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Mine gas and coal dust explosions and methane outbursts - their causes and prevention.

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Mines Safety Code dealing with headgear earthing, should be followed.

11. The following recommendations are made for the protection of transmission lines feeding power to sections of a mine by means of borehole cables:

- Earth wires should be located so as to shield the line conductors from direct lightning strokes. In this connection it must be appreciated that earth wires should be carried above the conductors and the height above and position relative to the conductors shall be such that the angle of protection i.e. the angle included between the line from earth wire to outside conductor and the vertical is not more than 45°. This will give 99.9% protection and an angle of 20° may be taken as giving 100% protection. The earth wires shall be of adequate section and due regard is to be taken of midspan clearances to avoid flashing over to conductors.

- Lightning arrestors should be installed at each point along the line where a borehole feeder is taken off. At these points an effective low resistance earth is to be established to which is solidly connected the lightning arrestor ground lead, the tower, the transformer low tension neutral, the cable armouring and all metalwork at this point.

In this connection it is recommended that the best arrangement is a long driven rod or pipe in the ground placed a distance of at least 6 metres from the base of the transmission line tower and connected thereto by a bare copper conductor at a depth of 400 mm below the ground surface. In certain cases due to the nature of the ground, it may be necessary to drive additional rods to ensure a low earth resistance and these additional rods are to be placed

at least twice the length of the rods apart and connected together.

Attention is also drawn to the necessity of ensuring that all explosive firing cables and conductors of firing circuits are kept away from underground electrical cables and apparatus and also from all pipework and tracks.

12. The standard instructions which deal with power failures and stoppages of main fans (with consequential methane accumulations) is dealt with in Chapter 2.

13. Mining Engineers should ensure, by studying all standards on the mine, that prior to commencing in their job, they acquaint themselves with all standards and exemptions issued by the Mines Department.

14. Each colliery has an Emergency Procedure Manual which should be studied by Mining Engineers. Only trained men wearing rescue sets should enter an area following a methane explosion. The Author appreciates the plight of senior officials following an explosion but foolhardy acts undertaken without group discussions and consensus can only result in the loss of further life.

15. Only flameproof lights and equipment should be used in sinking shafts where the shaft itself is the return airway from the face area. See Figure 12.

16. The use of bleeder roads and boreholes in stooping sections is discussed in Chapter 5.

17. Senior Mine Management's quarterly inspections of survey plans with the Mine Surveyor should include an inspection and discussion on whether all surface boreholes are shown on the mine plans and the condition of each hole. Does it pose

a danger insofar as lightning strikes are concerned, for example?

10. Tests for the presence of methane accumulations in sections where work has ceased should be made at least on a weekly basis, particularly if the ventilating quantity has been substantially reduced.

No brick stoppings should be removed, the consequence of which would short circuit ventilation unless the Mine Overseer has agreed to this in writing.

The Mine Overseer and Mine Environmental Officer should personally check and follow up immediately after changes have been made to ventilation to ensure the accuracy of air quantity calculations. Shift Overseers should not be empowered to make ventilation changes.

3.8 CONCLUSION

In workings in coal seams which are shallow (less than 200 metres deep) and where methane has accumulated in explosive proportions, lightning strikes and stray currents can result in surges leading to sparking which may initiate methane explosions.

Lightning may strike the surface above an underground working area and the surge could be propagated through the strata into the face. Alternatively, a strike occurs at the shaft entrance and is conducted into the face via metallic structures.

Several cases have been investigated where such incidents have occurred.

Well designed earthing systems in the vicinity of shafts are necessary to absorb and dissipate such voltage surges.

Disused boreholes should have casings removed where possible and they should be sealed.

The sealing off of old workings or the adequate ventilation thereof is discussed in order to prevent the accumulation of explosive concentrations of methane.

Reliable lightning warning systems have been installed on shallow mines in high density lightning strike areas. The charging of explosives ceases while thunderstorms are in the vicinity.

3.9 REFERENCES

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CHAPTER 4

IGNITIONS OF FIREDAMP BY BLASTING OPERATIONS AND MINE EXPLOSIVES

4.1 INTRODUCTION

4.2 THE LAG OF IGNITION OF METHANE

4.3 EXPLOSIVES

4.3.1 Definitions
4.3.2 Introduction
4.3.3 Development of Permitted Explosives
4.3.4 Nature of Ignitions
4.3.5 Temperature of Ignitions
4.3.6 Comparison of Blasting Effects of Direct and Indirect Ignitions
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4.1 INTRODUCTION

The greatest number of methane and coal dust explosions in South Africa has been caused by blasting operations and explosives. The number of such incidents far exceeds those caused by any other igniting source. More men have lost their lives as a result of this igniting source when compared with other types of ignitions and in addition several major coal dust explosions have followed these methane explosions.

Since a high percentage of coal is won by all mining (cutting, drilling and blasting) and coupled with the fact that almost all blasting operations take place in the working faces where gas is more likely to be present in dangerous quantities, it is not surprising that this ignition source is responsible for the greater percentage of explosions.

The role which blasting operations and explosives play in causing the ignitions is discussed together with the effect which the lag of ignition has on the propogation of explosions.

Coal dust explosions figure prominently where blasting is the ignition source. The nature of this type of explosion and the devastation which accompanies such a phenomenon is dealt with in this chapter.

Two incidents have been singled out for analysis - these are the explosions at the Durban Navigation Collieries No. 2 Pit on the 8th October 1926 and the Northfield Colliery explosion of 1943. 125 men lost their lives in the first incident (the largest loss of life in an explosion in South Africa) and the mine was almost totally destroyed underground following the resulting coal dust explosion. At Northfield Colliery 78 men were killed. The inquiry into this explosion was detailed and it is possible to

formulate accurately the successive events leading to the disaster (the recipe for a disaster).

The main points arising from blasting operations and explosives as ignition sources are described and precautionary measures to be adopted are suggested.

The table below depicts the number of explosions caused by blasting operations and explosives in relation to other causes from 1889 to 1988.

Table 4.1 Number of Explosions caused by Blasting Operations relative to other causes

	Number	Fatalities
Total incidents	138	761
Incidents due to blasting operations and explosives	48	427

Table 4.1 indicates that 34.8% of all explosions were as a result of blasting operations and explosives resulting in 56.1% of the total fatalities. Hardman (1988) states that the percentage of underground coal mined using explosives is currently 56% of the annual total. It follows that attention should be focussed on the safety aspects of this method of working.

4.2 THE LAG OF IGNITION OF METHANE

The ignition of a mixture of firedamp and air does not necessarily follow when a source of heat the temperature of which is higher than the ignition temperature of the mixture is introduced into it. An essential condition for ignition is that the source of heat, besides being of sufficient temperature, shall remain in contact with the mixture to which it is imparting heat during a sufficient length of time.

The reason for this is that there is a "lag" on the ignition of firedamp (methane). There is, indeed, a lag on the ignition of all inflammable gases, but it is more marked with methane than with some other gases; hydrogen, for example.

This lag on ignition has an important bearing on safety in coal mines. It accounts for the fact that it is very difficult, if not impossible, to cause an external ignition of firedamp by the gauze of a miner's flame safety lamp heated by firedamp burning within it; and it provides grounds for the belief that it should be possible to compound explosives the flames from which, being of exceedingly short duration, could not ignite firedamp despite their high temperature.

It is desirable, therefore, to study as fully as possible the phenomena accompanying the lag on ignition of gases; more particularly of methane, but not only of that gas, because the study of the behaviour of other and dissimilar inflammable gases provides a necessary aid to the interpretation of the behaviour of the one.

At comparatively low temperature, the combustion of methane is known to be a complicated process involving the step-wise

oxidation of the gas, but at high temperature the ignition can be regarded as a simple thermal process in which, providing the mixture is within the explosive range, a propagating flame is produced once a certain minimum volume (in the region of a cubic millimetre or so) is brought up to ignition temperature.

Some burning can be made to take place in methane/air mixtures of any composition but it is only within mixtures in a certain range of composition that a propagating flame can be produced. This range depends on the conditions but, near enough, the range runs from 5.3 percent to 14 percent methane in air at temperatures and pressures encountered in U.K. mines.

A detailed explanation for the existence of this inflammable range would be complicated but it is broadly explained by saying that at all stages of a propagating flame, the flame front has to produce enough heat to bring the ignited gas adjacent to it up to the ignition temperature; except within the range quoted, the heat released by the combustion is not enough to bring this about.

An important feature of the ignition of methane at other than very high temperatures is the delay between the first contact of the gas with the heated body and the spread of inflammation. At low temperatures this delay may reach 10 seconds but the data in Figure 1 are more normal.

Even at the temperatures of the electric arc (about 2 500°C or higher) there is still some delay, but it is much reduced, being in the region of microseconds.

Various reasons for this lag have been suggested and it is likely that there are several factors at work. The long lags at low temperature ignition are thought to be due to the slow build up of intermediate oxidation products that ultimately trigger off the inflammation of the methane. At high temperatures the delays are very short and although it is conceivable that the process of

ignition will still take place through the step wise oxidation of the gas, there must be some contributing to the delay by the time required to bring the required minimum volume of the gas up to ignition temperature; this time will be in the region of microseconds.

In the practical circumstances of mining, delay in ignition assumes importance in connection with ignition in some frictionel ignitions and in ignition from hot bodies, and is probably an important factor in the considerable increase in temperature required to ignite gas when the hot surface is freely exposed to convection or ventilation.

Various people have attempted to find the minimum temperature at which methane may be ignited. This temperature varies considerably with the conditions in which the determination is made, but ignition at hot surfaces can be fairly readily obtained at as low as 600°C to 700°C and ignition by adiabatic compression may take place at compressions producing temperatures no higher than 450°C. However, when a methane/air mixture is to be ignited by a hot surface freely exposed to convection ventilation, the minimum ignition temperature is in excess of 1 000°C. Figure 2 shows some typical results.

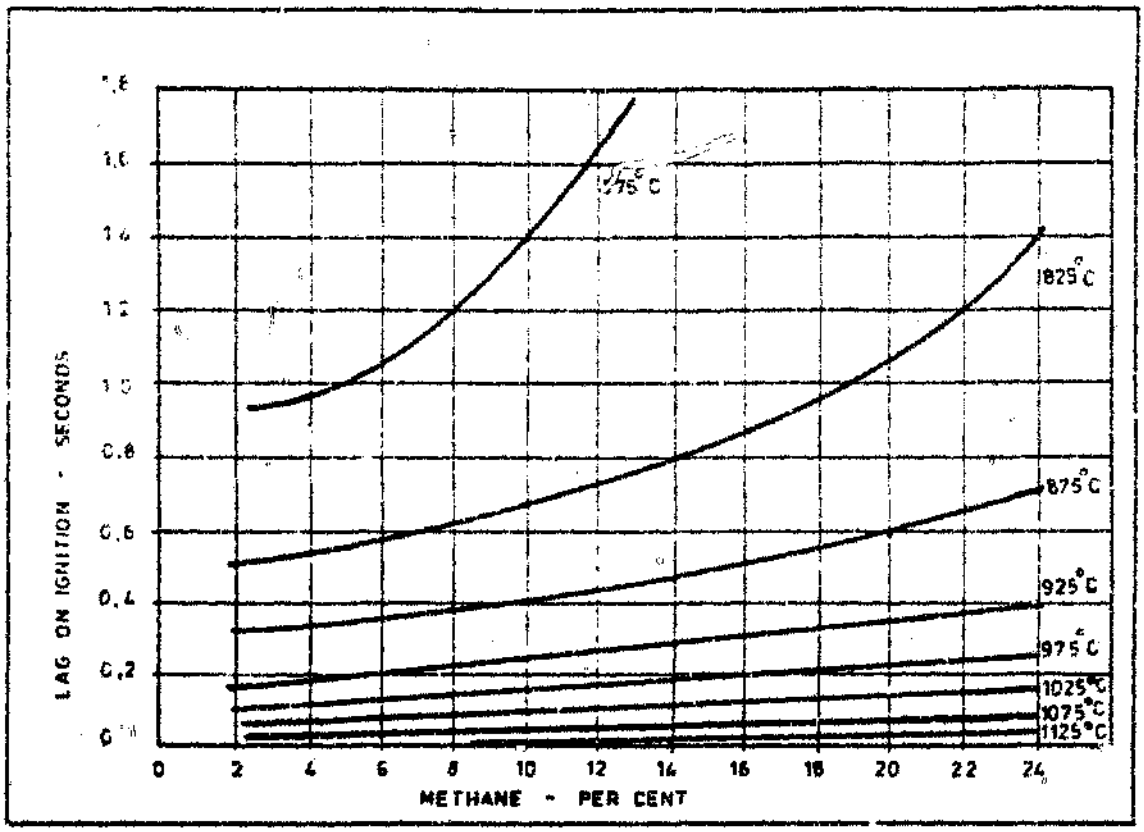


Figure 1
LAGS ON IGNITION OF METHANE AIR MIXTURES AT TEMPERATURES ABOVE THE IGNITION TEMPERATURE

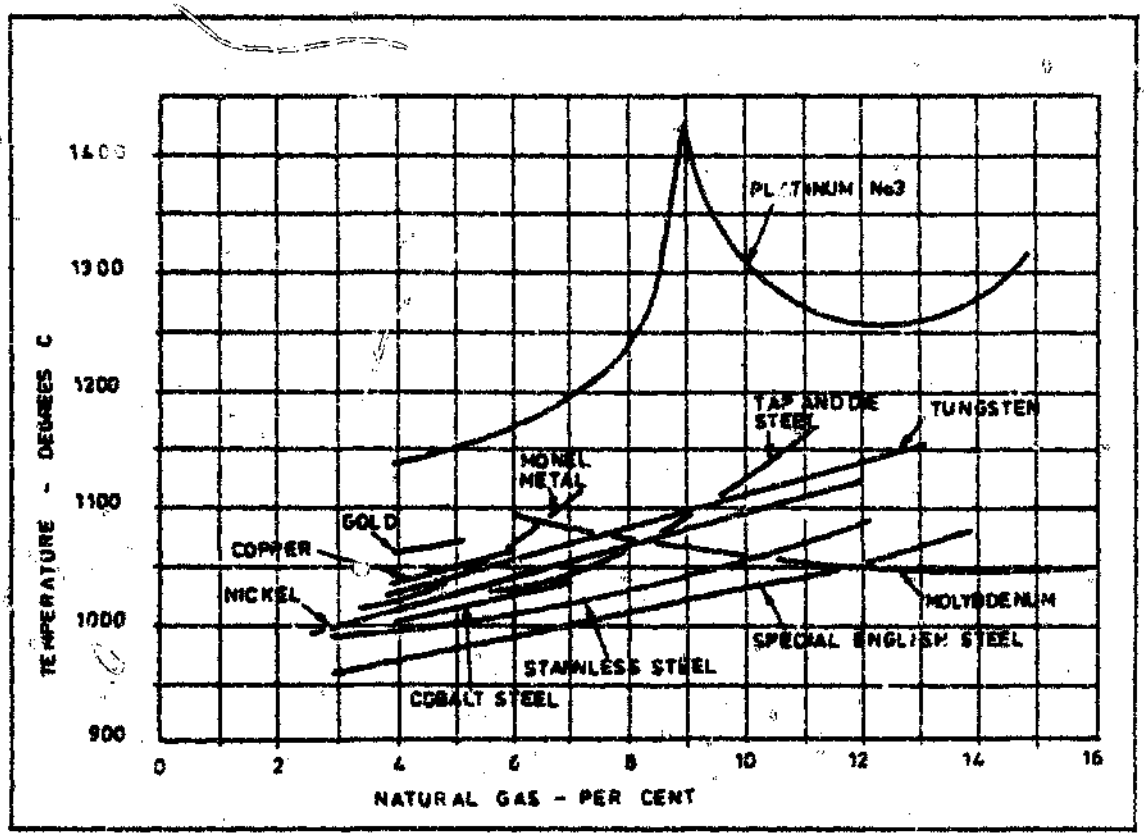


Figure 2
IGNITION OF NATURAL GAS BY HEATED BARS OF VARIOUS METALS

4.3 EXPLOSIVES

4.3.1 Definitions

It is important to differentiate between certain terms used when referring to the use of explosives.

a. Detonation

A detonation is an explosion travelling at a very high and constant speed (1 - 8 km per second) which is a characteristic of each explosive.

b. Deflagration

A deflagration is a combustion reaction propagating in the mass at a reduced speed (0,05 - 500 metres per second). It is generally described as a burning of the explosives. Deflagration of explosives increases the risk of gas or dust explosions.

c. Desensitisation

There are a number of ways in which desensitisation of explosives can occur. One must assume as a starting point that the product is loaded into the face in good condition. This implies that it is of the required sensitivity to initiation by a detonator, has the correct Velocity Of Detonation (VOD) and has an adequate sensitivity to propagation across an air gap (Ardeer Double Cartridge test (ADC)). Under normal circumstances such a product would detonate completely and adequately along the entire length of the column of cartridges in the borehole.

The two primary mechanisms by which high quality product can fail to detonate at full efficiency which have been identified are dynamic desensitisation, and desensitisation as a result of the channel effect.

Dynamic desensitisation is caused by an increase in the density of the product until it approaches or exceeds the critical density of the explosive. (The critical density is that density above which the explosive will not detonate). There are two possible causes for this. The first is as a result of an undetonated charge in the face (later delay) being shocked by the shock waves arising from the detonation of earlier firing charges (earlier delays). These shock waves move through the medium (coal in this case) and the high pressures associated with these shock waves have the effect of compressing the undetonated explosives with later delays. If these explosives are compressed beyond their critical density (this density being an intrinsic feature of the explosive) then a misfire will occur. The product has been dynamically desensitised.

It is also possible for this compression to result from hot gases at high pressure travelling through cracks in the coal and intersecting the undetonated charge. As mentioned above, the former mechanism (that of shock compression) is the most likely cause of dynamic desensitisation in South African coal mines.

Desensitisation as a result of the channel effect is also a common phenomenon in South African coal mining operations leading to reduced performance of the explosive, deflagration or even total failure. The channel effect is the result of a gap existing between the explosive and the borehole wall. An "air shock" travels ahead of the detonation from the point of initiation at the toe of the hole along the column of explosive. There is high pressure associated with this air shock and it has the effect of compressing the undetonated explosive ahead of the shock front (this is the leading edge of the detonation in the explosive) to densities approaching or exceeding the critical

density of the explosive. This induces a poor 'quality' detonation, deflagration or even failure. All exposives suffer from the channel effect to varying degrees, but permitted explosives are particularly susceptible as they are generally not well consolidated into the boreholes. Buckling the cartridges of a charge in a hole is sufficient to disrupt the channel effect.

d. Coupling

Coupling essentially refers to the extent to which the explosive fills the borehole along the length of the explosive charge. It is a simple concept defined by the volume of explosive along the charge length relative to the volume of the borehole along the same length. In section across a borehole, this relates to the area of explosive relative to the area of the borehole.

