameter holes. This permits free flow of gas into the pipe and protects it against blockage. The nylon hose is brought right to the collar of the hole where it is connected to the control panel.

The internal steel pipe is extendable and thus the packer can be placed at any desired position in the measuring hole. The second and third packers with the same arrangement are placed at desired points in the hole with separate active pressure measuring hoses coming right up to the collar of the hole. There is, however, only one dilation hose which passes outside the extendable pipe and feeds into the space between the stainless steel tube and the inflatable hose. The inflatable hoses for different packers are interconnected in series using suitable nipples. The multiple packer system can accommodate up to a maximum of four measurement packers in a borehole simultaneously.

The packers are inflated to 5000 kPa using water as an inflation medium. Gas pressure build-up in holes is monitored through the control panel, which is capable of monitoring 18 points (six triple packers). Fig. 2 shows the 'Inflation characteristics of petrometallic dilatable hose, lm. in length'.

Using the above technique holes were drilled at various distances from the shear zone. Gas pressures were measured at various depths from the wall of the roadway. The results of tests are given in Fig. 3. These clearly indicate that close to the shear zone, very high gas pressures could exist. The explanation of these changes in high gas pressure close to the shear zone is sought in the permeability of the shear zone. Permeability of sheared coal (parallel to bedding) is about 10 - 1000 times higher than that of intact coal samples. (Permeability of coal is of the order of 0.01 - 0.1 milli darcy). The strong influence of stress on permeability indicates that if the shear zone is in a region of high stress, it will not be possible to drain it. Other observations of the existence of higher pressure in the shear zone are described in the paper "Occurrence of Outbursts at West Cliff Colliery", included in this Symposium (Marshall, Griffiths & Lama, 1980).

DRAINAGE OF SHEAR ZONES

Studies indicated that both these factors, existence of high gas pressure in the shear zone and its high permeability should facilitate drainage of these shear zones. If these shear zones can be sucessfully drained, then perhaps the outbursts can be completely eliminated.

In an experiment, three boreholes were drilled (Fig. 4) in the ribside of a roadway. The central borehole (No.2) had a length of 45 m and intersected the shear zone running almost parallel to the rib at a distance of 45 m. A single packer was installed in this borehole at a depth of 30 m. A short borehole (No. 1) of 23 m length was drilled to monitor flow through this hole. Another borehole (No. 3) was drilled to intersect the shear zone. Gas flow measurements showed that flow rate from borehole No. 1 was only 0.35 ℓ/min/m length. Flow rate from borehole No. 3 was 63 ℓ/min/m which is about 20 times the flow rate compared to flow rate from hole No. 2. Pressure and flows were monitored over a period (Fig. 4). Results indicate that pressure in

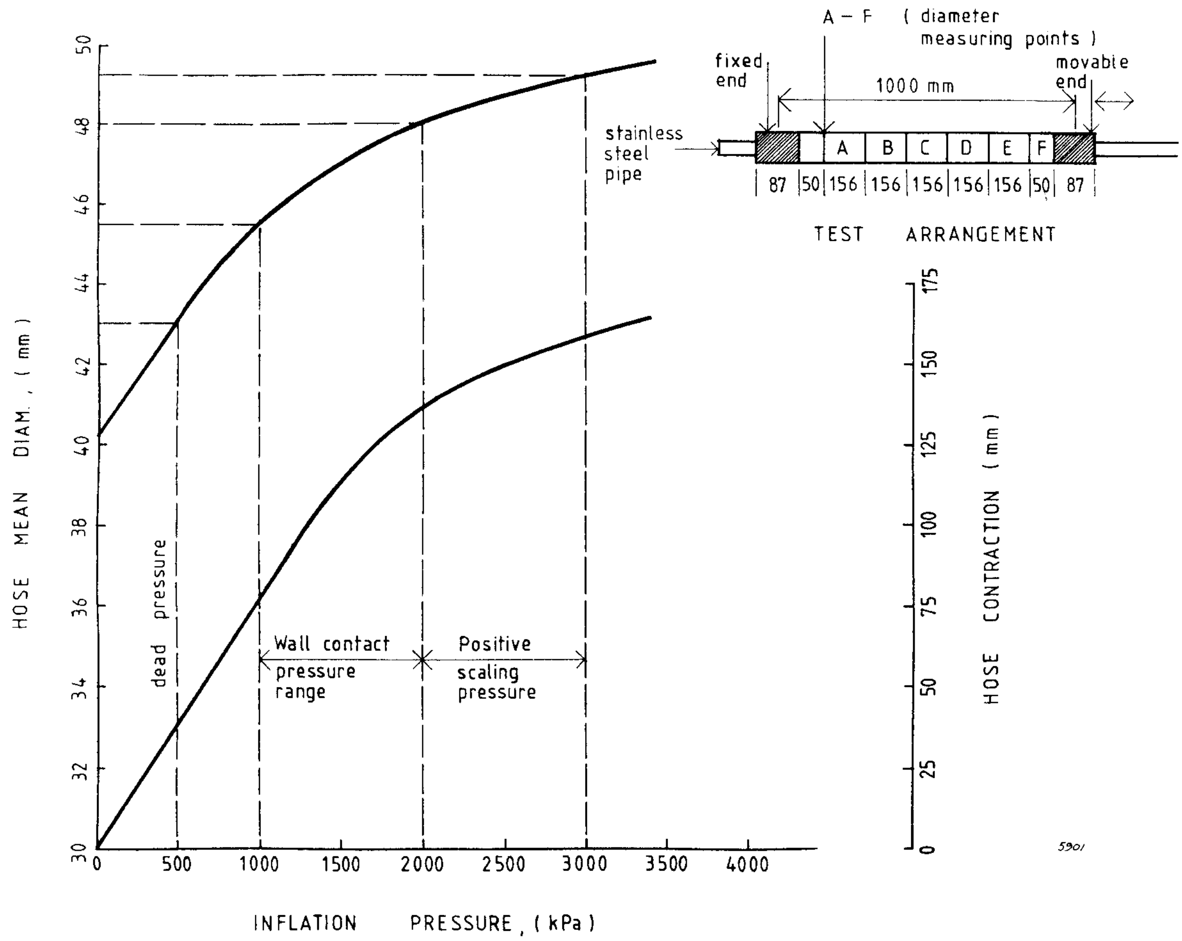

Fig. 2.- Inflation characteristics of petrometallic dilatable hose, 1 m in length.