Obviously, the primary determinant of the extent of coupling is the selection of explosive diameter relative to borehole diameter. The secondary determinant would be the extent to which the cartridges are compacted into the borehole during loading.

e. Explosion

An explosion is a rapid, self-propagating chemical decomposition, accompanied by a large amount of heat and gas. It follows from this definition that should an explosion occur in an inflammable mixture of methane and air that a methane explosion is likely to occur.

f. Socket and Blown-out Shot

A socket is the remains of the end of a shot hole after being blasted. Generally the diameter of the socket is much larger than the original diameter of the shot hole as shown in Figure 23.

The length of a socket (x) could vary from 50 mm up to 50 cms. It is difficult to set a limit on the length of a socket since a blown-out shot hole may also vary from say 20 cms to 180 cms. A blown-out shot will generally result in a drill hole remaining at its original diameter. (The explosive having exploded out of the hole as it would out of a test steel cannon for example).

g. Shockwave

The shockwave is a compression wave in an elastic medium travelling in the air at supersonic speed. It is defined by parameters which are a function of the distance and of the quantity of explosive; the peak pressure, which is the maximum pressure of the wave above, the atmospheric pressure and finally the duration, which is the time required for an overpressure wave to return to atmospheric pressure.

4.3.2 Introduction

The manufacture and safe use of explosives to break coal underground is a complex subject and little is known about it by those whose responsibilities it is to use the product on a daily basis. The elementary training given to Miners and their box Attendants is reflected in the evidence of the many inquiries held by the Mines Department into explosions as a result of poor blasting techniques. The most common causes of blown-out shots, unsteady detonation and deflagration are:

- firing charges in fissured ground;
- centre priming of holes;
- inclusion of coal dust between cartridges;
- not using bottom priming;

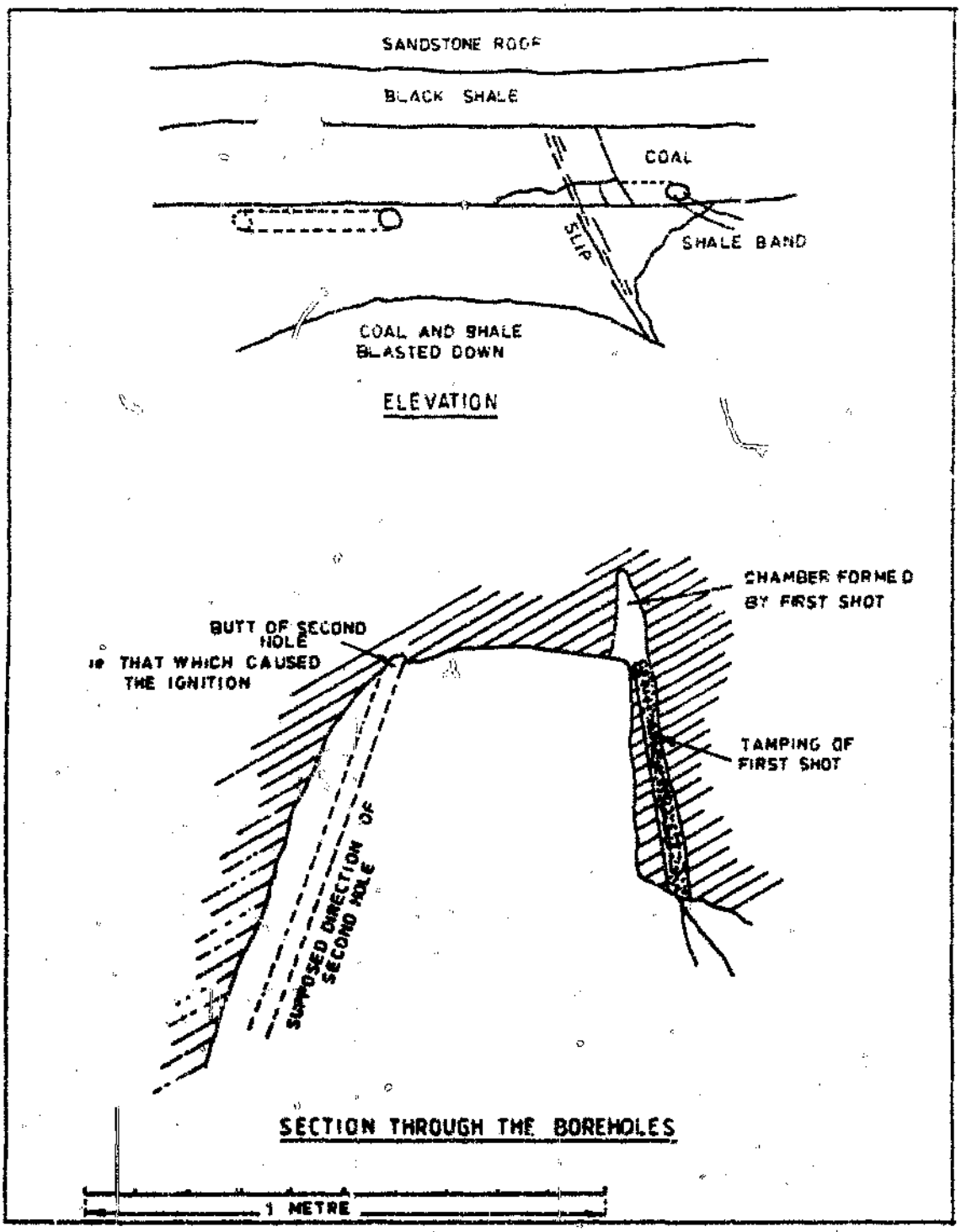


Figure 3

DIAGRAMS SHOWING THE REMAINS OF BOREHOLES
IN THE NORTH HEADING GLENCOE COLLIERY

- poor coupling of explosives (inadequate compaction);

- use of coal dust or wet clay as tamping or total lack of tamping;

- sympathetic detonation;

- mud blasting.

Figure 3 shows the effect of blasting a shot hole which chambered and then firing the second hole on the left hand side of the face.

The Commissioner of Mines for Natal states:

"When the end of the north heading was reached (after the disaster), charred timbers and coked coal were found, but it was clear the fire had not been a large one. Strong blowers of firedamp were issuing from the coal face. Observations confirmed the statements previously made by the Miner who fired the shot which commenced the series of explosions, that the shot was near a hole which had previously been fired, but had chambered and not broken the coal.

The entrance to the chambered hole was still standing with the tamping exposed and detonator wires hanging out, but the other shot had broken the coal away so that the chambered portion of the hole was exposed. The butt, about 12 mm deep of the second hole remained. Gas could be felt by the hand to be issuing in some quantity from the remains of each hole, while there was fine coal dust in the chamber of the first. The sketches given will illustrate the conditions as I observed them. According to the Miner three carbonite cartridges were used in the hole which caused the ignition.

The explosions taken together caused the deaths of 12 Whites and 65 Blacks."

The chief lesson which this disaster teaches is the danger of blasting when firedamp is present even though one of the best permitted explosives was used (carbonite). This short reference related to the Glencoe Colliery disaster in February 1908.

4.3.3 Development of Permitted Explosives

Until about 1880, gunpowder was the only blasting explosive and it was highly incendive to firedamp. Then high explosives were discovered and it was soon observed that they produced much less visible flame. This prime difference in behaviour between the two, stems from the speed of the reaction; gunpowder acts as a fast burning reaction whereas high explosives react detonatively, that is, the reaction takes place within a wave headed by a shock front moving through the cartridge at something greater than the speed of sound in the medium itself.

The detonative reaction with the resultant short duration flame confers a little safety but not enough. Conventional modern explosives owe their effectiveness and safety to a nice blending of ammonium nitrate with nitroglycerine in one form or another combined with a flame suppressant so that an explosive is produced with the appropriate heaving action accompanied by a flame suppressant atmosphere, the whole reaction proceeding as a detonation thanks to the presence of the nitroglycerine. The formulations are of course more complex than this and there are various additives to improve storage quality, to render the explosive insensitive to ambient temperature change, to confer freedom from fume in use, to adjust the density and so

Mining practice recognises four broad classifications of explosives suitable for use in coal mines:

I Explosives suitable for rock work in the absence of conditions likely to result in evolution of methane.

II Explosives sheathed with a flame suppressant suitable for use in coal and in rippings with appropriate safeguards.

III Explosives with a flame suppressant incorporated in the composition and permitted in the same situation as Class II. These are known as Eq.S (equivalent to sheathed) explosives.

IV A recently introduced group of enhanced safety explosives designed for delay firing in rippings and similar situations in which there is inherent a possibility of the charge firing into a gas-filled break or parting.

In addition, there is a new class under consideration for situations not yet finally specified but which will include delay firing in solid coal.

In general, the stronger the explosive, the more incendive it is and to some extent the problem facing the designer of explosives is that of finding the minimum strength that the user requires for good operational success and combining with it the necessary safety. Compared with the earliest explosives, modern formulations offer much improved safety for a given strength but the connection between strength and safety remains. In the recent past there has been a further improvement in this connection by the introduction of explosives employing exchanged-ion formulations (exchanged-ion is the description applied to explosives in which the chemicals ammonium nitrate and sodium chloride are replaced by ammonium chloride and sodium or potassium nitrate, that is, the ions are exchanged). These explosives have the property that they do not develop their full strength in the absence of full confinement with the result that dangerous situations arising from incomplete confinement (for example, firing in breaks, insufficient burden, and so on) are rendered less so by this autogenous weakening.

4.3.4 Nature of Ignitions

The more superficial question as to which of the phenomena (flame, hot gases, particles, and shock waves) are possible sources of ignition of firedamp has attracted most attention. It appears to be generally agreed that contact between the detonating explosive is highly conducive to ignition, especially if the explosive is partly confined as when it is in a parting in a strata, or in a corner. Some danger remains after the detonation, if the products then come into contact with the explosive mixture. A large flame is a sure sign of danger, but the size of flame of a safety explosive is not a measure of the danger. The subsidiary question as to whether the hot non-luminous products of an open flame are as dangerous as the still burning luminous products has not been answered. They must have equal intensities of energy until losses occur, but the greater thermal energy of the completely burnt products may be less effective than the combined thermal and chemical energy of the flame.

The shock wave of a mine explosive, by itself, may or may not be capable of initiating a self maintained flame in methane/air mixtures, but a mixture that has already been heated by the effects of a shot may be ignited by the delayed arrival of a shock wave reflected from a solid surface.

Projected particles of solid explosive which have escaped complete reaction in the detonation wave but are still undergoing reaction have been thought to be the main cause of ignition. Bigourd and Dangreaux (1973) in a publication on the mechanism of the ignition of methane by deflagrating explosives, observed, with motion pictures, that ignition of firedamp can occur immediately at the borehole mouth or later at an appreciable distance from the borehole, probably due to a thermal explosion of the unreacted explosive as it comes in contact with particles heated by the deflagration.

However, there is no doubt that flame, hot gases, projected particles of burning explosive and shock waves, can all contribute to an ignition of firedamp by an explosive. The relative contribution of each factor varies no doubt with circumstances.

It is well known that blasting has been a major source of ignition of gas and/or dust explosions in underground coal mines. The mechanisms by which explosives can cause ignition of methane/air mixtures are the following:

- By the hot gaseous products of reaction.

- By hot reacting particles of explosive escaping into the methane/air mixture.

- By direct action of the shock wave from the explosive.

4.3.5 Temperature of Ignitions

Hot particles from a detonation of explosives are of the order of 1000°C as are the gases which are emitted from this explosive.

Mallard and Le Chatelier (1883) studied ignition by explosives and their work led to the "French Doctrine of 1890". The postulate was that provided the detonation temperature of the explosive was not higher than 2200°C the cooling by the expansion as the explosive did its work would reduce the gas temperature to below that at which firedamp may be ignited. The Doctrine was good in that it led to the improvement of explosives, but it assumed that the firedamp was ignited only by contact with hot inert products of the explosion reaction and further work soon showed this to be but one of the ways in which ignition could be produced. The present view is that ignition may take place in one of three ways; these are:

- Ignition directly by flame or inert but hot gas.

- Ignition by compression.

- Ignition by solid particles. These may be particles of reacting explosive either at the cartridge or thrown out by the explosion and, in some cases, proceeding ahead of the shockwave front. Inert projected particles also may ignite gas either by being at a high initial temperature or alternatively brought to a high temperature by impact with an obstacle.

It is not known which of these is the most important or, indeed, whether any of them assume pre-eminence independent of the practical circumstances. However, it is certain that all three are ways in which ignition can be brought about and all should be taken into account both in usage and in laboratory examinations of new explosives.

Coward (1962) states that the firing of an explosive is accompanied by the almost instantaneous transformation of chemical energy into thermal energy, concentrated in a small space. The temperature of the products of an explosive, before they expand into the atmosphere, is 1500°C to 2500°C.

Mallard and Le Chatelier (1883) demonstrated that firedamp ignites at 650°C, a temperature which is always exceeded by explosives.

A recent discovery currently being investigated is the energy generated according to the ballistic missile tests. Where the energy generated is less than 67% the permitted explosives do not cause explosions whereas energy released above 67% causes explosions when the explosives are detonated in an explosive atmosphere.

Speeckhaert (1952) has conducted experiments which show the influence and role of inhibiting salts in anti-firedamp

explosives (permitted explosives). Indeed all permitted explosives used in South Africa today of the granular type contain a proportion of salt as an inhibitor.

Speeckhaert concludes that the alkaline salts and, in particular, the first of the halogen group are particularly suitable for this role.

Their heat of fusion and of vaporisation is not too high, but the heat absorbed by fusion and above all vaporisation is crucial, producing a good cooling capacity at the moment of gas release.

Speeckhaert proposes reasons (mechanisms) why other salts also work in absorbing energy (heat).

At the moment of detonation of the explosive in the mortar, the temperature being very high, a fraction of the heat is absorbed by the salt. This fraction corresponds with the specific heat of the salt added to the latent heat of fusion. However, because of the higher pressure, the heat of vaporisation cannot be absorbed and vaporisation cannot take place.

When the products of the explosion leave the mortar, if the temperature of vaporisation has been reached - as is the case with most salts except for refractory oxides - a new quantity of calorific energy is absorbed by this vaporisation and this brings down the temperature of the gases to the temperature of vaporisation, resulting in a cooling, the speed and scope of which depend upon the strength of the heat of vaporisation. It is to be noticed, in this case, the dimming of the flames which follow the shock waves.

As firedamp has a delay in ignition, it is necessary to eliminate these flames as completely as possible.

This makes it clear why alkaline salts and in particular the first of the halogen group are particularly suitable for this

role. Their heat of fusion and of vaporisation is not too high, but the heat absorbed by fusion and above all by vaporisation is crucial, producing a good cooling capacity at the moment of gas release.

Examination of variations in the power of the explosive mixture - using Trautzl's lead block test - shows that most salts decrease the strength according to their concentration (Figure 4).

Dolan (1960) has carried out extensive research into the suppression of methane/air ignitions by fire powders.

There are two functions of flame suppressant:

- Reduction in temperature of detonation.
- Material forms a dust cloud - affects the ignition characteristics of methane ahead of igniting source.

The efficiency of alkali halides due to low melting and boiling points and comparatively high latent heats of fusion and vaporisation is discussed.

Dolan carried out experiments to measure the effect of dust particle suspension on the velocity of combustion of methane.

The major physical factor of inhibiting material is the surface area of the powder exposed to the advancing flame. The ability of dust to reduce flame temperature below 1350°C depends on whether particle size is sufficiently small to allow heat transfer to take place.

Figure 5 shows the effect of the initiator on the flame velocities observed with upward propogating flames.

The quasi-detonations of firedamp caused by a gunned shot is best simulated by this type of initiation. The behaviour of these

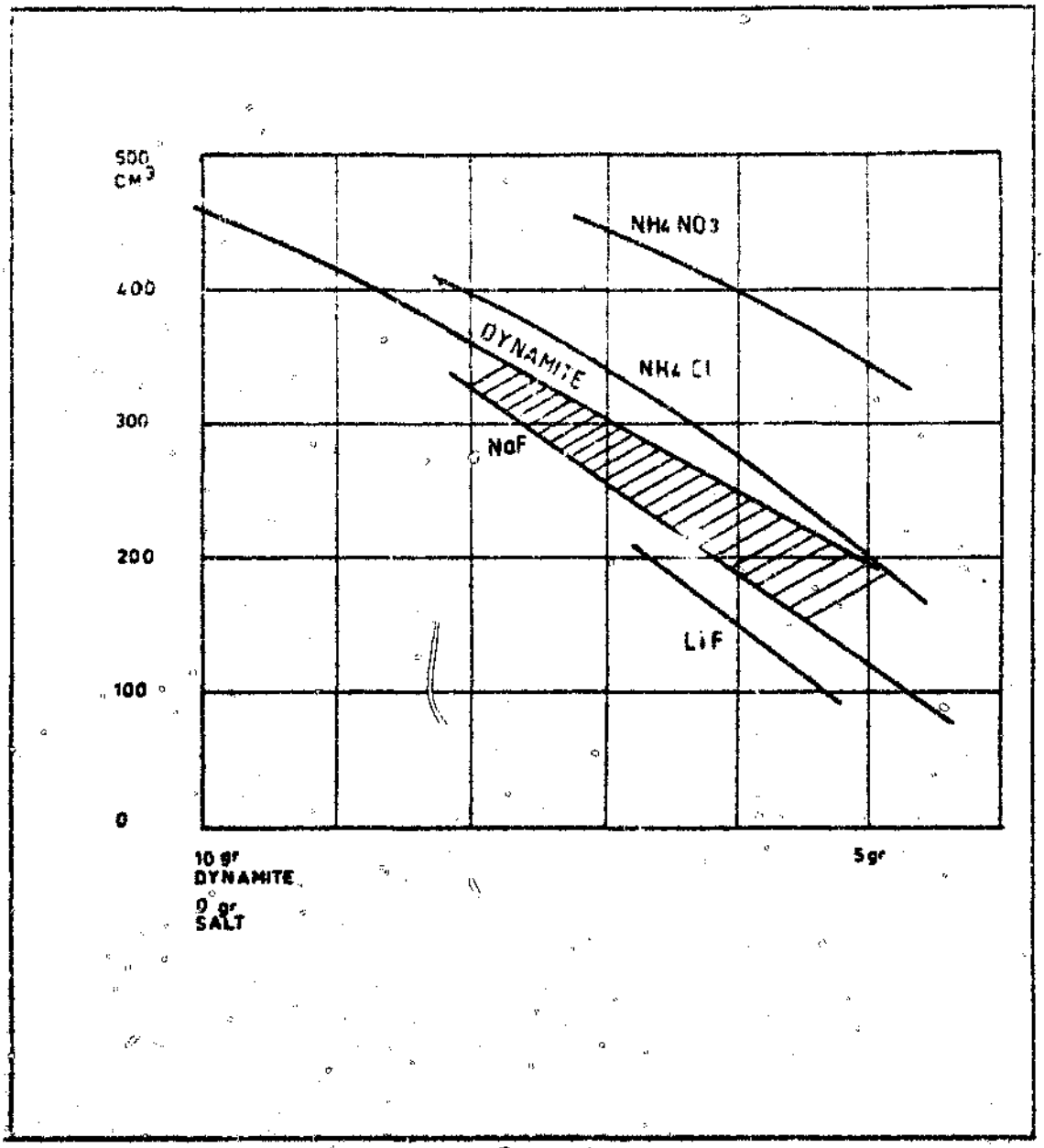


Figure 4

VARIATIONS IN POWER OF THE MIXTURES

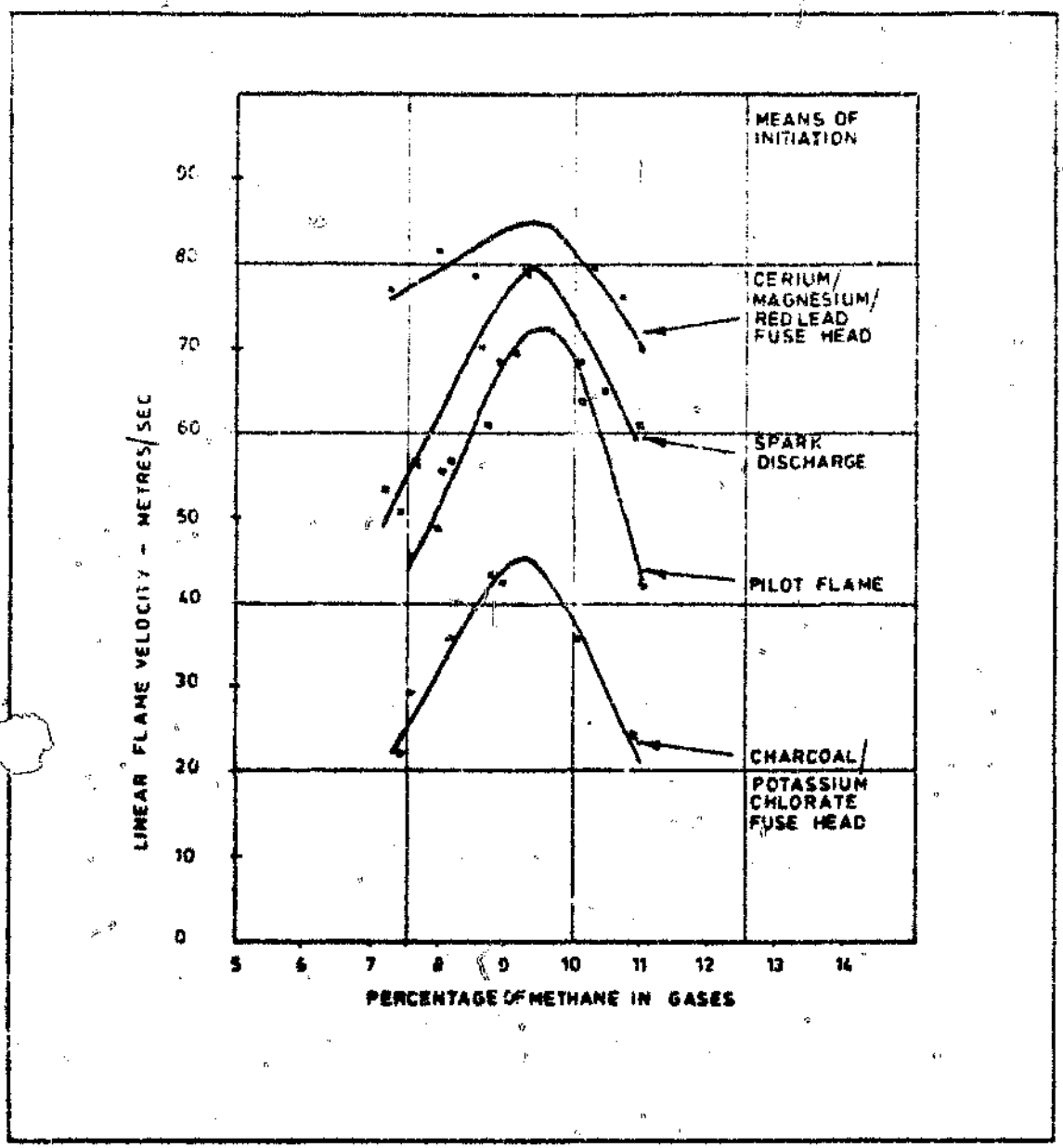


Figure 5

THE EFFECT OF THE INITIATOR ON THE FLAME VELOCITIES OBSERVED WITH UPWARD PROPAGATING FLAMES

accelerating flames under the influence of an inhibiting dust is therefore of great practical importance in assessing the value of a suppressant for inclusion in an explosive. Under the experimental conditions observed in this investigation, the spatial velocity of the flame in the explosion tube is dependent on the nature of the initiating source as is shown in Figure 5. The fastest flames were obtained with cerium fuseheads but the reproductibility was poor, owing to probable irregularities in the weight of fusehead composition. Much better regularity was obtained with the spark discharge and pilot flames, which were adopted as the standard methods of ignition.

When a non-inhibiting dust is introduced into the gas the effect on the flame speed is very limited. Some depression does occur, but even with heavy concentrations of dust the drop is seldom more than 20 percent.

Historically dynamites were the first permissible explosives to be used successfully in underground coal mines. In order to lower the temperature of the reaction products, salts such as NaCl, KCl or $CaCO_3$ have been introduced into their composition. Moreover the nature of the ingredients was adjusted so that the reaction of the oxidizer and the combustible is slow, for example, by using coarse grains of ammonium nitrate and sodium nitrate in the composition. In another method which is popular in Germany, some of the ammonium nitrate in the composition has been replaced by an equimolar mixture of ammonium chloride and potassium or sodium nitrate. During detonation the ammonium and the potassium ions are exchanged, for example;

$$NH_4Cl + NaNO_3 = N_2 + 2H_2O + 1/2O_2 + NaCl$$

Thus, a flame shortening smoke of very fine salt particles is produced by the decomposition reaction itself. Combinations of the two previous kinds of permissible dynamites are possible.

4.3.6 Comparison of Blasting Effects of Direct and Indirect Ignitions

Koga et al (1986) srate that in order to promote safe blasting in coal mines the blasting must be planned, starting with the method of initiation. The placing of the primer cartridge at the rear of the shothole ensures that the release of the energy upon firing is directed outwards and downwards thereby ensuring a better fragmented blast. When the primer cartridge is placed on the middle or the front of the hole, there is a greater chance of the energy being released in three directions - front, backwards and downwards, for example, with resulting poor fragmentation and the ignition of methane by the detonator cartridge. See item 4.3.8.

4.3.7 Observation of Explosion Flash

Three types of boreholes, single boreholes, 2 parallel boreholes and 2 acute angle boreholes (these boreholes are not used under normal circumstance; an abnormal blasting situation has been simulated in which the central axes of the boreholes formed an acute angle due to misalignment of the direction of the borehole bit) of length 1100 mm and diameter 40 mm were drilled in the advancing tunnel in combined sandstone and shale seams of the mine shaft as shown in Figure 6. These holes were then charged with 300 g or 400 g of No. 1 Tokubai dynamite without any stemming, and blasted using direct or indirect initiation. The explosion flashes were observed by taking still photographs. The length of the explosion flash was obtained by placing a scale parallel to the direction of the axis of the borehole before blasting. The scale was then photographed from the same place as the explosion flash and used as a standard for the measurements. The camera stop was set at 5.6 and the photographs were taken with Fujicolour HR100, the shutter being opened at the moment of initiation.

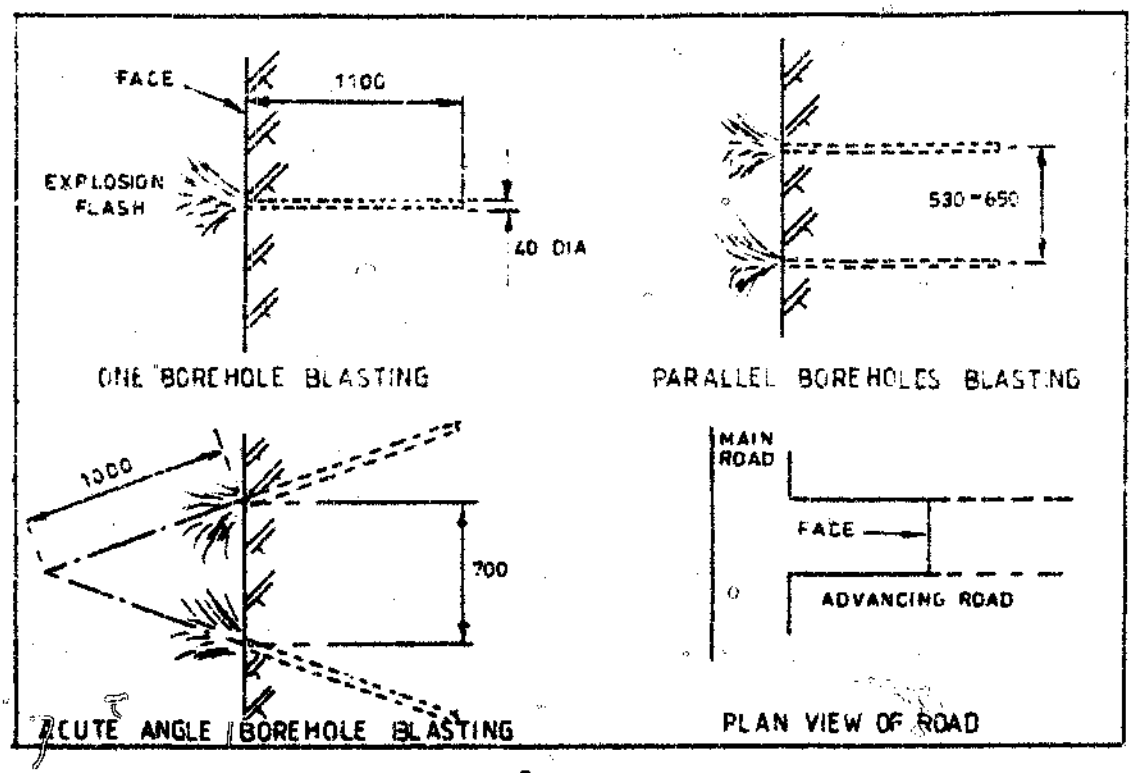


Figure 6

BLASTING PATTERN

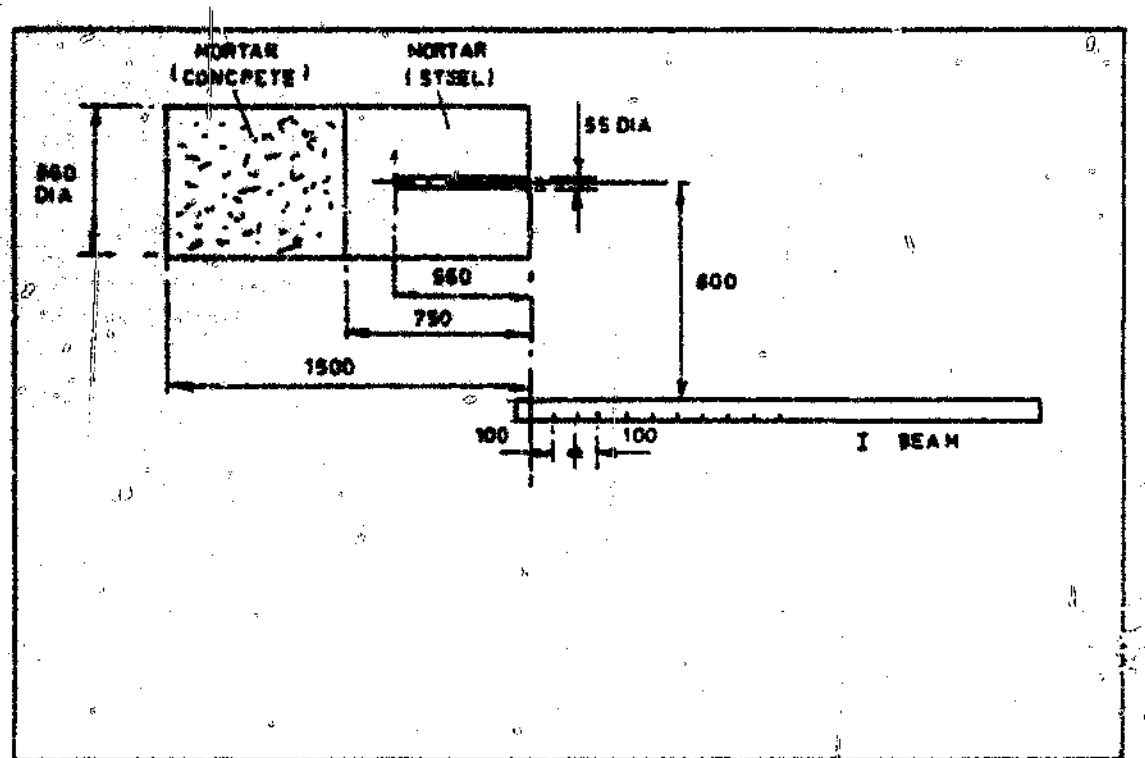


Figure 7

EXPLOSION FLASH OBSERVATION

ALL DIMENSIONS mm

The explosion flashes in steel mortars and concrete mortars were observed by taking still photographs while exploding 400 g of No. 1 Tokubai dynamite. The steel mortar is as shown in Figure 7 (concentric mortar, diameter 560 mm, length 750 mm, with a central blasting hole of 55 mm diameter and 560 mm depth) and the concrete mortar was the type used in the safety trials, made of 1 part sand to 1 part cement (only the blasting hole was of the same dimensions as the concentric mortar). The length of the explosion flash was measured by positioning an I-section beam with gradations of 100 mm spacing parallel to the axis of the blasting hole and at a distance of 800 mm from it, and photographing it at the same time as the explosion flash. The camera was positioned 5 m from the mortar blast, the stop was set at 5.6, as for photographing in the rock tunnel, and the shutter was opened at the moment of the initiation.

The measurements of the length and width of the explosion flash were based on the shape shown in Figure 8.

Table 4.2 shows the details of the length of explosion flashes obtained from experiments conducted in steel and concrete mortars and in an actual solid rock face.

Table 4.2 Length of Explosion Flash

No	Explosives	Weight (g)	A	B	C		D		E		F		
					a	b	a	b	a	b	a	b	L
1	K	400	DI	1	40	41	38	43	37	41	15	27	–
2	K	400	II	1	105	160	80	150	70	120	35	105	–
3	K	300	DI	2							18	20	93
				PB							21	35	53
4	K	300	DI	2							23	28	102
				AB							37	37	88
5	K	300	II	2							43	107	70
				PB							60	85	77
6	K	300	II	2							38	56	105
				AB							45	83	102
7	K	300	DI	2							21	20	99
				AB							23	23	110
8	K	300	II	2							87	115	110
				AB									102

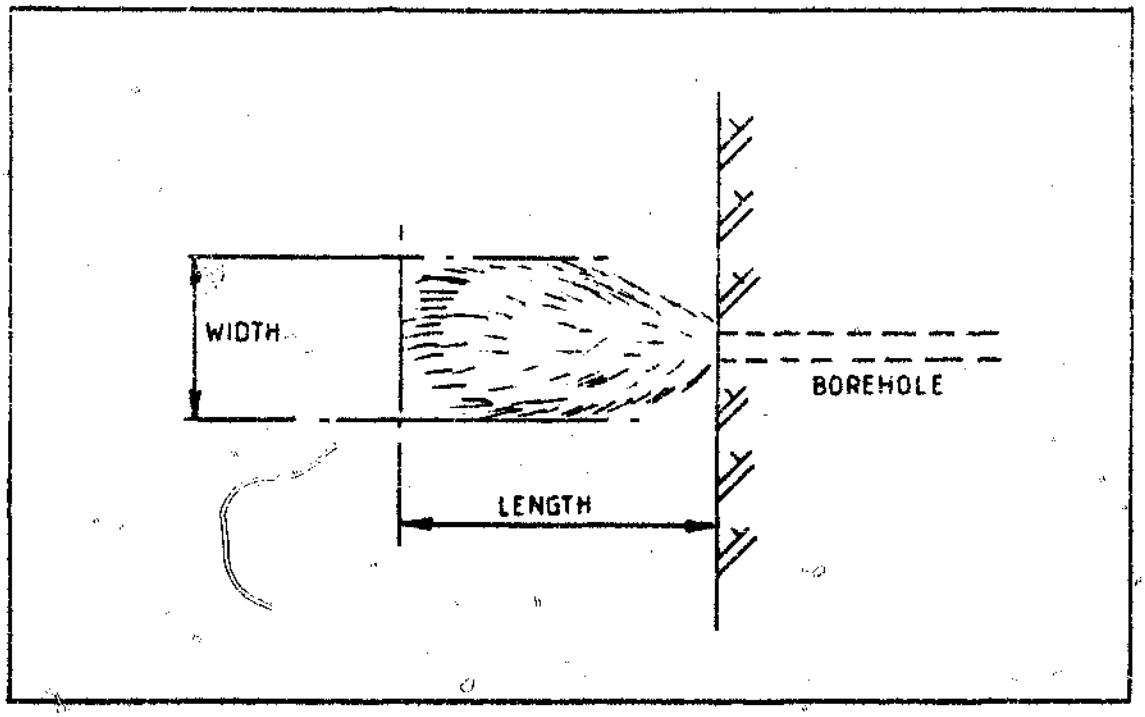


Figure 8

METHOD OF MEASURING THE EXPLOSION

K: Gelatine dynamite (permitted)
A: Initiation method
B: Number of blasting boreholes
C: Steel mortar
D: Concrete mortar
E: Smaller concrete mortar
F: Practical rock
a: Width of explosion flash (cm)
b: Length of explosion flash (cm)
L: Remaining length of borehole (cm)
DI: Direct initiation
II: Indirect initiation
PB: Parallel boreholes
AB: Acute angle boreholes

4.3.8 Various Methods of Initiation

Koga et al[9] (1986) have the following to say about initiation. "Indirect initiation was found to be a better method of initiation than direct initiation.

- There was no residual explosive.
- There were few cut-offs.
- The blasting effect was more satisfactory than with direct initiation."

Wallace and Ireland (1987) support the views that indirect (bottom) initiation is thought to be safer from the point of view of cut-offs and residual explosives. The breaking ability of the explosive is also generally considered to be better when indirectly initiated.

It is abundantly clear that the practice of mid-priming, which requires the detonation to run both ways from the initiator, places the greatest demands on the explosive for reliable performance. This is borne out by studies using X-ray photographic techniques conducted by Kennedy. These have shown that small increases in density of permitted explosives are more

likely to lead to failure in the reverse than the forward direction.

4.3.9 Factors which adversely affect the Behaviour of Permitted Explosives

In almost every instance in all the explosions initiated by explosives and blasting operations, one or more factors relating to the incorrect use of explosives is evident. These factors have been highlighted in the two incidents dealt with in this chapter (Durban Navigation Colliery No. 2 Pit and the Northfield Colliery disasters).