the shear zone can be dropped to almost half the original value over a period of 30 days and almost to one-tenth of this value over a period of 140 days under free flow conditions. These results have been confirmed in subsequent measurements. Fig. 5 represents the flow rates from boreholes. Boreholes 28 - 35 do not intersect the shear zone running parallel to the ribside. Borehole 36 intersected the shear zone and the flow rate almost quadrupled. After drilling borehole 36 the next borehole, 37, was drilled, and this resulted in sudden drop in the flow rate from borehole 36.

Figure 6 gives a plan of the excavation where the shear zone was drained over a period of almost five months as indicated in Fig. 4. A total of 4.5 million litres of gas had been drained from the shear zone. When the development roadway of Panel 402 intersected the shear zone AA, it gave rise to serious outbursts. When this shear zone was intersected five months later, by driving development roadways for Panel 121, no activity was observed. The deputy on site reported only a slight loosening of coal and no marked deterioration of roof conditions. This proved

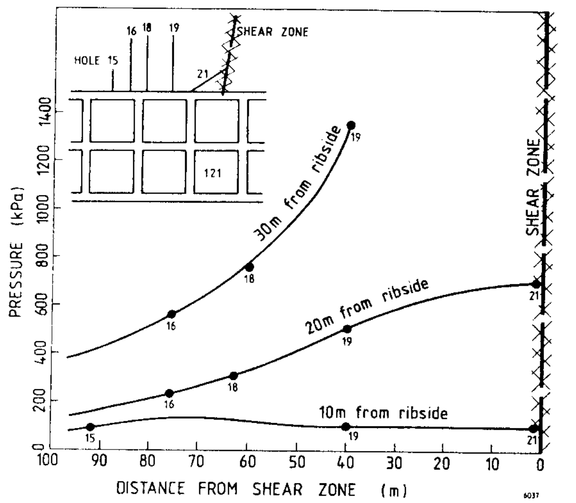

Fig. 3.- Gas pressure changes close to a shear zone.

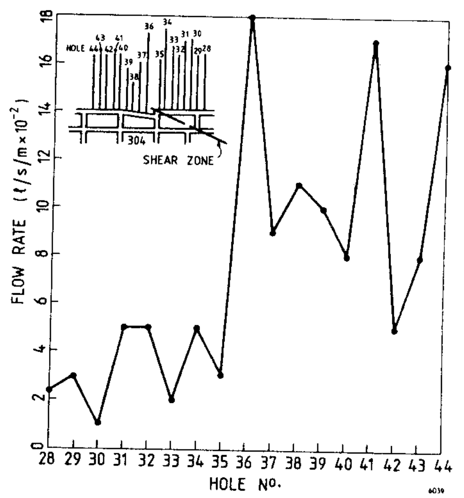

Fig. 5.- Gas flow rates from holes and location of shear zones.

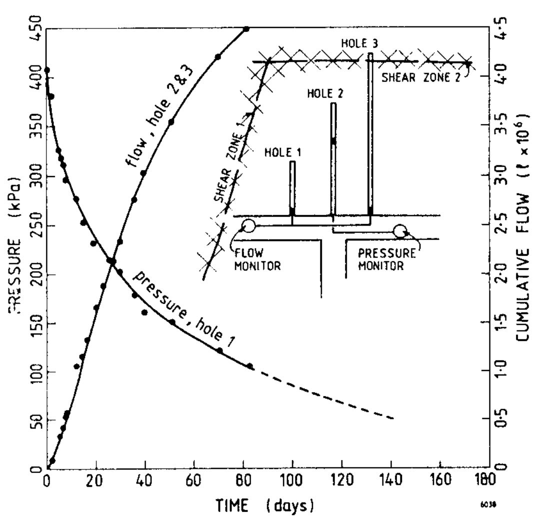

Fig. 4.- Gas pressure and gas flow from shear zones.

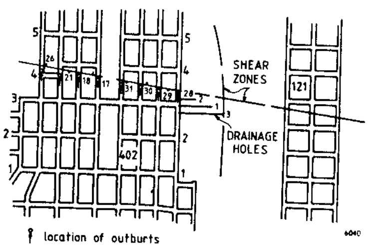

Fig. 6.- Layout of Panel 121, shear zones and drainage holes.

conclusively that the drainage of shear zone AA had helped lower gas pressures and control outbursts.

These initial experiments showed that shear zones can be successfully drained.

Analysis of outburst occurrence at the Colliery also indicated that successful drainage of shear zones is possible when these lie at some distance (1> 10 m) away from the development roadway (Marshall, Griffiths, Lama, 1980). When these shear zones have been already intersected, running parallel to the roadways and lying in the centre of the roadway, it is almost impossible to drain these zones. It would mean that very long holes will have to be drilled in advance of the workings to successfully drain shear zones and coal in Bulli seam at West Cliff Colliery.