Deflagration Tests

The ease with which the above permitted explosives can be induced to deflagrate was studied by means of the "deflagration cannon" as described by Plant and Barbero (1969). It consists of a thick walled steel tube 57 mm in internal diameter (Figure 9), sealed at one end and restricted at the other by a disc containing an aperture through which the gases vent. The degree of confinement in the tube can be altered by choosing hole sizes in the disc between 1,6 and 9,5 mm in diameter. The charge consists of two explosive cartridges each of mass 56 g separated by a plug of coal dust. This assembly is supported centrally in the tube. One cartridge (the "donor") is primed with a No. 6 strength copper-cased detonator. The occurrence of deflagration in the "receptor" cartridge under various degrees of confinement is indicated by the presence of ash.

The channel effect and its influence on the performance of permitted explosives has been studied by means of the double pipe test. In this test the explosive cartridges are placed in a mild-steel pipe simulating a borehole (Figure 10). They are arranged to represent as far as possible the realistic conditions of use, for example with coal dust between them. This pipe is

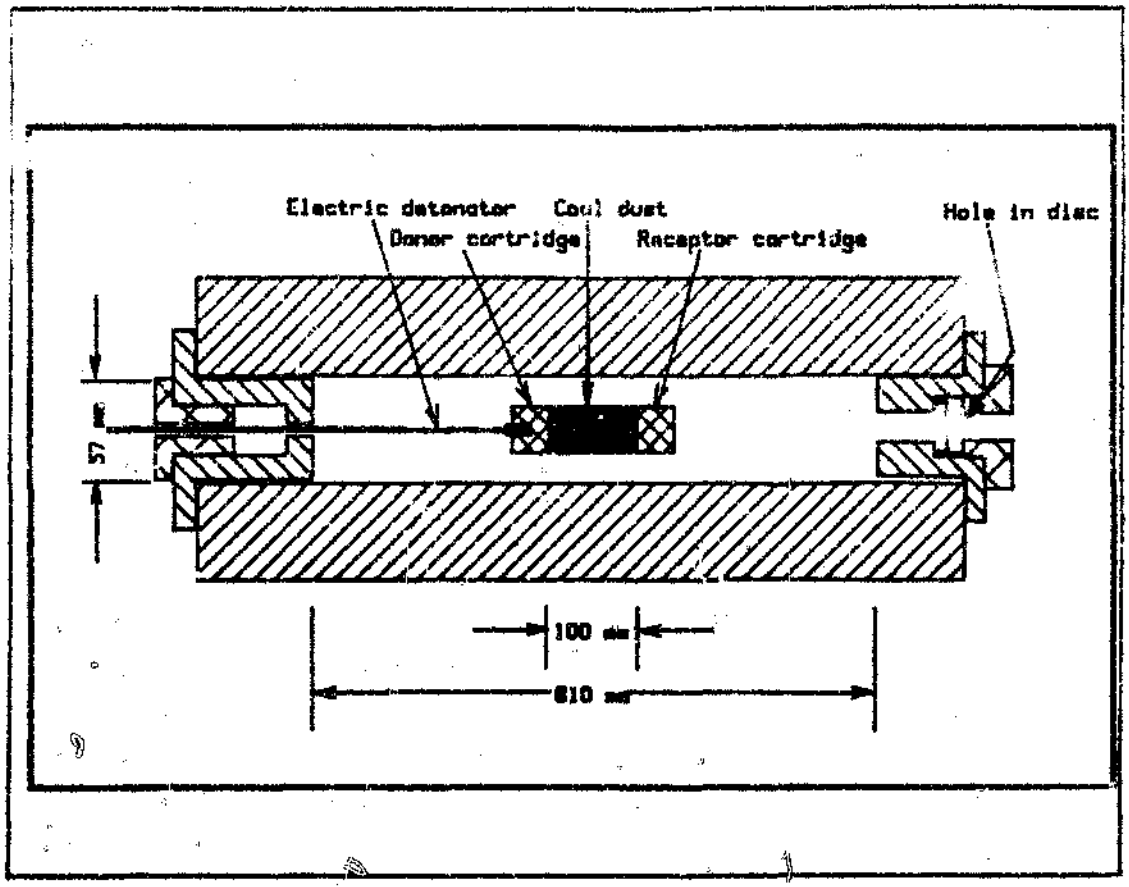


Figure 9

SCHEMATIC DRAWING OF THE DEFLAGRATION CANNON

placed on top of a length of hollow bar which acts as a witness pipe which is then sandwiched between the test pipe and a heavy steel anvil. Detonation of the explosive charge results in a dent along the length of the witness pipe, the depth of which correlates with the peak pressure generated.

Channel Effect

All the P1 explosives tested suffer from the channel effect as shown in Figures 11, 12 and 13. The peak borehole pressure (as measured by the depth of dent) declines along the length of the charge as the air shock becomes established. This generally occurs in the third cartridge from the point of initiation.

The ratio of the volume of the explosive to that of the borehole (i.e. the degree of coupling) will influence the intensity of the air shock and hence its effect on the explosive. In tests conducted in which the diameters of the top pipe (34 mm) were chosen to provide a snug fit to the cartridges (32 mm), no trace of desensitisation was detected as evident by the uniform depths of dent in Figures 14 and 15. To simulate unconfined detonation where no channel exists, tests were conducted with the explosives laid in "open" pipes as in Figure 10. The performance of the first cartridges in the charge is clearly sustained as illustrated in Figures 16, 17 and 18.

Effect of Consolidation

To ensure optimum performance of permitted explosives it is important that the charges be pressed home as far as possible while remaining within the constraints of the law. The dent profiles of Figures 19 and 20 show that consolidation of the test charge in the pipe significantly reduces the channel effect. (Even relatively mild consolidation of cartridges, so as to buckle rather than squash them, has been found to effectively eliminate the channel effect). It has been established that the air shock can be disrupted and its effect significantly reduced

merely by fitting cardboard spacers between the cartridges and the borehole wall.

Gaps in an Explosive Column

Carelessness in the loading of boreholes is a potential cause of problems, leading to low-order detonations, deflagrations or misfires. Poor charging practices can lead to gaps between cartridges in explosive columns. These include pushing cartridges individually to the back of the hole rather than as a 'train', or pulling the detonator wires whilst loading the hole. The latter is particularly likely if the primer cartridge is at any position in the explosive column other than at the toe of the hole as recommended. These gaps may be empty (air gaps) or filled with coal dust. Since gaps filled with coal dust are potentially more serious as an interrupter of detonation, they were studied by the use of the double pipe test. (See Northfield Colliery disaster, 1943).

Cartridges of coal dust of the same diameter as the test pipe were prepared (40 mm). As a thorough test of whether the detonation could jump a chosen gap, the gap was repeated between all cartridges in the column. The detonator was therefore placed in a cartridge in the middle of the explosive column, so that detonations running forward and backwards from the detonator could be observed. Figure 21 depicts the results of such a test conducted on COALEX 1 in which the cartridges immediately behind the initiator detonated but at such low order that it was unable to propagate across the next pocket of coal dust.

Dynamic Desensitisation

The resistance of some permitted explosives to dynamic desensitisation has been studied in the underwater test. In this test the charge, protected from water penetration is fired under 6 m of water. The gas energy of the explosive is derived from measurements of the dimensions of the bubble of product gas using pressure gauges.