INVESTIGATIONS INTO DRAINAGE OF METHANE GAS FROM THE SOLID

In gas drainage investigations, the following aspects need to be examined.

1. Gas content of coal.
2. Virgin seam gas pressure.
3. Permeability *in situ* or expected flow rates from boreholes.
4. Optimum suction pressure to be applied in a continuous gas drainage installation.
5. Influence of cleat on the gas make in a borehole and location of borehole in the seam with respect to its structure.
6. Optimum diameter of drainage hole.

Preliminary observations at West Cliff Colliery showed that the two cleat systems in Bulli Seam are more or less equally developed. This would indicate that perhaps the direction of drill holes for drainage would not very much affect drainage rate. Structure of the seam also indicated that the mid-section of the seam is most permeable. The top 500 mm of the seam contains larger, widely spaced joints, while the mid section contains a network of many closely spaced joints. This therefore suggested that all drainage holes must be placed in this mid section or must intersect this section of the seam.

The influence of diameter of the borehole on the gas flow rate has been studied by a number of investigators and conflicting opinions have been reported. From indirect evidence from work by Russian investigators (Nedashkovskii, Panteleev and Voronkov, 1977) it is concluded that holes of small diameter are most effective. Japanese investigations (Kiyoma, 1972) report quite the opposite, while investigations conducted in Poland are inconclusive (Cias, 1976). No attempt has been made to investigate this factor at West Cliff Colliery up till now.

Investigations designed to determine other parameters are described briefly. More detailed information is given in Lama (1980).

GAS CONTENT OF COAL

Three different methods were used to determine gas content of coal from Bulli seam at West Cliff Colliery. These include:

1. Methane based upon general gas make in the mine.
2. Localised gas emission technique and adsorption method.
3. Direct measurement of gas content of coal.

The first method is based upon the total gas emitted into the ventilation stream of the mine with respect to the tonnage mined. Observations were made over an extended period when the mine was working and over a period when mine was closed. The difference in the gas emissions was calculated and compared with data from overseas mines. This data gave a gas content of coal about 13 m^3/tonne.

The second method is based upon the emission of gas at the face during mining operations and measurement of gas pressure at the face. This data was further extrapolated to the maximum gas pressure measured in the coal. Coal samples were subjected to measured gas pressures (both at low pressure and higher pressure adsorption) and amount of gas adsorbed by coal was measured. Based upon the adsorption capacity of coal, the gas content of Bulli seam was estimated at 17 m^3/tonne (Fig. 7).

Gas content of coal was measured directly using core samples from surface boreholes drilled to Bulli Seam in adjoining areas of the mine. (These tests were carried out by Mr. P. Lamb of Coal Development Company - a sister organization of West Cliff Colliery.) The results of these measurements gave gas content of Bulli seam varying from 9 - 15 m^3/tonne.

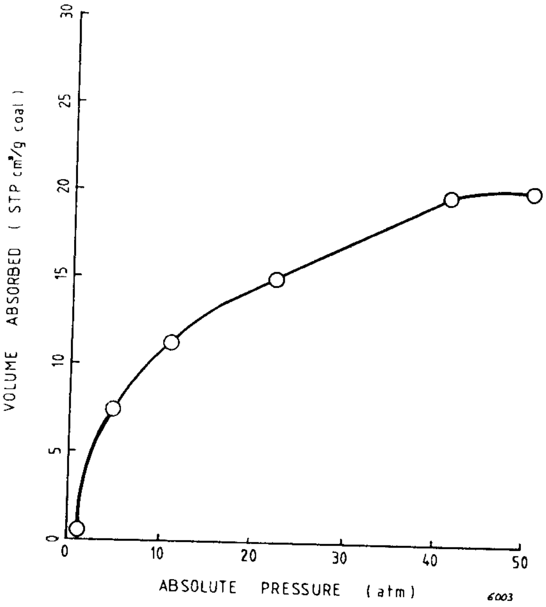

Fig. 7.- Methane adsorption in coal at high pressure, Panel 304, West Cliff Colliery (Temp. ?20^{o}C). Courtesy Intercomp, U.S.A. 1979).

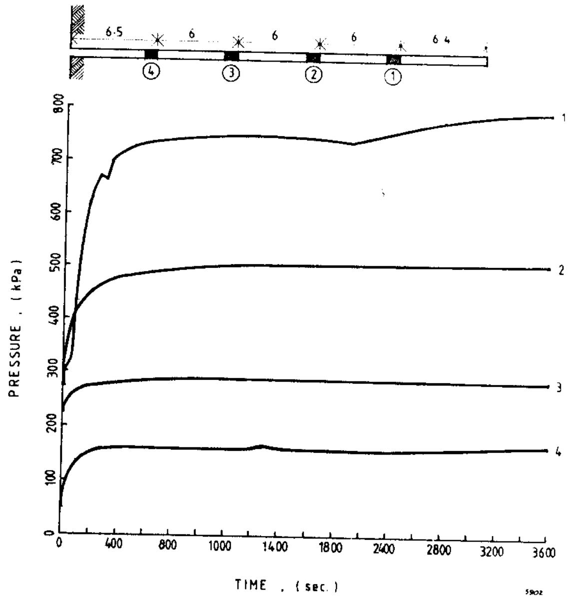

Fig. 8.- Pressure build-up as a function of time using a Quadruple packer system. Hole 33, Panel 121, length of hole 40.5 m, seals placed at 76.5 m from collar of the hole, with 6 m chambers between the scale, inflation pressure 13000 kPa.

IN SITU GAS PRESSURE MEASUREMENT

In situ gas pressure measurements were carried out using the triple packer system as described earlier. Tests were conducted in eight holes at depths varying from 10 m to 40 m. Fig. 8 gives results of a typical measurement in a borehole. Pressure rises slowly in the borehole chambers and needs almost 100 to 500 hours to stabilise to its maximum value, depending upon the distance from the borehole. Pressures were measured using both a single packer and triple packer system in the same borehole to determine any difference between the two systems. Pressures measured using a triple packer were 1 - 1.5 times those measured by a single packer. Maximum pressure at any point depended upon the depth of measuring point, time elapsed between the drivage of headings and measurement and presence of geological discontinuities.

Pressure gradients in the rib sides were determined using the multiple packer system. These measurements were interpolated to estimate the depth of the fracture zone. This depth of the fracture zone has been taken as the point where pressure drops to zero (Table 1). These results show that the zone of fracturing in general extends to about 6 m but never beyond 9 m. This is an important conclusion and has been used in deciding the length of the grouted pipe.

The maximum pressure measured was 2670 kPa in a hole 40 m deep drilled in panel 121 at an angle of 45^{o} to the main cleat. This corresponds to a vertical depth of 267 m (assuming gas pressure = 0.4 times the gravitational pressure). It is expected that in deeper holes pressures of the order of 3000 - 4000 kPa may be present. A 10 - point

Table 1

Depth of fracturing around an excavation, West Cliff Colliery, Section 121, roadways driven north-south, east rib side.

Hole No.	Distance from rib side, m
33	3.07
34	3.32
16	4.16
18	6.32
19	8.02

packer system has since been developed and it is expected that better pressure profiles will be obtained in holes up to 100 m depth using this system.

GROUTING SYSTEM FOR DRAINAGE HOLES

Tests using pressure packers proved that the first 6 metres of the rib side may be fractured and hence seals in this part of the roadway would be effective. It was therefore decided to use seals of 6 - 9 m in length.

A number of grouting materials (Table 2) were tested to determine their suitability under simulated borehole conditions. All these materials, with the exception of "Patternstone U" and Polyurethane Foam ICI, are suitable as a grouting material. The permeability of these materials is almost one-tenth that of coal under high stresses.

The technique of pumping these grouting materials and the difficulty of placing these materials at depth in a borehole, however, creates some problems. Laboratory investigations designed to inject these materials into boreholes at depth, and subsequent field trials, showed that non-shrink grout LL893* is the most suitable material for the purpose. At a later stage a water-based chemical grout Celtite** was also tested in field conditions and has been found successful.

A number of overseas systems were investigated from cost and safety point of view. PVC pipes made in Australia were tested to measure the development of static electrical charges (if any) under simulated field conditions and were found unsuitable. Subsequently, a new stand pipe arrangement was designed and tested successfully. This arrangement costs less than about 20% of the overseas systems. It consists of a copper pipe (Fig. 9) followed by a galvanised pipe and jointed together using brass self-soldering Yorkshire coupling. Two polystyrene seals are slipped over the copper pipe and secured using PVC tape. A 12 mm diameter nylon hose passes through the polystyrene seal placed close to the Yorkshire coupling and extends close to the in-bye end of the copper tube to serve as a bleed hole. The second nylon hose extends only a short distance from the Yorkshire coupling and serves as a grouting hose.

A 48 mm diameter hole is drilled first to 9 m length and then reamed to approximately 88 mm to accommodate the stand pipe. The prepared grout is then pumped into the hole using a mono pump taking care that the operation of mixing and grouting can be completed within 30 minutes.

Results of tests on the grouting system have shown that the system seals the hole effectively even at a vacuum of 40 kPa, without any dilution of the gas by leakage through the grout or grout-coal interface. The cost of the grouting system is only 10% of the total cost of a drainage hole.

*LL893 is a product of Embecon Pty. Ltd., 79 Victoria Avenue, Chatswood, N.S.W. 2067.

**Celtite is a product of Celtite Selfix Ltd., 81 Baxter Road, Mascot, N.S.W.

TABLE 2

AIR PERMEABILITY* OF GROUTING MATERIALS

Material	Permeability*, md	Remarks
Polyurethane Foam ESCON Foam A Foam B	0.0179	Mixing Ratio : 1:1 by weight Setting Time : 4.0 minutes after mixing
Polyurethane Foam I.C.I. Suprasec DND Daltolac 41	7.6280	Mixing Ratio : Mix by weighing 1.8 parts of Suprasec DND with 1.0 parts of Daltolac 41 Setting Time : 3.0 minutes after mixing
Polyurethane Foam Fomofill	0.0205	Setting Time : Ready mix, approximately 72 hours
Polyurethane Foam Baygal K55B Baygamur K88	0.0980	Mixing Ratio : 1:1 by weight Setting Time : 4-5 minutes after mixing
Non-shrink Grout L893 Grout	After : 6 days = 0.0216 17 days = 23.276	Mixing Ratio : Mix by weighing 2.75 parts of cement with 1.0 part of water
Non-shrink Grout LL893 Grout	After : 6 days = 0.142 16 days = 29.096	Mixing Ratio : Mix by weighing 1.83 parts of cement with 1.0 part of water
Gypsum Patternstone "U"	281.100	Mixing Ratio : Mix by weighing 100 parts of Gypsum with 38 parts of water

* It is not the permeability as defined by ISRM, but some form of leakage coefficient as measured under simulated test conditions for use in drainage installations.