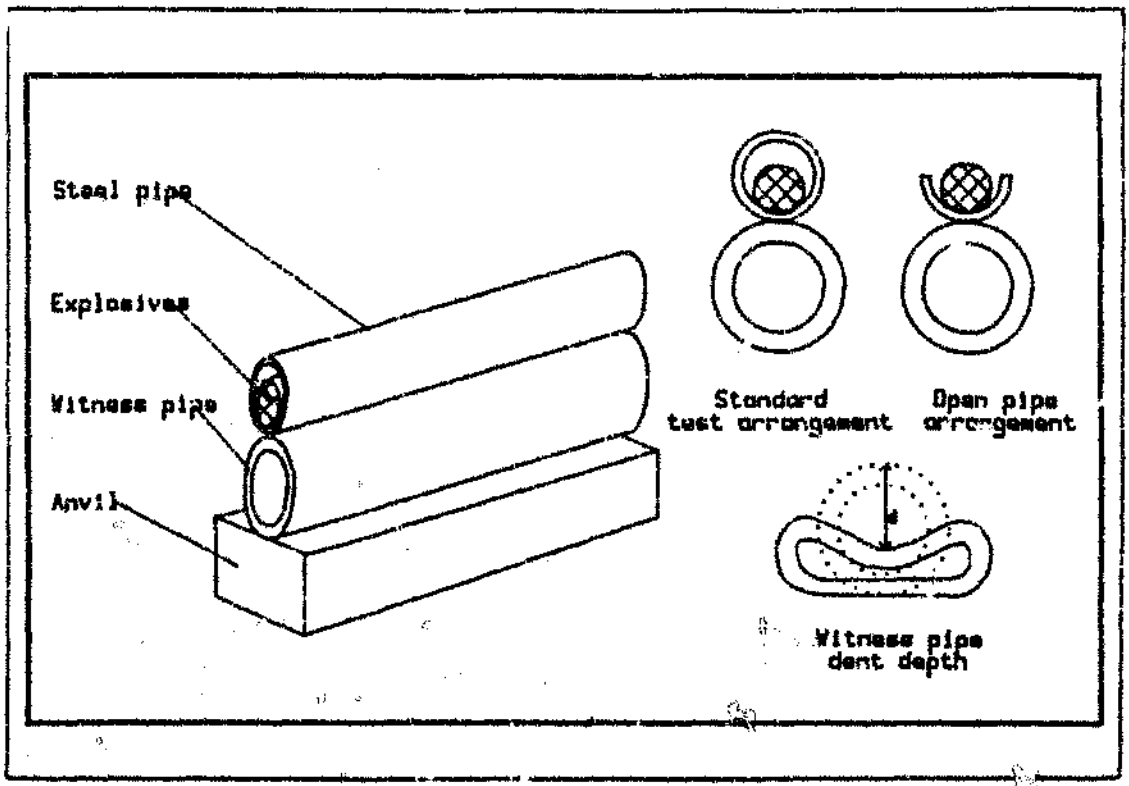


Figure 10

THE DOUBLE PIPE TEST

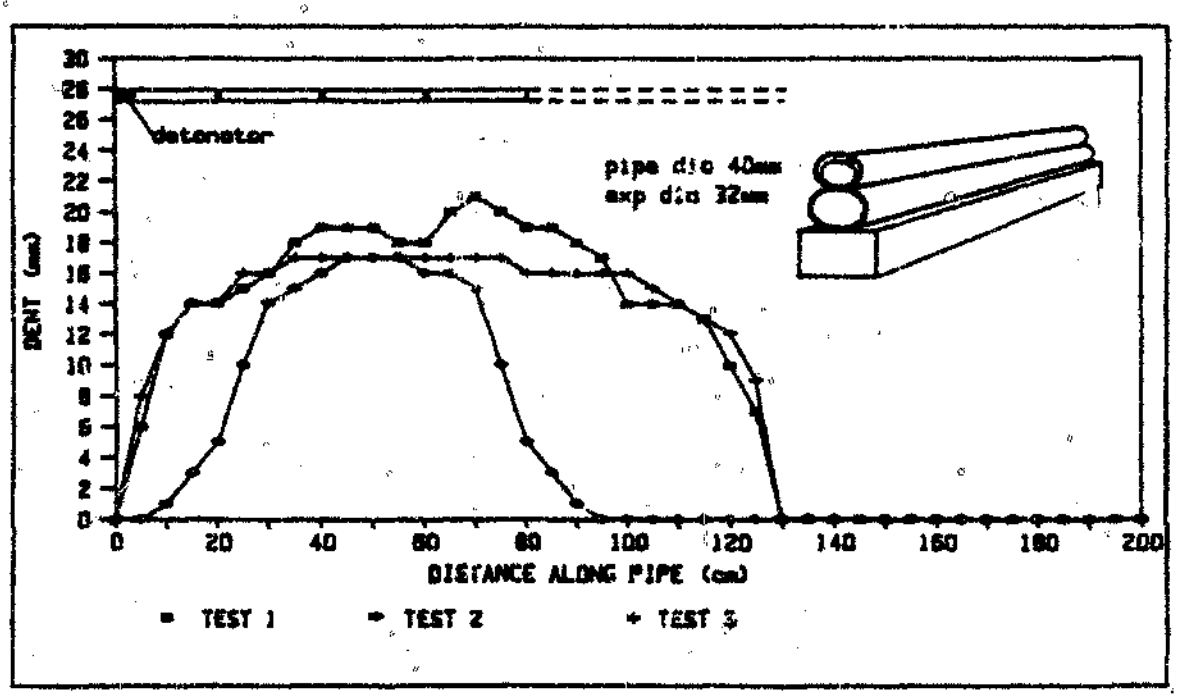


Figure 11

CHANNEL EFFECT - COALEX 1-32x200

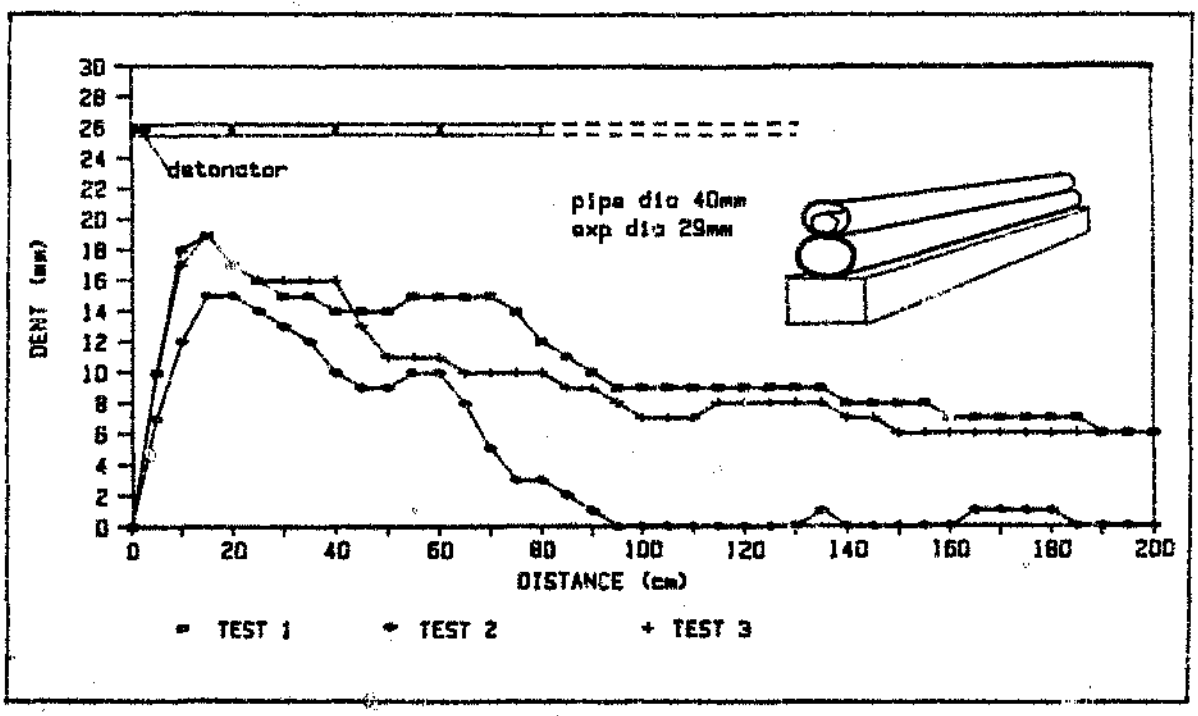


Figure 12

CHANNEL EFFECT - AJAX - 29 x 200

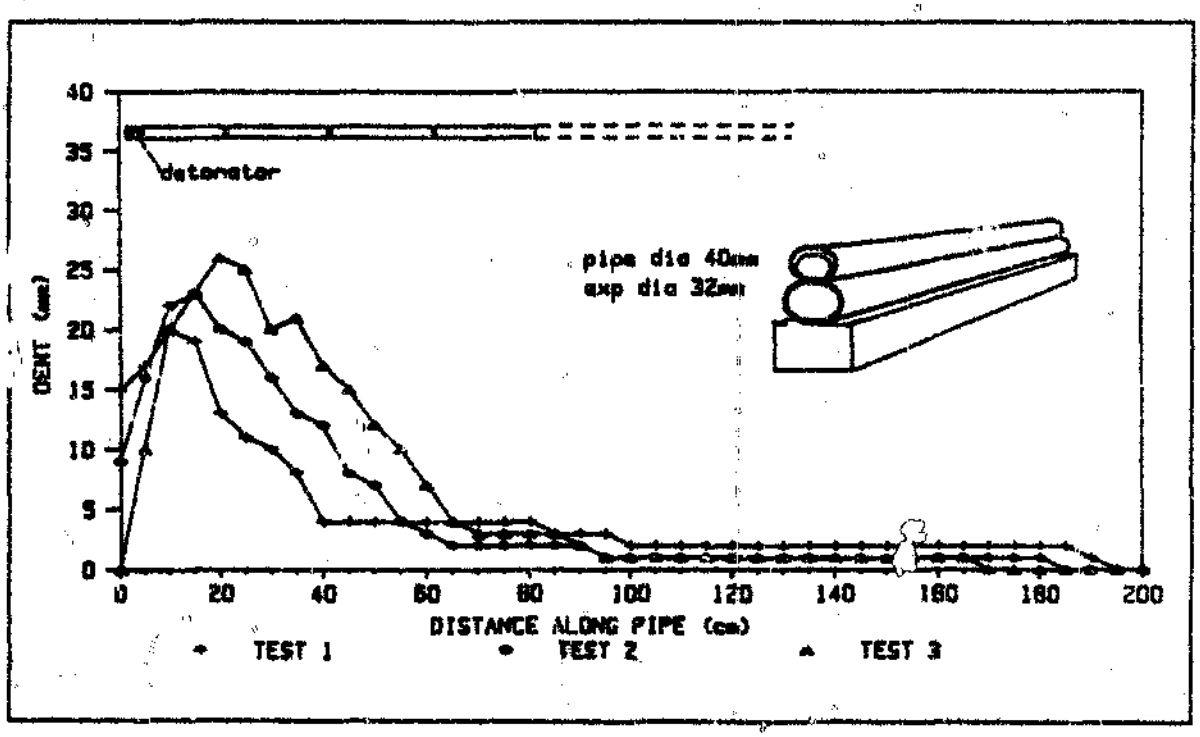


Figure 13

CHANNEL EFFECT - ENERCOAL - 32 x 200

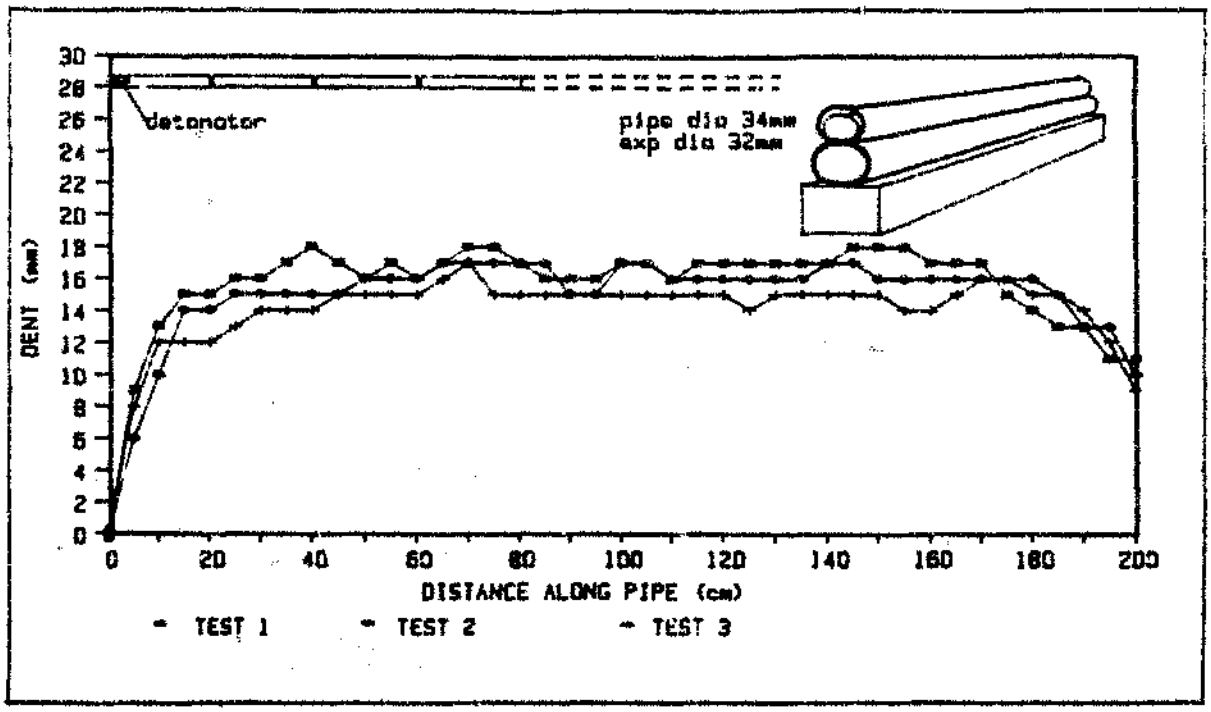


Figure 14

COALEX 1 32x200

EFFECTS OF ENSURING A 'SNUG' FIT BETWEEN THE CARTRIDGES AND THE TOP PIPE

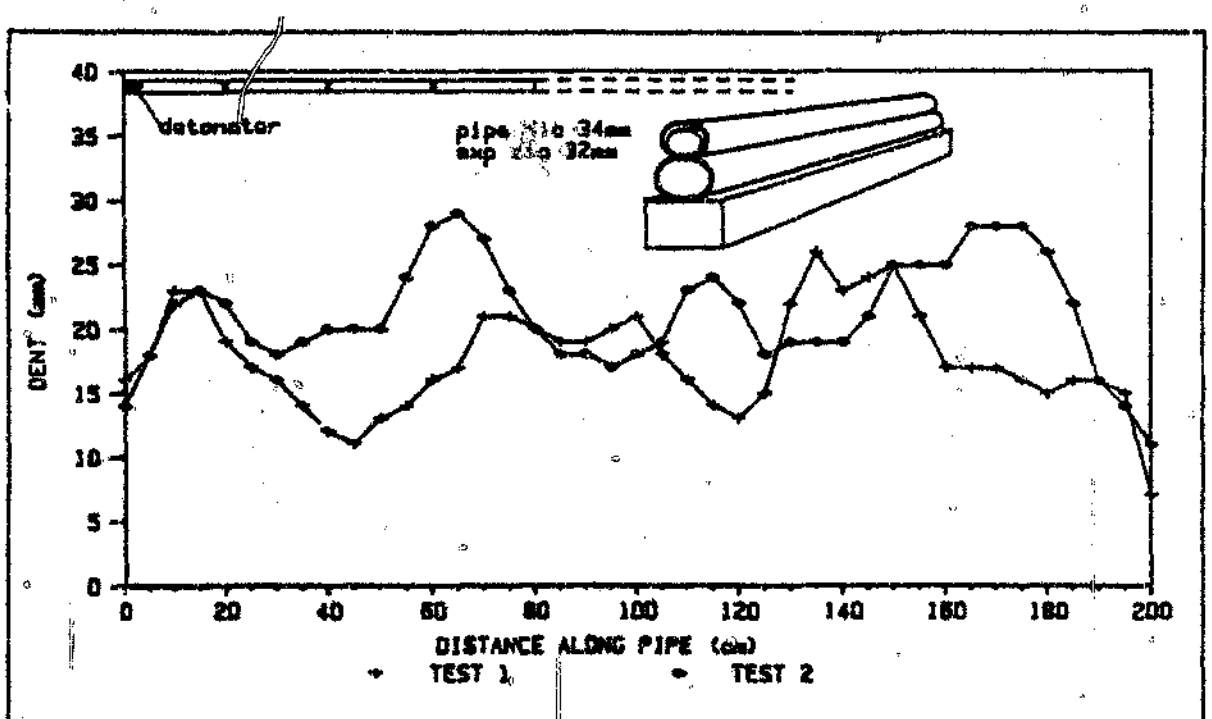


Figure 15

ENERCOAL 32x200

EFFECTS OF ENSURING A SNUG FIT BETWEEN THE CARTRIDGES AND THE TOP PIPE

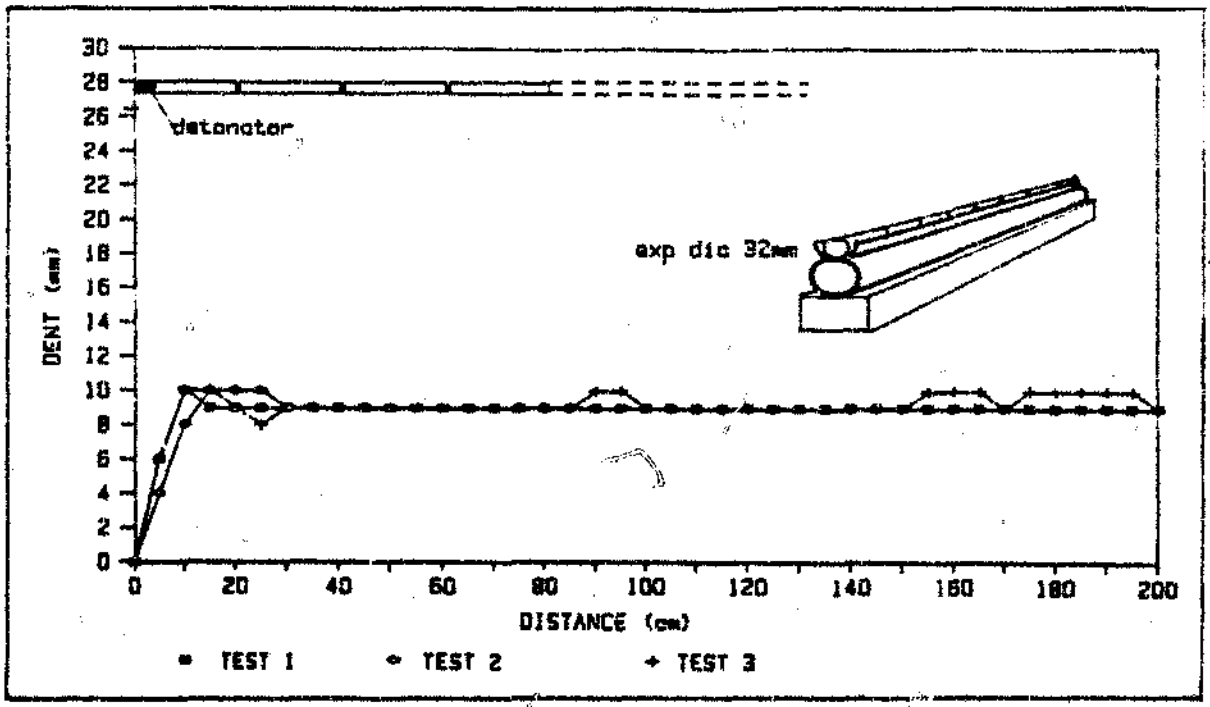


Figure 16
COALEX 1 - 32x 200
DENT PROFILE OBTAINED WHEN THE CHARGES
WERE UNCONFINED

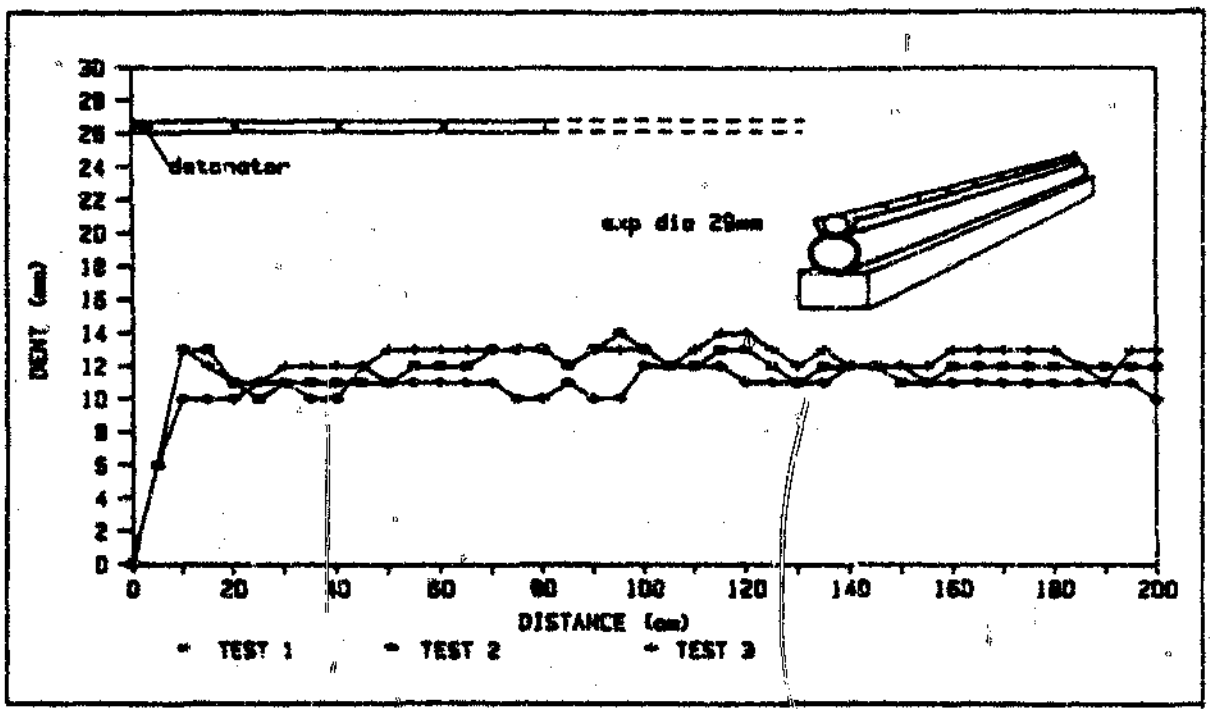


Figure 17
AJAX 29x 200
DENT PROFILE OBTAINED WHEN THE CHARGES
WERE UNCONFINED

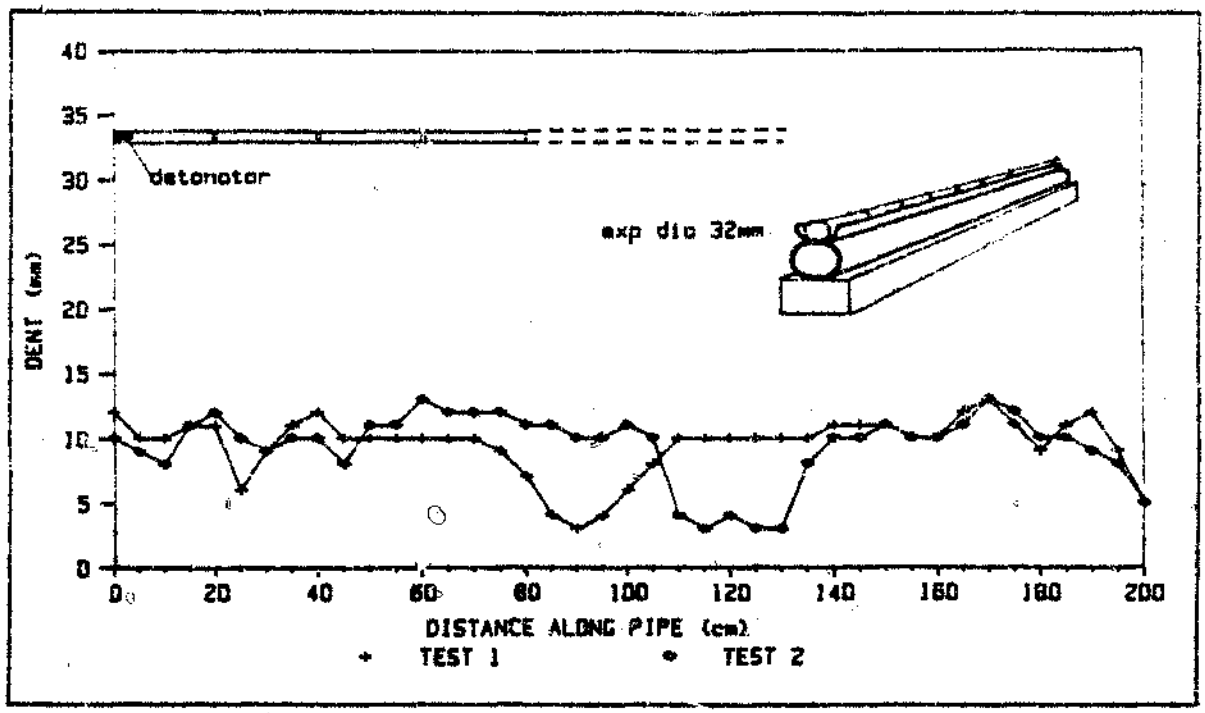


Figure 18
ENERCOAL 32x200
DENT PROFILES OBTAINED WHEN THE CHARGES
WERE UNCONFINED

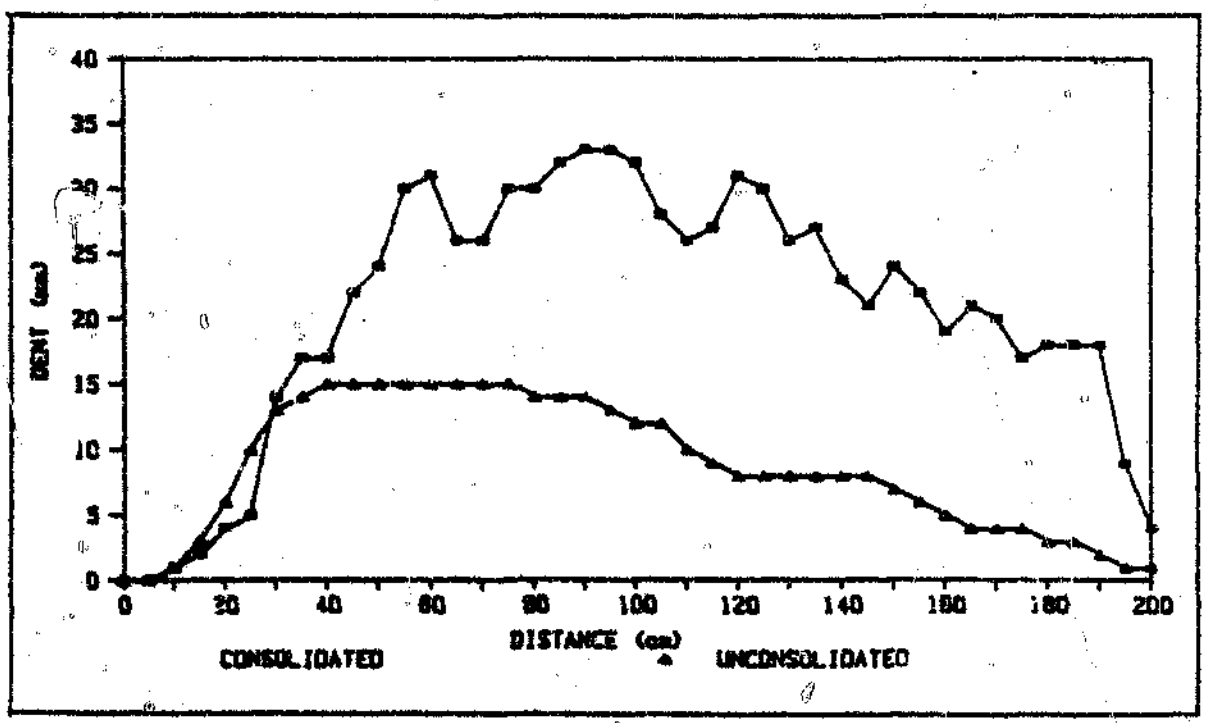


Figure 19
EFFECT OF CONSOLIDATION - COALEX 1-32 x 200
TOP INITIATION

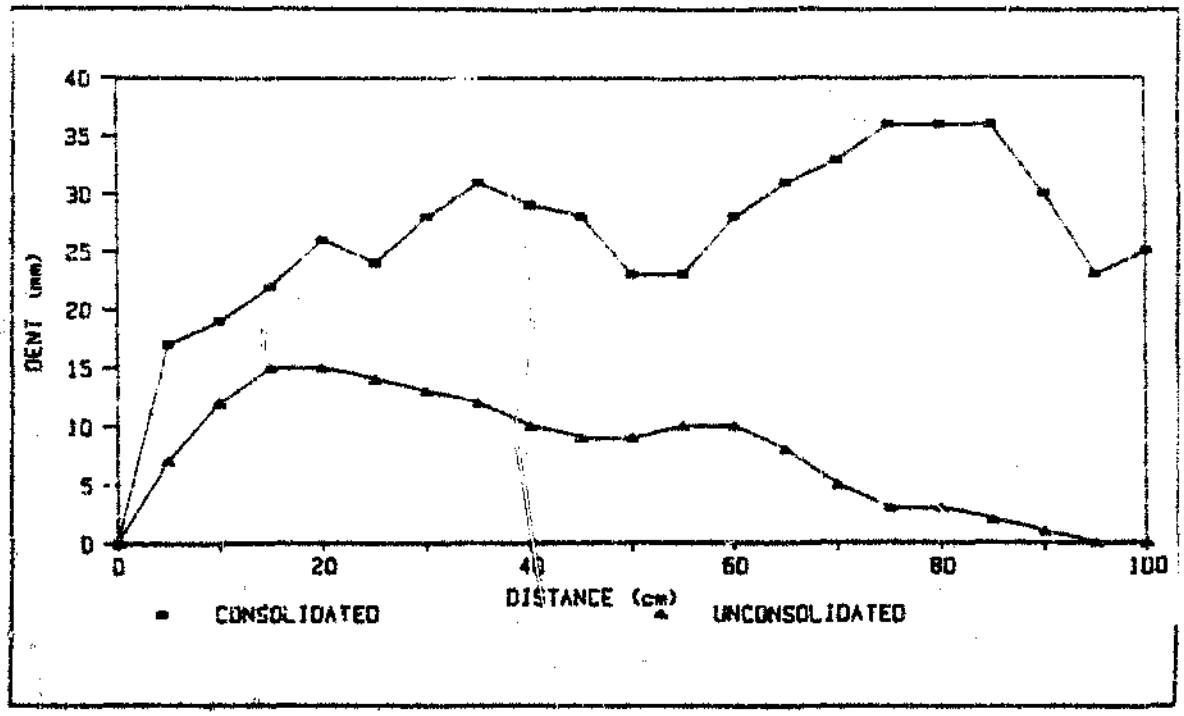


Figure 20
EFFECTS OF CONSOLIDATION - AJAX 29 x 200
TOP INITIATION

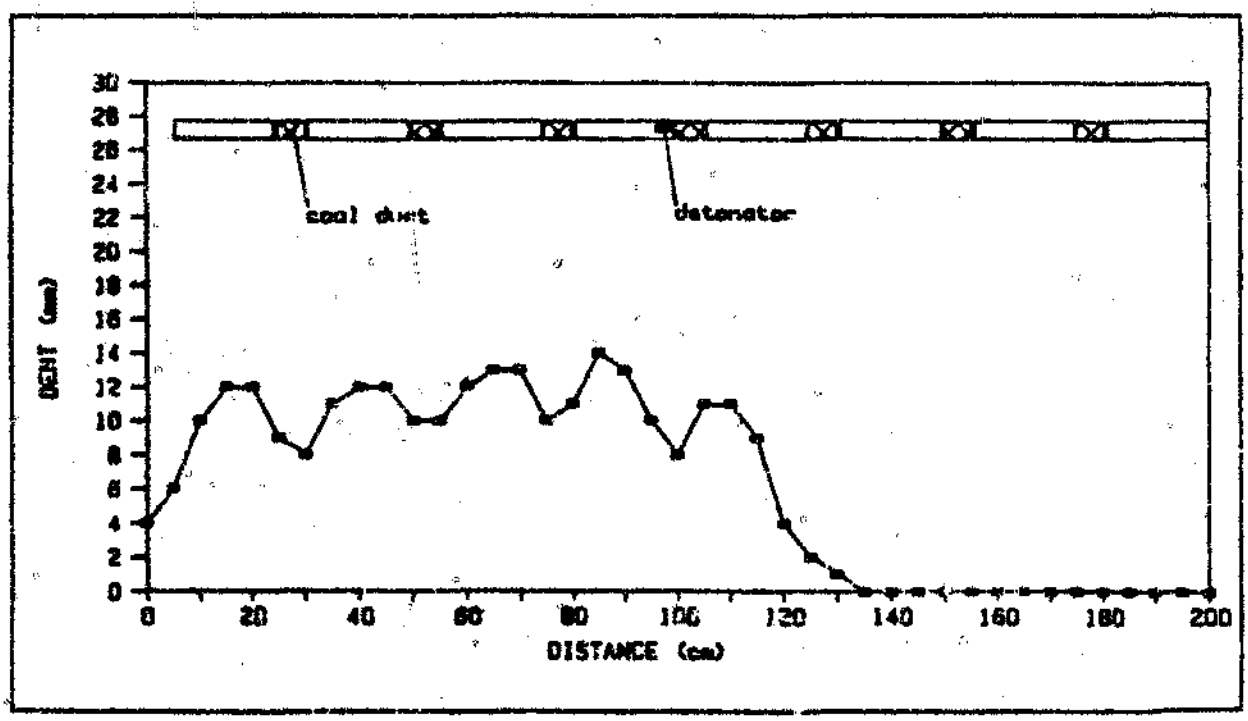


Figure 21
COALEX 1 - 32 x 200
EFFECT OF 50 mm POCKETS OF COAL DUST
BETWEEN CARTRIDGES
CENTRE INITIATION

In the case of AJAX and COALEX 1 shock desensitisation can play a significant role in reducing the energy output of the charge. This can be attributed to compression of the explosives to densities approaching or greater than their critical densities. The watergel, ENERCOAL is unaffected by shock pressures in this regime. This is due to the fact that the gas bubbles in this composition rapidly expand back to their original volumes after compression by a shock front.

To obtain the optimum performance of the explosive it is necessary to match the diameters of the hole and the explosive cartridges as closely as possible and to prevent coal dust collecting between cartridges. When using soft explosive compositions such as watergels, the air gap should be disrupted by consolidating the explosives in the hole. In certain European countries the cartridges are placed in a cardboard sleeve, primed and inserted as a single unit into the hole. This technique, employed with great success, serves to prevent obstructions between cartridges, disrupts the channel effect and ensures bottom point initiation.

The ignition of explosive when subjected to hot fumes at high pressure takes place very rapidly. In practical shotfiring the occurrence of such deflagration might be indistinguishable from normal detonation, apart from reduced performance, and will not present a serious fire or gas ignition hazard. The occurrence of deflagration that is recognisable as such is the relatively slow burning of explosive in broken strata. It is essential that this be prevented by the adoption of the correct charging techniques.

Centre priming, aggravated by the inclusion of coal dust between cartridges presents the single greatest hazard in terms of deflagration of permitted explosives. Bottom priming will not eliminate desensitisation of the explosive by the channel effect, but it will ensure that desensitised cartridges will not retain their integrity and remain deflagrating on the face after the blast.

4.3.10 Emulsion Explosives

Recent studies have shown that the nitroglycerine granular permitted explosives, when detonated in conventional headings, produce carbon monoxide and nitrous oxide fumes in excess of that allowed in the Mines and Works Regulations. The solution to this problem is to:

- ventilate the headings by means of force/exhaust fans or line brattice;

- use the newly developed emulsion explosives whose fume characteristics after detonation comply with legal requirements.

About ten years ago slurry explosives were submitted for test in the United Kingdom. Generally speaking the power of these explosives is the same as that of existing nitroglycerine types, but velocities of detonation are higher, being in the region of three to four thousand metres per second compared with the two to three thousand metres per second of nitroglycerine types. Not many compositions were examined or tested because it became clear during trials that these explosives were prone to desensitisation when fired in delay rounds. However, during testing it emerged that whereas slurries passed the 142 gram shot test easily, there was a tendency to fail the 800 gram stemmed shot test, which originally was designed to show that the explosive was relatively safe in methane gas when fired at charge weights similar to those likely to be used in practice. The problem posted by desensitisation has meant that slurry safety explosives of the group P1 type have never been used in UK coal mines, and so, few have been tested or examined in great detail. Slurry explosives of other safety groups have never been submitted for examination or test.

About two years ago an explosives manufacturer submitted samples of emulsion explosives for tests as group P1 explosives. The explosives contained no recognisable flame inhibitor, detonated at about 5000 metres per second, and produced some powerful gas ignitions that hastened the decommissioning of a test gallery for repairs to its foundations. Subsequently, some flame inhibitor was introduced into the explosive compositions, and this appeared to have the desired effect of reducing both the violence and the frequency of ignitions; compositions that passed the P1 tests were eventually developed. However, these compositions still had a tendency to pass the 142 gram test and fail the 800 gram stemmed shot test. Clear explanations of this difference between nitroglycerine and slurry and emulsion explosives are not currently available.

Possible gas ignition agents include: hot gaseous detonation products, hot particles and the shock waves produced by detonation. It is difficult to know or predict accurately the temperatures reached by the detonation products of explosives. However, the powers of group P1 emulsions and nitroglycerine explosives are similar so that it seems unlikely that there will be substantial differences in these temperatures.