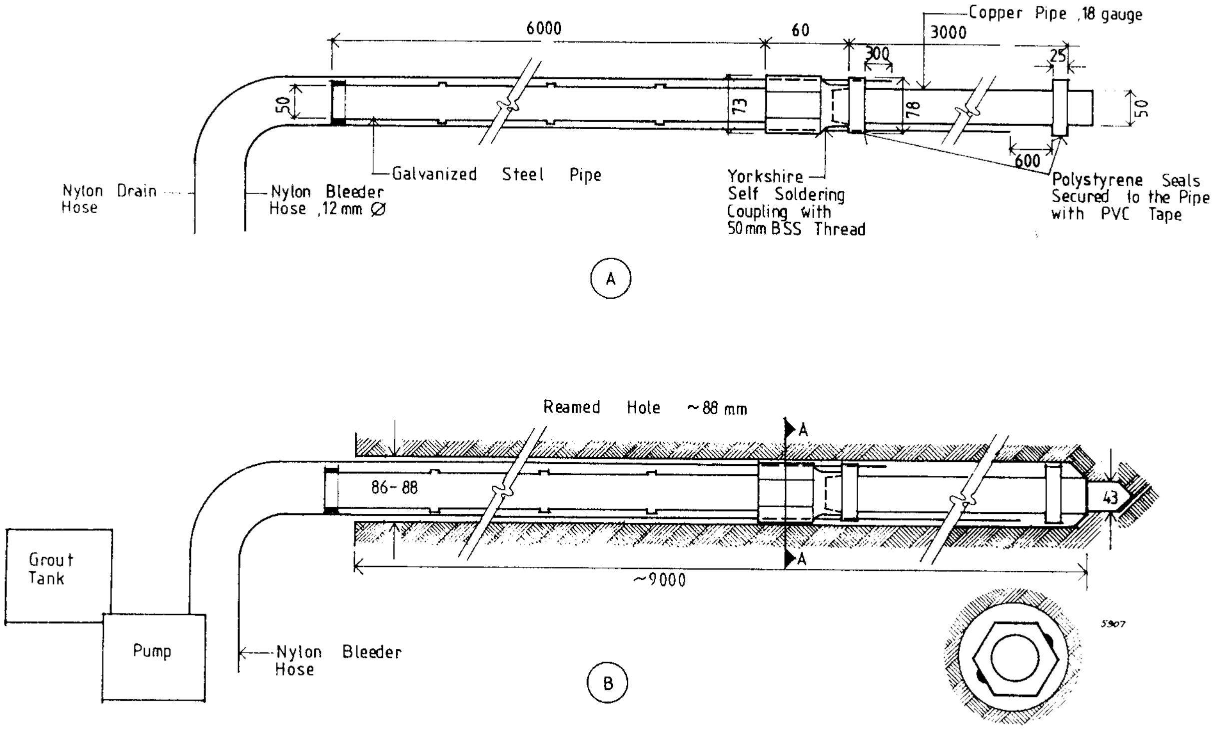

Fig. 9.- (A) standpipe ready for grouting; (B) standpipe grouted in hole using a grouting pump.

SPACING OF DRAINAGE HOLES

The distance between drainage holes becomes important from two viewpoints :

1. Total cost of drilling and equipping holes may make drainage uneconomic, when holes are too closely spaced.
2. If the holes are too widely spaced, time required to successfully drain may be so long that efficient drainage may not be possible.

Optimum spacing of drainage holes, therefore, has to be considered both in the light of economics and the lead time available. The method used to determine the spacing at West Cliff Colliery is based upon the influence of a drainage hole on the changes in gas pressure in the area around it. The planned layout of holes is given in Fig. 10. In practice, however, it is hard to drill holes parallel to each other. Precautions were taken to stabilize the machine and accurately orient the direction of drilling. Some deviation occurred, but no survey of holes was conducted after drilling. The actual length of holes was as follows:

Hole 15	=	18.0 m
Hole 16	=	41.2 m
Hole 18	=	44.2 m
Hole 19	=	44.2 m

Packers were installed in holes and pressures were measured over a period of up to 300 hours to ensure that the system had stabilised and the maximum pressures had been achieved. In this period, the pressure slowly dropped due to natural drainage after the peak was achieved at different measuring points. A drainage hole (100 mm diameter) was then

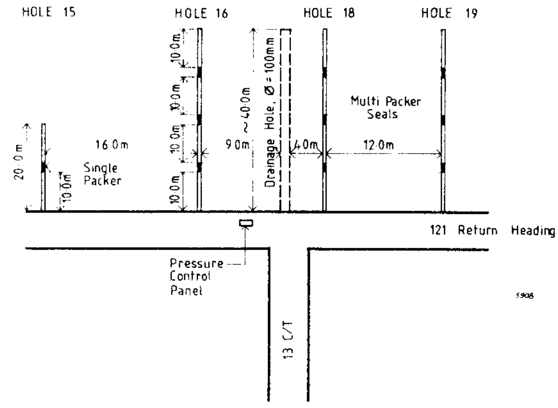

Fig. 10.- Arrangement of test to determine sphere of influence of a drainage hole, West Cliff.

drilled. The length of this hole was planned to be 40 m, but due to deviation from its plane, it struck the floor at a depth of 24 m. The drop in pressure in the various holes at different depths was measured on completion of the drainage hole. The results are given in Fig. 11.

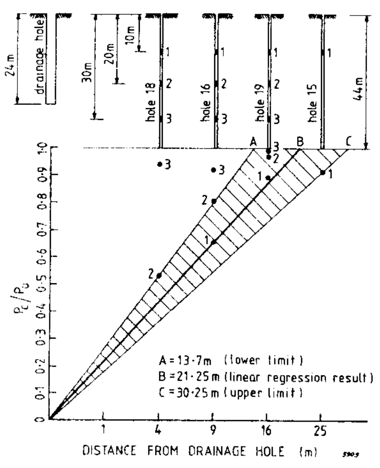

Fig. 11.- Effect of sphere of influence of a drainage hole.