The differences between nitroglycerine and emulsion explosives that have emerged during testing have caused the UK testing authority to impose temporary increases in the severity of the group P1 test requirements until more information and experience of them becomes available. In the case of the 142 gram test the requirement has been changed so that an explosive must cause less than 7 ignitions in 26 shots to pass. For nitroglycerine explosives up to 13 ignitions in 26 shots are allowed. The difference between these two requirements was chosen to be significant at the 95 percent confidence level, which is the level normally used by the UK testing authority. Normally manufacturers aim to produce group P1 explosives which cause considerably fewer than the thirteen ignitions allowed in this test. The 800 gram stemmed shot test, which can cause heavy

ignitions that are damaging to the test apparatus and surrounding buildings, was changed to require that no ignitions occur in ten shots.

Samples of emulsion explosives that have successfully passed the group P1 tests have been kept in the magazine and their detonator sensitivity examined at regular intervals using a standard range of detonators of steadily increasing size of base charge. It has been found that the detonator sensitivity of some emulsions has deteriorated within three months of manufacture to a point where they could not be detonated even with a detonator having a base charge of one gram of PETN. This seems to be a problem with emulsion explosives, and one which requires vigilance.

4.3.11 Stemming

The sole requirement for good stemming is that the stemming should not be ejected from the shot hole upon detonation of the explosive; rather it should contain the detonation and thus break the coal or stone layer. Clay was long used as a stemming material when the legal requirements were that the hole, unoccupied by explosive cartridges, had to be totally stemmed to the collar as shown in Figure 22.

During the past two decades many new materials have been tested such as gels and water encapsulated in plastic sheaths, stone dust, sand and special clays wrapped in waxed paper to prevent the clay from drying out. Clay tamping to be effective has to be of the correct moisture content - if too dry it cannot be forced into occupying the total shot hole circumference and if too moist it is easily ejected at the instant of detonation.

The results of incorrectly stemmed shot holes are exemplified in the case studies discussed in this chapter.

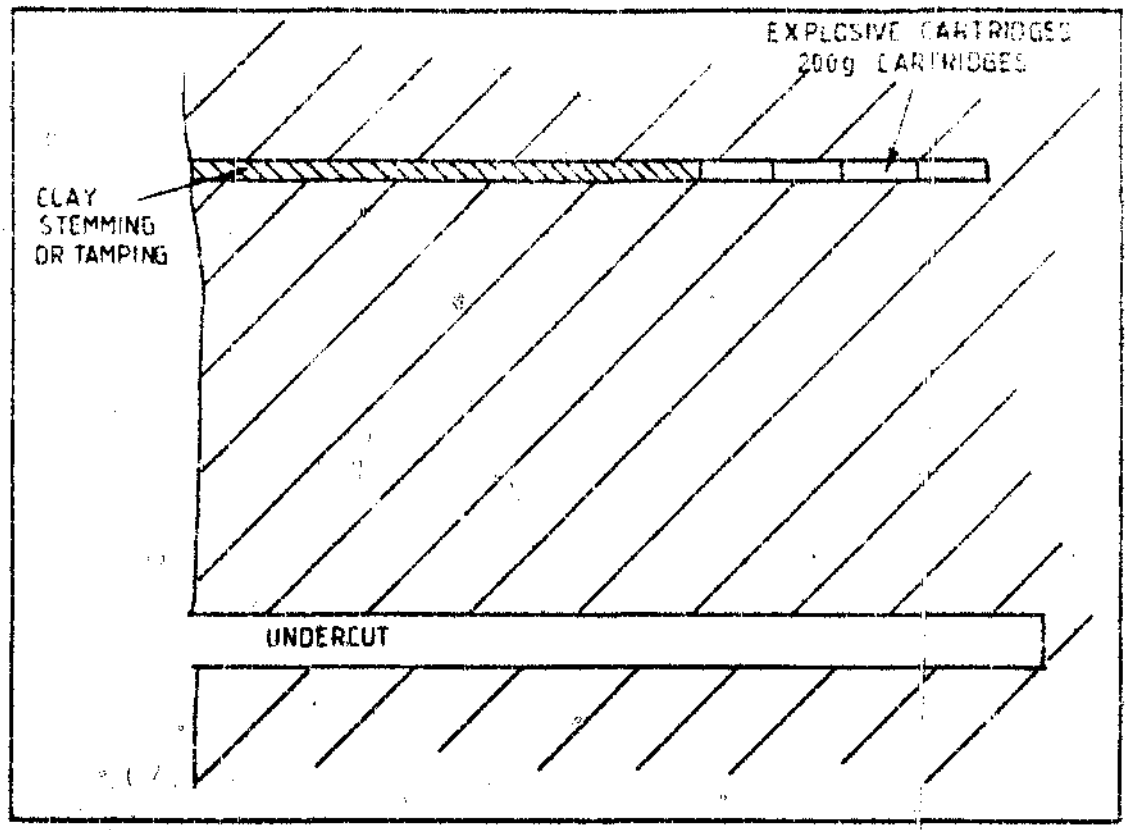


Figure 22

SKETCH OF A PROPERLY STEMMED HOLE

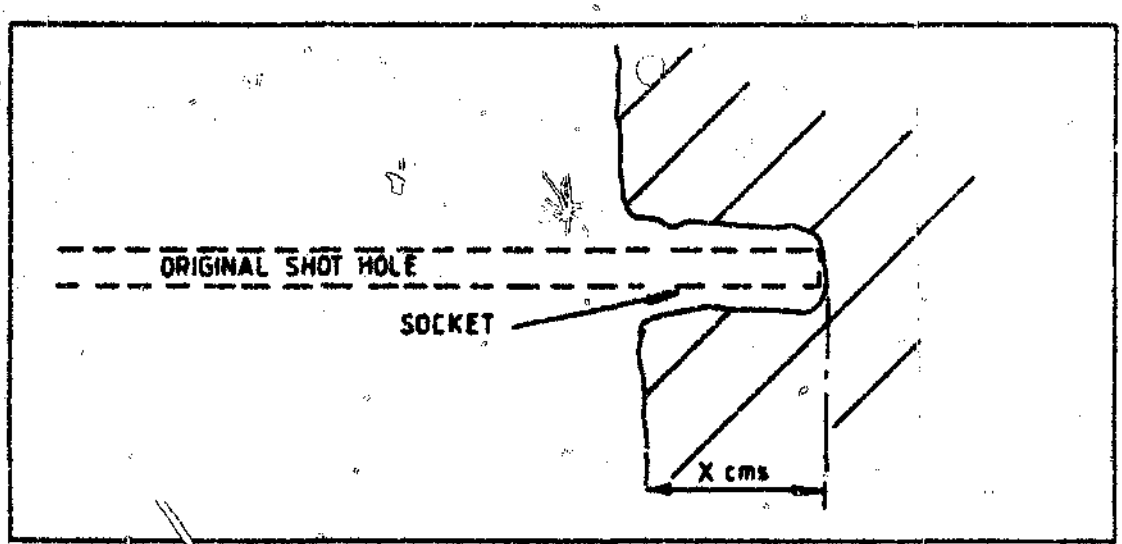


Figure 23

SKETCH SHOWING SOCKET

4.4 COAL DUST EXPLOSIONS

A high percentage of the recorded coal dust explosions in South Africa have had their origin in the ignition source of explosions. It is intended to deal with this aspect (coal dust explosions) under this chapter.

4.5 NUMBER AND DATES OF INCIDENTS

Table 4.3 Number and dates of incidents

Date	Colliery	Fatalities	Remarks
1899	South African	2	Blown out shot of "Thorite" - a locally made chlorate explosive.
1905	South African	1	Blown out shot.
1908 February	Glencoe	77	An electrically fired "carbonite" shot (permitted explosives) blew out and ignited firedamp which set brattice cloth alight; the fire spread to timber and coal; there were no immediate fire fighting facilities. Attempts were made to wall off the fire but the walls could not be built to confine the fire closely with the result that the volume of workings inbye of the walls was excessive. A series of methane explosions was caused by the fire and these culminated in a coal dust explosion which wrecked the mine and killed the persons including the Inspector who were engaged in sealing and rescue operations.

Date	Colliery	Fatalities	Remarks
1908 June	Cambrian	Nil	An electrically fired "carbonite" shot blew out and ignited firedamp and started a brattice fire. The circumstances were practically identical to the Glencoe Colliery situation. Persons, however, were withdrawn from the workings when it became apparent that the fire could not rapidly be extinguished or closely confined. A series of minor explosions again followed and finally led to a great coal dust explosion which swept through the mine.
1918	Hlobane	Nil	Blown out shot. The miner in all probability had not tested for gas before blasting.
1918	Northfield	Nil	Shaft bottom ignition in course of sinking.
1919	Northfield	Nil	Blown out shot. Miner in all probability had not tamped the hole properly or tested for gas before blasting.
1921	Bannockburn	Nil	Blown out shot when blasting in gas.
1921	Burnside	6	Blown out shot when blasting in gas in a dyke heading. The hardness of the dyke and insufficient tamping were the probable cause of the shots blowing out.

Date	Colliery	Fatalities	Remarks
1922	Natal Navigation	Nil	Blown out shot. The miner had probably not tested for firedamp before blasting.
1926	Durban Navigation No. 2 Pit	125	A sliping hole, that is, one without two free faces, was blasted in the presence of firedamp and blew out. A firedamp explosion resulted which generated a great coal dust explosion which travelled through most of the mine.
1930	Burnside	38	Blasting in the presence of inflammable gas. The explosion was reported as not having been augmented to any extent by coal dust.
1935	Consolidated Marsfield	±95	Blasting.
1943	Northfield	78	An uncertificated learner Miner blasted an insufficiently tamped shot hole in an unventilated stall filled with firedamp. The shot blew out and a violent explosion resulted. There was evidence that coal dust in the section may have augmented the violence locally but the main access roads to the section were either well stonedusted or wet and the explosion did not propagate.
1943	Consolidated Marsfield	Nil	Blasting.
1945	Landau/Navigation Colliery	10	Blasting.

Date	Colliery	Fatalities	Remarks
1951	Durban Navigation No. 2 Pit	9	Blown out shot. Miner failed to test for firedamp before blasting.
1951	Cornelia Colliery	9	Blasting in the presence of gas.
1952 March	Hlobane No. 2	Nil	Non-fiery mine. Blasting caused a coal dust explosion.
1952 April	Hlobane No. 2	1	Non-fiery mine. Blasting caused a coal dust explosion. Incident concealed from Mines Department. Firedamp was not suspected; it had never been detected in the mine.
1954	Newcastle-Platberg	1	Blasting in an inflammable atmosphere.
1961	Klandsberg Anthracite	Nil	Firedamp was ignited by a worker wafting out blasting smoke with a piece of hessian which happened to be smouldering possibly as the result of blasting operations.
1961	Natal Coal Exploration	Nil	Blasting in an inflammable atmosphere.
1962	Northfield	10	Blown out shot into gas filled goaf in a pillar extracting section.
1965	Hlobane No. 1	Nil	Ignition of coal dust during blasting operations.
1965	Indumeni	Nil	Blasting in inflammable gas. Miner failed to test between successive shots.
1965	Kilbarchan	Nil	Blow out of underburdened or overcharged shot hole when shotfiring in firedamp.

Date	Colliery	Fatalities	Remarks
1965	Newcastle-Platberg	Nil	Blasting in firedamp in a stone raise. Miner failed to test and to carry out correct procedure in dealing with a misfire.
1966	Kilbarchan	Nil	Blown out shot from overburdened shot hole. Miner failed to test for firedamp between shots and to tamp adequately.
1966	Natal Coal Exploration	Nil	Blown out shot. Underburdened shot hole or excessive charge.
1967	Kilbarchan	Nil	Blown out shot. Overburdened holes and failure to test between shots.
1968	Natal Coal Exploration	Nil	Ignition of firedamp when blasting. The miner failed to test between successive shots.
1968	Kilbarchan	Nil	Ignition of firedamp when blasting.
1968	Indumeni	Nil	Ignition of firedamp when blasting.
1968	Enyati	3	Blown out shot. Insufficient tamping. Poor ventilation.
1968	Utrecht	Nil	Blown out shot or unconfined explosives.
1969	Northfield	Nil	Underburdened shot hole in a lift in a stooping section blew into goaf. It is also suspected that the shot holes were not properly tamped.
1970	Durban Navigation	Nil	Miner failed to test for inflammable gas before firing charges.

Date	Colliery	Fatalities	Remarks
1972	Newcastle-Platberg	Nil	An underburdened shot hole and shot holes not properly tamped.
1976	Durban Navigation	2	Miner blasted in an unventilated working face.
1978	Newcastle-Platberg	Nil	Hole blew through to a split which contained gas.
1979	Balgray Colliery	1	Blasting out snook in stooping section.
1981	Balgray Colliery	Nil	Blasting ignited methane blower.
1981	Newcastle Platberg	10	Blasting in presence of gas.
1981	Newcastle Platberg	Nil	Old explosives used for blasting.
1982	Durban Navigation	1	Blown out shotholes.
1985	Middelbult Colliery	30	Blasting aircrossing.

With regard to the Consolidated Marsfield disaster in 1935, the author has not included the fatalities in the totals in Table 1. It is understood that 5 Whites and some 73 Blacks lost their lives in the explosion which destroyed 4 or 5 producing sections.

The cause was attributed to either blasting or an electric arc. The author has been unable to locate the official file on this incident.

4.6 DETAILS OF INCIDENTS/PHENOMENON

Arising from the 47 methane/coal dust explosions due to blasting and explosives on record, two incidents have been chosen for detailed examination. These are the explosions at the No. 2 Pit of the Durban Navigation Collieries on the 8th October 1926 and the Northfield Colliery explosion on the 26th May 1943. Both explosions resulted in considerable loss of life, and the inquiries were conducted in great detail. Although both explosions were as the result of blasting operations, several important extraneous factors which had a bearing on the explosions have to be considered.

4.6.1 Durban Navigation Colliery, No. 2 Pit

a. Introduction

On the 8th October 1926 at 18h15 a coal dust explosion traversed the whole mine and erupted out of the two vertical shafts destroying the upcast shaft fan installation and the decking arrangements at the downcast coal hauling shaft. 125 men (the total night shift labour force) lost their lives and the mine was rendered inoperative for 4 months while recovery operations were in progress. Tragically the Mine Overseer lost his life also in the subsequent rescue operations.

The mine was considered to be one of the best managed in Natal and is "well known to be a particularly well-ventilated shaft and thus there can hardly be a question of the explosion being due to gas". (Natal Mercury, 1926).

The Committee of Inquiry appointed to investigate the disaster was preceded by much ill-feeling resulting from the exclusion of representatives of Trade Unions at the investigation.

The details of the standards of mining and operational aspects at the colliery prior to the explosion are discussed. The evidence which totals 412 pages covers, inter alia, much detail of the rescue and recovery operations, the great courage displayed by the rescue teams, and the unflinching determination of the Mines Inspectorate to arrive at the cause of the explosion and prevention measures.

b. The Colliery

Photo No. 1 shows the surface installations at the colliery circa 1922. The coal seams were situated at a depth of 650 feet (198 metres) from surface and were overlain by a dolerite sill as shown in Figure 24 below. This dolerite sill was not present over the No. 1 Colliery workings and this geological aspect was put forward as a theory as to why the No. 2 Colliery experienced more gas problems than the No. 1 Colliery, where cracks through to surface allowed the methane to bleed off to surface; a position not possible at No. 2 Colliery where the gas could not escape to surface due to the dolerite sill.

c. Ventilation

The mine was ventilated by a 4,8 metre diameter Sirocco fan delivering approximately 190 m^3/second. The two shafts were each 5,4 metres in diameter. The fan pressure was 746 Pa. The ventilation distribution in m^3/second is shown on the underground plan Figure 25.

Stonedust was supplied underground in tubs at the rate of 20 tubs per week. The mine tubs were 0,84 m^3 capacity, thus the quantity



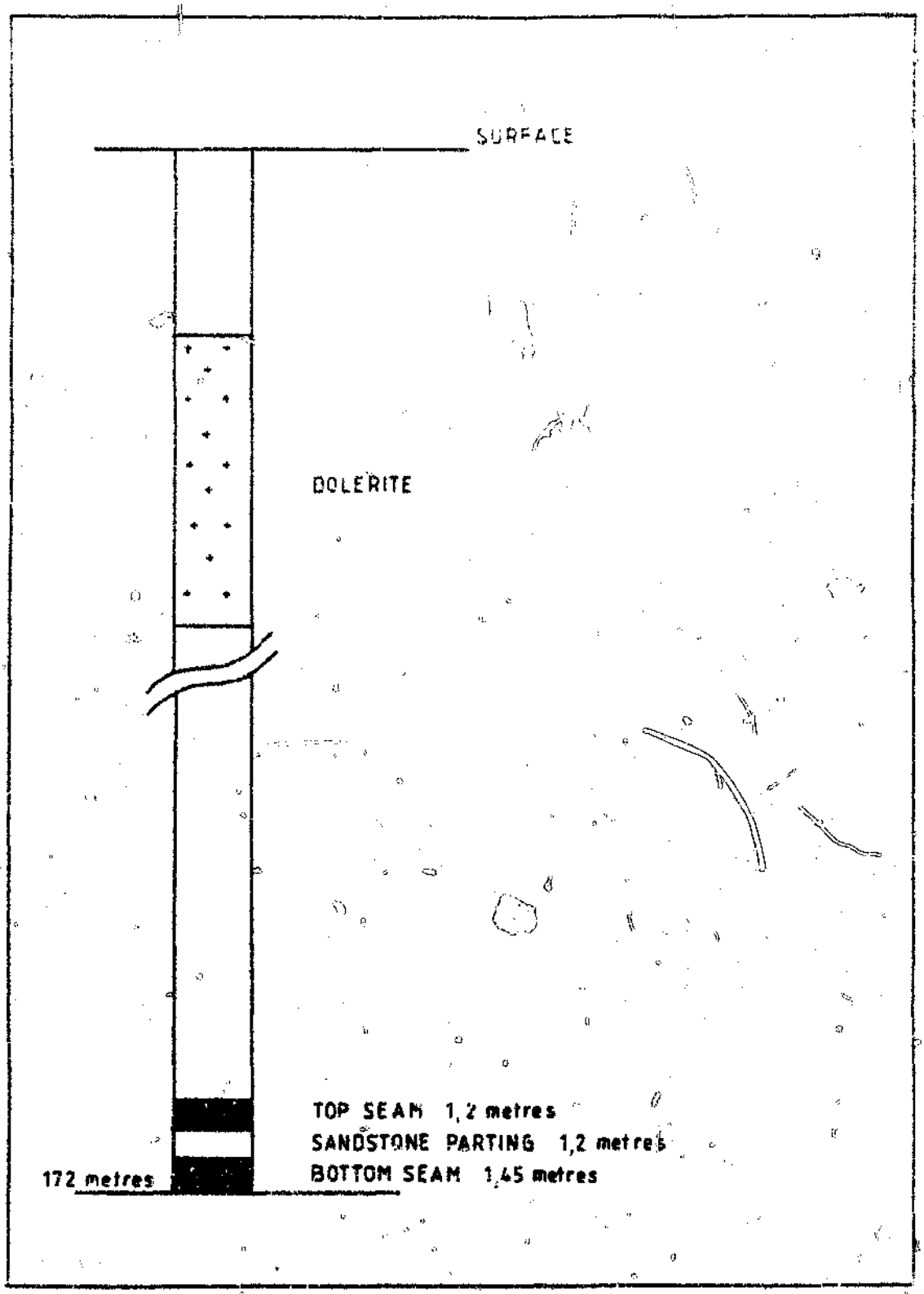


Figure 24

STRATIGRAPHY OF No2 COLLIERY
© DURBAN NAVIGATION COLLIERIES

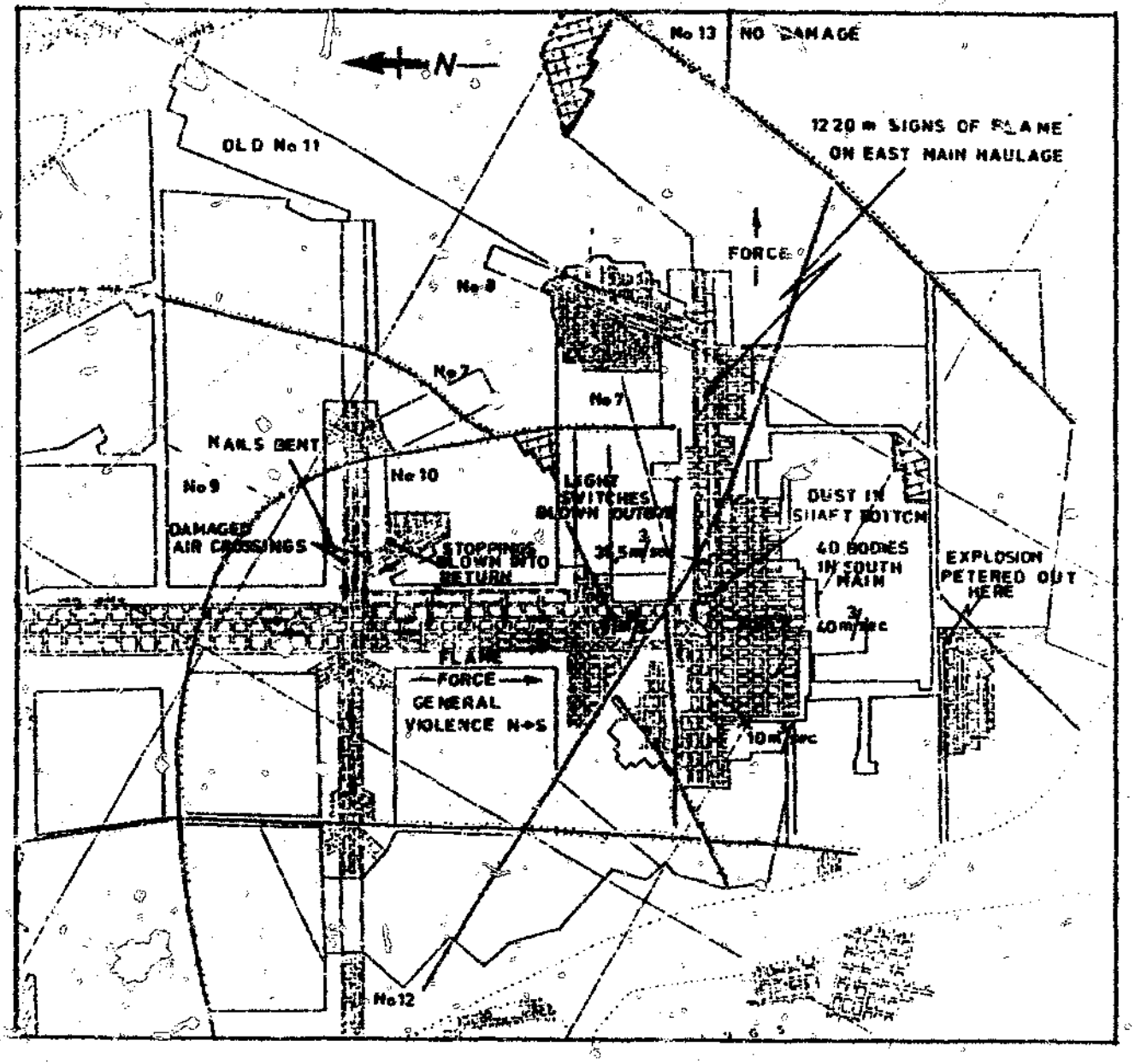


Figure 25
PLAN OF DNE No 2 WORKINGS
1926

The Electrician was working in the shaft bottom, 40 metres from the shaft in the East Main haulage. He was busy installing new cables using a draw vice and after the explosion, 5 of the seven wires showed signs of fusing. The power was on at the time of the explosion. The Electrician's body was discovered 8 metres from the position of the draw vice and 20 metres from the fused 250 volt wires.

The evidence of the dying Indian and the fused wires in the shaft bottom led Management to the conclusion that the Electrician had possibly fallen off a ladder while working on the cables, seized the wires to save himself from falling and had thus severed the live wires causing an arc which had ignited the coal dust and finally caused a coal dust explosion. Watson and Edgecombe (1987) argue that this was probably the verdict which the mine Management would have liked (an accidental electric arc) since it would indicate a pure accident and not affect insurance claims.

However, after several weeks of careful examination of the workings, the Mines Department offered a different explanation - a blown-out shot in section 10 in the North Main ignited firedamp which exploded and developed into a great coal dust explosion which traversed the whole mine.

Section 10 was developing in the North main (Figure 25) to the rise which would have resulted in methane layering against the roof.

By the 8th October 1926 the main drive had been developed off line to the left and the Shiftboss, on the 7th October 1926, had given the apprentice Miner on dayshift instructions to get the main back on line.

Figure 26 depicts a plan view of the face of the main in Section 10 as it was on the 8th October 1926. Shot holes had been blasted on the right hand side of the roadway where attempts to get the face back on line had been made by undercutting the

right hand ribside. This had resulted in two ribs of coal being left on the sidewall in which sockets were evident including 3 sockets in the face. The face was described by the Inspector of Mines after the explosion as "being in a horrible condition", meaning off line and not having sidewalls properly on line. In addition no brattice cloth had been erected into the face to remove methane which, by experience, would have been given off freely from the slip plane as shown on the plan (Figure 26).

In addition, a large pile of loose coal was found, after the explosion, lying on the floor on the right hand side of the heading. This coal had been blasted off the right hand side of the main drive presumably to get the main back on line.

The Miner in charge of the section on night shift (he was killed in the explosion) was, like his dayshift counterpart, an apprentice Miner who legally was not permitted to blast. No blasting battery or cables were found at the site of Section 10 main drive.

A detailed examination by the Mines Department traced the origin of the explosion to the main road in section 10. This was arrived at by the following deductions:

- Experience has shown that little damage is recorded at the 'eye' or 'origin point' of an explosion. The haulage return wheel in Section 10 was intact after the explosion.

- The recording of coking or burnt coal dust on one particular side of installations such as mine props, haulage frames, tubs, canches and aircrossings.

- The destruction of ventilation stoppings and other permanent installations and the position of debris from these devices after the explosion relative to their position prior to the explosion.

of stonedust delivered underground per month would have been 20 tubs x 4,33 weeks/month x 0,84 m^3 x 1,8 specific gravity of stonedust x 1000 = 130 939 kgs.

The colliery produced approximately 1000 R.O.M. tons of coal per day or 24,500 R.O.M. tons/month giving a ratio of 5,3 kgs of stonedust applied per ton produced, a figure far in excess of current standards of application.

The mine was worked on the panel system as shown in Figure 25. Line brattice was used to direct the ventilation into each working face.

A practice existed on the colliery to clear up and remove all dust which collected on intake roadways. This dust was loaded into bags and put into tubs and hauled to surface. Obviously any stonedust mixed with the coal dust would also be removed as evidenced in the Inquiry when the Manager gave evidence:

"In this place here, referring back to the bottom of the downcast shaft, where Maclachlan was working, did you consider that that was adequately stonedusted? - that night, previously to the accident, any time - that night it was not.
You had removed it all? - Yes.
With the stonedust you would have removed the coal dust? - Yes.
Most of it? - I would not say most of it.
Do you think that you removed all the stonedust? - Yes, I think so.
And do you think it was quite safe as it was? - It was going to be stonedusted immediately.
If you cleared it away and put down no more stonedust, would it have been safe? - I do not think so.

Surely, if you take away something that is dangerous, then it must be safe? - We were taking away stonedust which would also render it unsafe. The more stonedust we take away, the more safe it would be.
You were trying to take away as much stonedust as coal dust? - You cannot do that.
You took it away, both mixed? - Yes.
If you took anything away, you took both? - Yes.
And the remaining which you did not take away might have been only coal or only stonedust? - Yes.
Therefore you would say that it might be dangerous? - Yes.
Surely if you clear away this stuff which is dangerous, it must be made safe? - It would not be made safe.

The only evidence of stonedust sampling and analysis is given by the Inspector of Mines who stated in evidence that; "In the ordinary course of my duties I took them in December of last year (1925). There were three samples taken along the east road. The one at the pit bottom, very near to where the Electrician was found, contains 90 percent of incombustible matter, which is very good.

Other points? - At other points they were 200 metres from the face of No. 6 section - 84 percent incombustible. 600 metres from the face and 600 metres from the pit bottom - that indicates the same spot - 90 percent of incombustible matter.

From the general appearance of the dust in future examinations, what was your opinion of it? - It was always kept very well stonedusted."

From this evidence, it would appear that no records of stonedust sampling and analysis were done on a regular basis and it is therefore difficult to gauge the state of affairs regarding stonedusting on the 8th October 1926. Since the mine was ravaged by a coal dust explosion, it is concluded that the stonedusting was not adequate.

A point of great importance is the fact that no gas was apparently reported in the three years prior to the explosion. The Inspector of Mines stated however that during a visit to the mine on the 18th February 1926 in Section 9 he found 5%-6% methane in a face. The face was fenced off and the necessary precautions were taken. The mine was generally dry.

d. Mining Methods

The mine was worked on the bord and pillar method. All workings were developed in the bottom seam which was 1,4 metres high. Only the main haulages were brushed to the top seam floor by blasting down the midstone parting which was 1,2 metres thick. Consequently, the return airways, being low in height, were firstly infrequently examined and secondly would be seldom stonedusted. Since the mine was dry, fine coal dust collected in all the return airways and would build up in thick (10 mm) layers on timber supports and ridges left on the roof and sidewalls. In 1950 the author carried out surveys of the ventilation at the colliery in the northern areas and found this situation to be the case 24 years after this disaster. This was to have disastrous consequences on the night of the 8th October 1926, by which date the mine had already been in operation for seven years. Figure 25 reveals that the mine was blocked out in panels which could be isolated or sealed off at short notice. Double seam pillar extraction was practiced in the panels.