In the analysis of results, the in-bye triple packer results have not been taken into account, because of the limited length of the drainage hole. These results show that the influence of a drainage hole could extend up to 30 m. The minimum distance could however be about 13.7 m. The effective value is taken as 21.25 m.

Results of tests conducted on hole 42 (Panel 304) and holes 2 and 3 (Panel 121) over an extended period have been monitored. Fig. 12 gives results of tests on these holes. Normally one would expect flow rate to follow an exponential decay function with time. Laboratory investigations were conducted on desorption of gas from coal samples and results showed that the adsorption of gas can be approximated to a logarithmic law. Field results, however, tend to show flow rates from boreholes as a linear function with time. The reason is possibly the lag time between drilling and grouting and testing of holes. This period under field conditions has varied from 2 - 3 days to a week. As a result the later part of this curve can be approximated to a linear function with time. For shorter holes with smaller initial flow rate, the flow rate would drop to almost 10% of the initial flow rate after 100 days. For larger holes, with high flow rates (hole 42), considerable flow rate would be maintained and a drop to 10% of initial flow rate may take much longer.

Based upon these two results, projections have been made to correlate flow rates as a function of time (Fig. 13). These projections are based upon limited data and need to be refined, but they do give some idea about the possible flow rates and active life of drainage holes.

The above points must be considered in deciding upon the spacing between the holes. If the life of the hole is decided upon by cutting it off when the flow rate has dropped

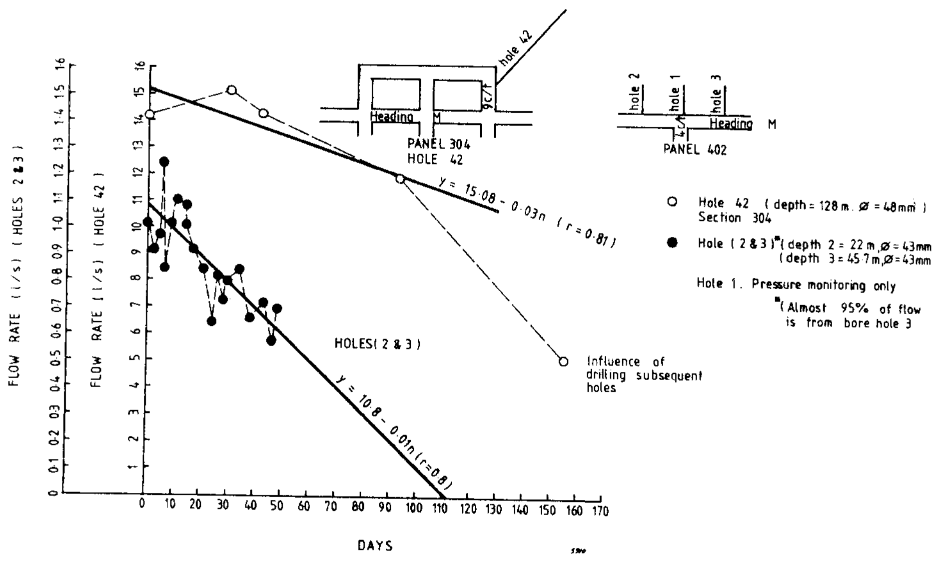

Fig. 12.- Influence of time on flow rate from holes under free flow conditions.

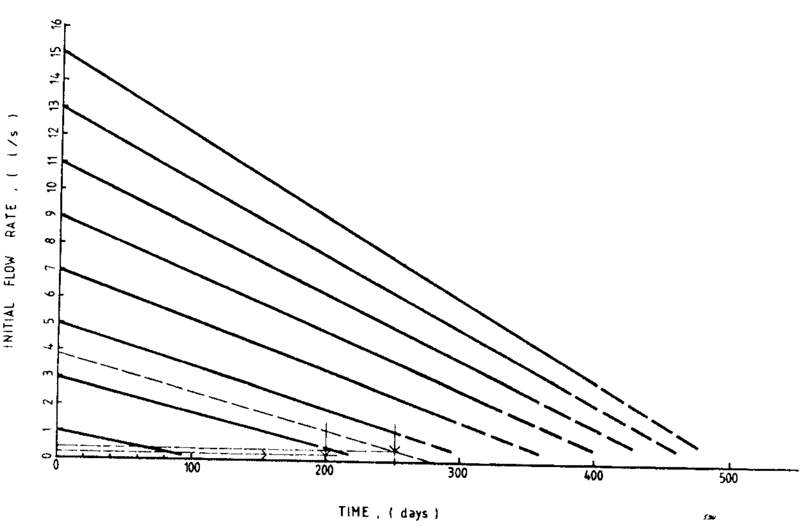

Fig. 13.- Flow rate as a function of time, West Cliff Colliery.

to 10% of its initial value, this then might become a deciding factor in drainage hole spacing.

Initial studies conducted on free flow of gas from various holes showed that flow rates of 1.0 - 2.0 ℓ/min/m length of hole could be expected. These results were based upon the rate of build up of pressure in boreholes, or flow rate measured by varying the position of the seal in the borehole. The mean of 42 measurements gave a value of 1.96 ℓ/min/m for a borehole. In Panel 304 where 120 m long boreholes are being planned, the expected gas flow readings should be about 235 ℓ/min for each hole. This would mean that the productive life of a hole could be estimated at about 246 days. The spacing of holes under these conditions would be 10.27 m.

The distance between holes recommended for Panel 304 is 10 m and holes in this Panel are being drilled regularly at this interval. This would also ensure superposition of areas of influence and eliminate blind spots.

INFLUENCE OF SUCTION ON FLOW RATES

The main purpose of applying suction to the drainage holes is to cause movement of gas in the hole and grouted pipe and to eliminate any development of positive pressure at the collar of the drainage hole that could cause liberation of gas into the mine roadways through fractured ribs. On the other hand, suction may cause dilution of the sucked gas due to leakage through the grouted seals, pipes, or the rib sides.

For the purpose of the exercise a system for flow measurements, sampling and suction pressure generation was designed and tested. This was associated with the grouting system, to ascertain the optimum suction pressure which permits sufficient velocities in the hole and gas ranges and does not cause any dilution of the gas.

Fig. 14 is a schematic diagram of the method. Fig. 15 shows the test apparatus underground, which is now used as a standard method at the Colliery for testing grouting efficiency and for collection of data on the flow of gas from drainage holes.

The method of testing holes consists of applying suction pressure in stages from zero (free flow) to a maximum of 40 kPa. Suction is applied using a venturi connected to the end of the test pipe fitted to the end of the drainage pipe. Suction applied to the drainage hole is regulated by varying the amount of air supply to the venturi. Standard venturies used for stone dusting underground can develop suction up to 40 kPa, which covers the range of suction applied underground in any gas drainage investigations. Flow in the test pipe is measured using an ellipsoidal-nose pilot tube inserted into the test pipe, with the centre of the nose located accurately at the centre of the pipe and aligned with its axis. Temperature and suction pressure is measured just behind the pilot tube point at a distance equal to at least 5 times the diameter of the test pipe. The length of the test pipe is 30 times the diameter and the pilot tube is placed at 15 times the diameter from the 50 mm shut off value located at the out-bye end of the drainage hole.

On application of suction, gas samples are collected using a compressed air driven suction pump. The rate of sample collection is so regulated that it does not disturb the suction applied. Samples are tested on site for percentage of methane using a Riken interferometer with a range of 0 - 100%, and an accuracy of ± 0.2%.

During testing, drainage holes are dewatered by applying maximum suction and allowing the hole to run till no water flows out from the drainage hole. Supply of compressed air is stopped to the venturi and gas flow from the drainage hole is allowed to

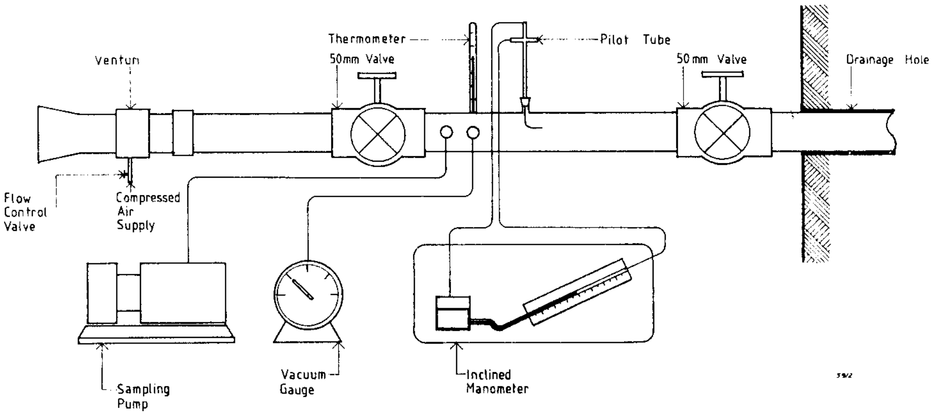

Fig. 14.- Schematic arrangement for testing drainage holes under suction.

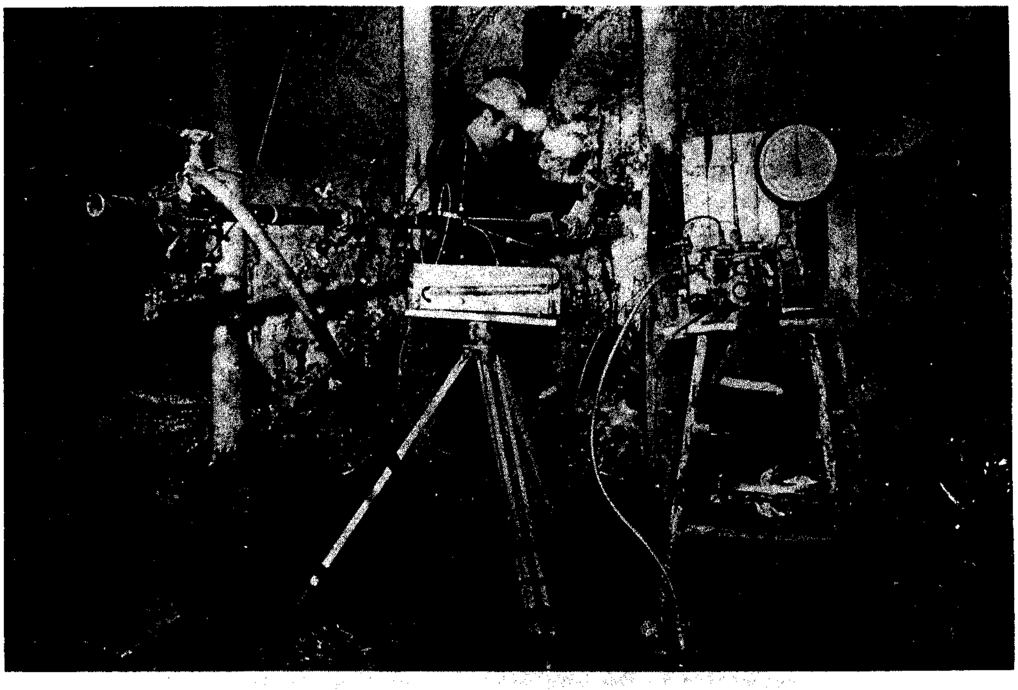

Fig. 15.- Testing a gas drainage hole under suction at West Cliff Colliery.

stabilize. Free flow is measured and gas samples are taken for analysis. A minimum of six readings are taken for flow pressures and a minimum of five gas sample analyses are carried out. Suction pressure is then raised in steps of 3.3 kPa (1 inch of mercury). Flow is allowed to stabilize for a period of at least five minutes during which flow pressure, temperature, and gas sampling and analysis is carried out. All tests were carried out with the increasing suction pressure. Barometric pressure is recorded both before and after the test.

Results obtained on application of suction to drainage holes have been calculated using the relationship

$$V = 1.6038\sqrt{\frac{1000}{P_o} \times \frac{T}{289} \times \frac{100\ 000}{1000+P_s} \times P_v}$$

where V = velocity (metres per second)

P_o = barometric pressure (milibars; 1 milibar = 100 Pa)

T = absolute temperature (degrees K)

P_s = duct static pressure (Pascals; negative in this case)

P_v = velocity pressure (Pascals)

The factor 1.6038 takes into consideration the composition of gas sucked which has been assumed as follows :

CH_4 = 95% v/v

CO_2 = 2.5% v/v

H_2O = 2.5% v/v

Volume of gas flowing through the borehole is further corrected for the percentage of CH_4 to estimate the amount of methane sucked in relation to the total amount of gases (including air) sucked and reduced to STP. A typical test result is given in Fig. 16. These results show that suction increases the flow rate in holes by a factor of 2 - 4 times. This would not be normally expected. It is possible that this increase is only temporary due to sudden desorption of gas close to the boundary of the hole.

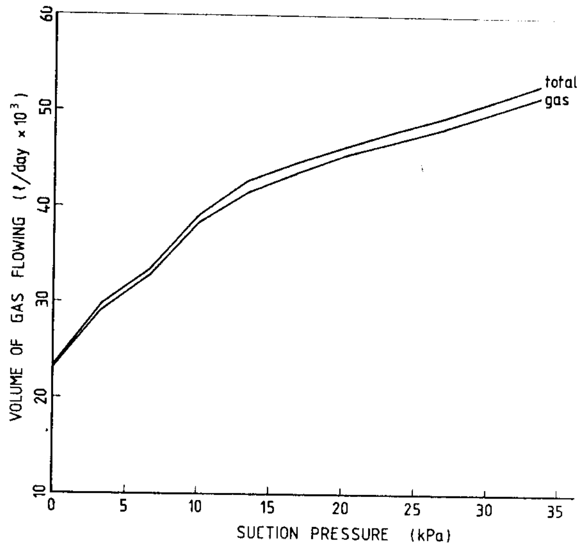

Fig. 16.- Influence of suction on flow of gas from borehole 304-29, drilled on 4/12/79, tested on 12/12/79, depth 90 m.

Laboratory tests conducted on adsorption of methane on coal samples at pressures below and above one atmospheric show marked non-linear effects. This effect is being further investigated.

Results of tests indicate that suction at the holes could be as high as 33 kPa (10 inches of mercury). By increasing suction if high flow rate is maintained the productive time of a hole could be reduced depending upon the increase in flow rate, thereby decreasing the lead time required. It is also hoped that percentage recovery of methane from the solid could be greatly increased.

The results of 36 holes drilled at a regular distance of 10 m in Panel 304 gave a flow rate of 2.1 ℓ/min/m length of hole. This flow rate is a bit higher than that obtained from pressure and flow measurements from earlier observation, but the difference is not very great. This value will improve as fresh areas are exposed.

CONCLUSIONS

The results of this study prove conclusively that shear zones are regions of much higher gas pressures and these changes in gas pressure can be picked up almost 50 m ahead of the shear zone. This can be an effective method of predicting the presence of a shear zone ahead of a face. With the design of new equipment, it should be possible to use this technique to forecast the existence of shear zones.

Shear zones are also zones of higher permeability, and can be effectively drained provided they are not influenced by stress distribution due to excavations (which could affect the permeability of these zones very adversely).

Investigations conducted on advance drainage of methane gas from the solid, using horizontal holes drilled from existing excavations at West Cliff Colliery, have proved positively that drainage of methane is possible under mining conditions at the Colliery. Research conducted to date has established the following parameters:

1. The gas content of Bulli coal lies between the limits of 9 - 13 m^3/tonne, and perhaps closer to 13 m^3/tonne.
2. The maximum gas pressure measured in the Bulli Seam using a multiple packer system was 2670 kPa. However, higher virgin gas pressures must exist.
3. The depth of fracturing around excavations is less than 6 m, and a grout length of 3 m placed at 6 - 9 m from the rib side is effective in sealing drainage holes.
4. A new grouting system for drainage holes has been developed and proved successful. This system is much cheaper than overseas systems.
5. The optimum spacing between holes in Bulli Seam at West Cliff Colliery has been established as 10 m, depending upon the lead time available for drainage. The influence of a drainage hole, however, extends to a maximum of 30 m. If larger lead times are available, the distance between holes could be increased to 60 m.
6. Application of suction to the drainage hole increases flow rates to 2 - 4 times the free flow rate values. It is not clear whether this is a temporary increase in flow rate or a permanent feature of the pattern of flow of gas in the coal seam and holes.

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