In the sections compressed air, supplied from surface, was used to drive the Jeffrey coalcutters, drilling machines and pumps.

Power was fed underground at 500 volts to the Main East haulage engine and a pump in the North Main. The haulages were lit by 250 volt electric lights. The Main North haulage engine was situated on surface and was steam driven. (This engine was still in use when the mine ceased production in 1958).

The only other electric apparatus used underground were electric cap lamps. Miners used the Marsant Davy type safety lamps with double gauzes for gas testing purposes.

e. The Incident

On 8th October (a Friday) the day shift, consisting of some 1100 men, had not produced any coal due to a lack of railway trucks.

At 17h00 the night shift, consisting of 4 Whites, 111 Blacks and 10 Indians, descended the mine to carry out the general work of brushing, cleaning up and undercutting coal.

At 18h14 a dull bump was recorded followed by a flame which erupted from the fan drift of the upcast shaft which was then followed by a cloud of dust and smoke from both shafts. It was 23h00 before the first rescue teams reached the shaft bottom where 3 injured Indians were discovered still alive. These men were brought to surface by 03h00 on the Saturday (9th October 1926).

Before dying on the Monday, the one Indian was able to give evidence of his experience in the shaft bottom.

By the Court:
"Will you tell us, please? - The translator replied: He said that the Electrician - must I put it into broken English, in the way he put it to me? Well, he said the Electrician had been into a section and he told him (the Indian) to wait near the Mine Captain's office until he returned. He waited. He said the Electrician with the Helpers, his usual Helpers, returned and was fixing a cable. He said a flash of light, a ball started from the Electrician and ran along the ground towards the Mine Captain's office. He ran then towards the engine; the flash did not come from the section but was started by the Electrician. He and Kullen went towards the office to shut themselves in and then there was the explosion.

Did he say anything about there being an explosion? - Yes, he said that there was an explosion.

Did he say anything about a flame and the course of the flame? - He said that it ran along the rail like a paper football.

Mr Milne: Was the Indian examined at all on his statement; were any questions asked of him? - No.

Did he say that "the Electrician first went to the section and told me to stay at the office; he and two Helpers worked with him close to the switch. I stayed close there. I was snuffing. Just after the Electrician got there, there was a big flash of light, like a ball; then an explosion"? - Yes, he said that.

The flash came from the Electrician? - And not from any section.

You say that he said that? - Yes.

"I don't know how that occurred, but there was just a ball of light and then an explosion" - Yes.

A flash of light ran towards the Mine Captain's office; and then there was an explosion? - Yes.

"I ran towards the engine"? - yes.

The light seemed to run along the ground towards the Mine Captain's office? - Yes.

You agree that this is a fair representation of what he said? - Yes, I do."

The evidence of the Police Sergeant who took the statement from the dying Indian is that "We have had medical evidence as to his physical condition. Mentally you think he was in a condition to make a clear and sane statement? - Certainly.

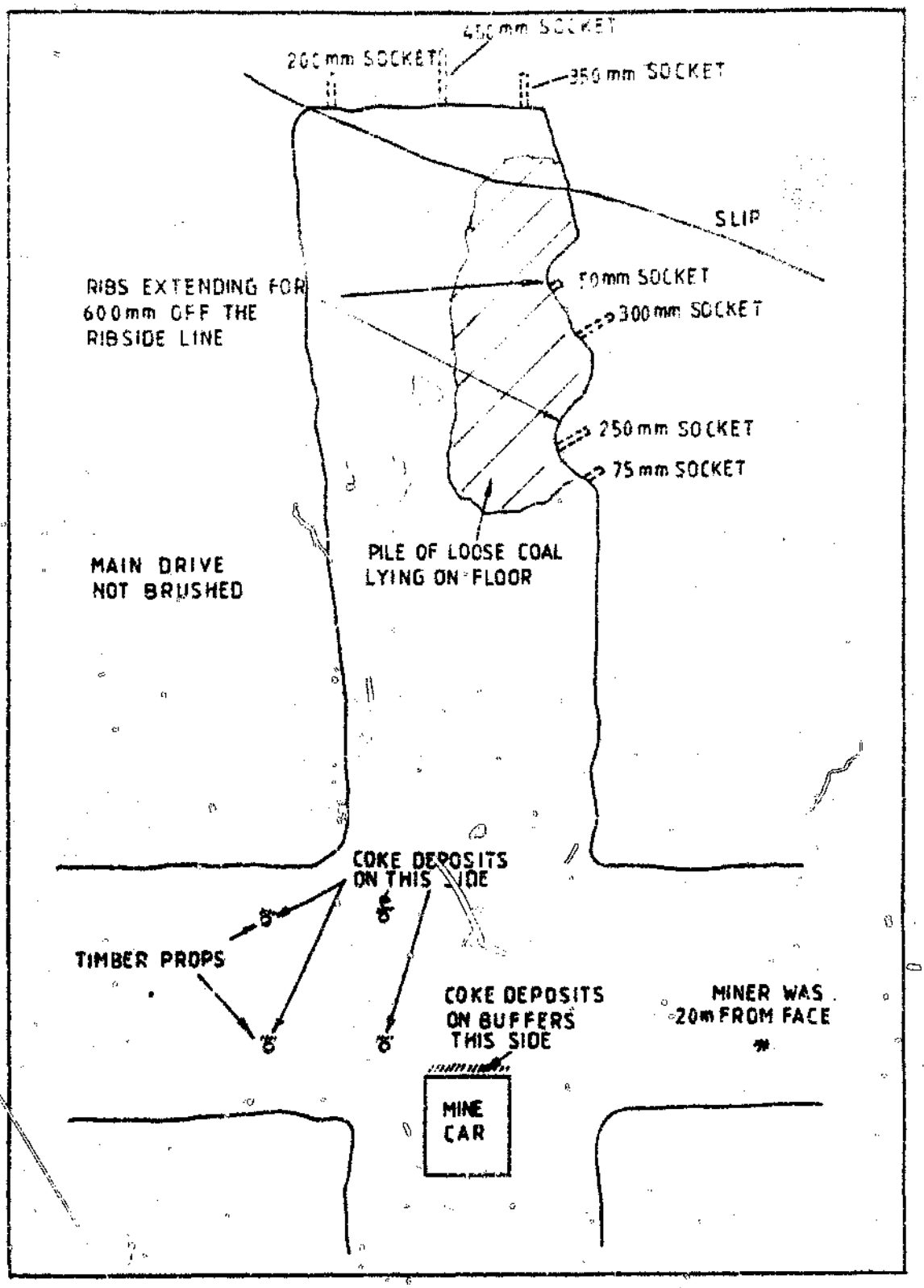


Figure 26

PLAN OF FACE OF MAIN IN No 10 SECTION

- Nails and cable hooks, for instance, which had been bent by the force of the three shock waves in a coal dust explosion would provide evidence of the direction of the force of the explosion.

The evidence of the Inspector of Mines is quoted and the direction of the force of the explosion is shown by arrows on the plan Figure 25.

"Now from what you have seen, could you give us an idea of the course which the explosion traversed? - Yes, I think it came out at No. 10 Section, somewhere here (witness indicates on plan), I think it went down the intake and probably down the west side return. When it came out of No. 10 intake, it split. The force went along the intake into old No. 11. It also went in along new No. 11; it also went along on the road towards the main north, along both the intake and the south return. I think it went into the int[illegible] of No. 7 and it also continued towards the main north intake. Part went north along the main north this way (witness indicates on plan). Part went along the west, along the main north inbye, and it also travelled inbye No. 8, and travelled inbye No. 12. It also went up the main north intake towards the downcast shaft and went up the secondary intake, the downcast shaft, and I think it went up all these returns on both sides the main north intake. I would like to add that some of the nails sticking out of plugs in the roof on the main north intake show violence from north to south and also two 150 mm girders, fixed in an upright position 30 metres from the downcast shaft show signs of violence; they were bent 15 degrees from their vertical position. They were bent out by force. By force going inbye; they were bent towards the direction of the shaft. Then I think part of the force went up the downcast shaft; part went through the three separation doors between the downcast and the upcast and then part went up the upcast shaft: the explosion continued past the downcast. It split at the "Y"; the force went south along the intake and then gradually petered out, died out.

Mr Milne: There is an expression you used once or twice; you used "outbye force" when you meant the other way? - I did not notice it. It was coming from inbye and going outbye; that was what I meant to convey. I might unconsciously have said it once or twice.

The Chairman of the Inquiry: Have you formed any theory as to what caused the explosion, assuming that it did start from No. 10? - Yes, well seeing those three sockets in the right hand side of the main heading, they struck me as being very suspicious. The undercutting was not properly done, and some of these holes had been drilled in the solid. I came to the conclusion that it might have been due to the blasting of these holes there. They were partially blow outs, and I came to the conclusion that the blasting might have been done in the presence of gas, which might have come out of the slip in the face of the main heading."

Watson and Edgecombe (1987) state that it would appear, from oral evidence gathered some years later, that blasting had taken place in No. 10 Section, and that the battery used for blasting had been removed and thrown into a boiler before the Mines Department had got into the section to investigate.

The flame which the dying Indian saw 'emanating from the Electrician in the shaft bottom' was in fact the flame of the explosion passing through the shaft bottom.

In order to have a coal dust explosion develop from an electric arc in the shaft bottom, a large coal dust cloud would have had to be raised prior to the flash occurring; the mine was idle at 18h15 on the 8th October 1926 and all the haulages were on stop.

A damaged safety lamp (the glass was missing and the washers were out) was found some distance from the face of the main in Section 10 with two other undamaged lamps. Evidence suggests that the lamp was damaged by the explosion.

A disturbing aspect was the fact that the Shiftbosses were blasting in sections where apprentice Miners were in charge and so were apprentice Miners blasting; Miners also were not employed in sections for long periods. The Miner in Section 10 was actually working in the Main East haulage on the morning of the 8th October 1926.

Evidence suggests also that the Miner in Section 10 was in a hurry. This is supported by the evidence at the Inquiry:

"Of course, you looked at every possible theory but it is no good unless you can substantiate it in every possible degree? - No, I quite see that (reply by the Inspector of Mines).

And you have not got anything to support your theory that the Miner was in a hurry to get to the working face? - Well, I have the theory which I have already put forward that he had to get back to his workers, the main crowd of his workers.

To start them at their work? - Yes."

Just outbys of the main drive haulage wheels in Section 10, a water barrel on wheels had been tipped up on its end between the roof and floor (1.4 metres high) and beneath it was a damaged wheelbarrow; the force to cause this must have come from inbye Section 10 travelling in an outbye direction.

The author intends dealing with coal dust explosions in Chapter 7 and reference to this and the Glencoe and Cambrian Collieries disasters will be referred to in that chapter.

f. Conclusion

The face of the main haulage in Section 10 had advanced off line to the left of centre; in order to get the face on line the right hand ribside had been undercut with a Jeffrey coalcutter in

a jagged fashion and shot holes had been drilled above the undercut to blast down the coal. A slip was present in the face. The condition of the face was described as a "horrible mess". The face was rising to the south which would have encouraged methane to collect against the roof. There was no line brattice into the face to remove methane. On the 7th October on dayshift, the apprentice Miner in the section had loaded out all loose coal in the main and no further coal was blasted for the next two shifts (night shift of 7th and dayshift of the 8th October).

On the night shift of the 8th October, the apprentice Miner with 4 Helpers entered the section in a hurry and blasted the 3 sliping holes on the right hand side of the main drive. The face was dry. It was his intention, after the blast, to rejoin the rest of his crew in another section. The drill holes were drilled into the solid, resulting in a blown out shot which ignited methane and which subsequently developed into a methane and then a coal dust explosion which traversed the whole mine killing the total workforce on night shift. When the rescue parties reached the section several days after the explosion, loose coal was found lying in a heap on the floor of the main drive on the right hand side (Figure 26).

4.6.2 Northfield Colliery

a. Introduction

On the 26th May 1943 at 07h30, a methane explosion occurred in No. 0 section as a result of blasting operations. 78 persons lost their lives and it was only due to the quick-thinking and experience of the Miner in the neighbouring section, who managed to lead his crew through the return airway and to safety, that another 40 men did not lose their lives.

The Northfield Colliery, situated 5 kms north of Glencoe, formed part of the Natal Navigation Collieries and Estate Company Limited. The shafts were sunk in 1918 and production commenced in 1919.

The Inquiry and Joint Inquest into this accident was carried out with great diligence and attention to detail; it reveals how a succession of events led up to the disaster which was finally initiated by a non-qualified Miner blasting a shothole in the presence of an explosive mixture of methane and air. The damage resulting from the explosion was extensive.

b. Stratigraphy

The two coal seams occurred at a depth of approximately 200 metres from surface. Figure 27 shows a section through the two coal seams which were separated by 1,5 metres of micaceous sandstones.

Generally the lower seam was worked in the development phase, the midstone parting being brushed down on the haulage roads to provide a height of 1,8 metres as required by law. Brushing and cleaning out the stone on the main haulage always required a great deal of supervision and follow-up. If brushing, as was the case in this incident, was allowed to fall far behind the face of the main, the haulage could not be extended, tramming became excessive and permanent ventilation stoppings could not be installed.

The top seam in the Klip River Coalfield was generally worked by the double seam stooping method. Both seams were exceptionally gassy and often the gas from the top seam would bleed off under pressure into the bottom seam workings.

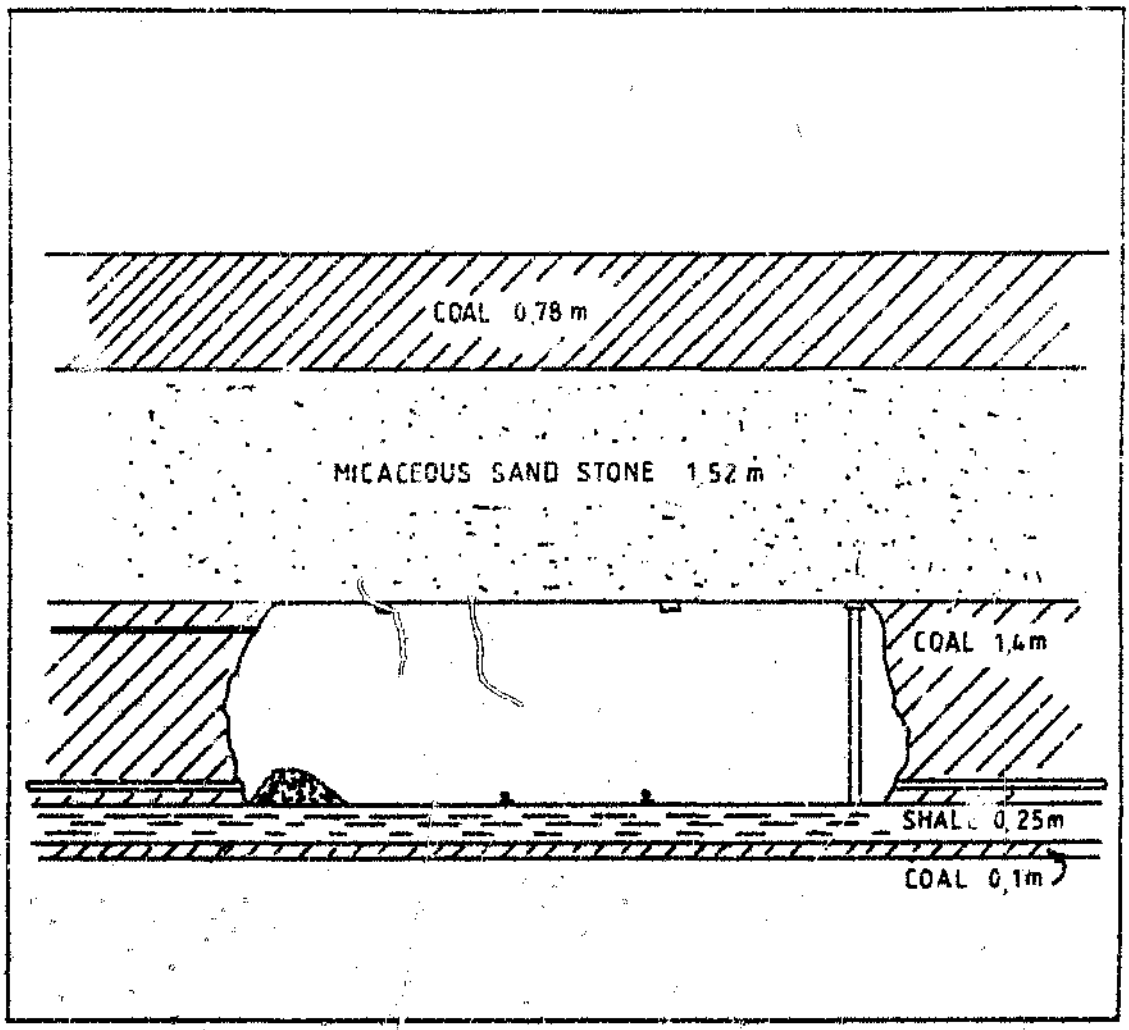


Figure 27

NORTHFIELD COLLIERY

SECTION THROUGH THE TWO COAL SEAMS

c. Ventilation and Stonedusting

The main surface fan was a double inlet Sirocco fan with a capacity of 150 m^3/second and a pressure of 1245 Pa.

Table 4.4 shows the ventilation quantities delivered to the various sections before the explosion and Table 4.5 the situation after the explosion. The explosion had its origin in Section 0.

Table 4.4. May 1943. Ventilation quantities to the sections before the explosion

	m^3/second
No. 2 Main Haulage Intake at aircrossing	24,7
No. 2 Main Haulage Intake Outbye No. 0 station	17,7
No. 2 Section Return at last working face	4,9
No. 6 Section Return at last working face	7,6
No. 2 Main Haulage Intake inbye No. 0 Section	16,5
No. 0 Main haulage Intake at aircrossing	12,8
No. 7 Split approaching No. 2 Main Intake	13,1
Quantity of ventilation available for Sections 2,6 and 0	30,8
No. 0 Section last through road at face to right	5,8
No. 0 Section last through road at face to left	3,8
No. 4 Drift Return	34,6
Main Return Upcast Shaft	95
Pressure in Fan Drift - 1145 Pa	

Table 4.5. June 1943. Ventilation quantities to the sections after the explosion

	m^3/second
No. 2 Main Haulage Intake at aircrossing	23,4
No. 2 Main Haulage Outbye No. 7 Split Intake	16,2
No. 7 Split Intake to No. 2 Main Road	11,8
Quantity of air available for Nos. 0, 2 and 6 Sections	28,3
No. 2 Main inbye No. 0 Station	11,0

No. 0 Section Intake at aircrossing	11,5
No. 0 Section Last Through road at face to right	6,3
No. 0 Section Last Through road at face to left	3,8
No. 2 Section Return at Dyke 61 metre from last face	2,3
No. 6 Section 30 metre from last face	5,4
No. 5 Section (Main West) inbye of aircrossing	1,2
No. 1 Section Intake outbye No. 7 Main Road	6,4
Main West Top 2nd Incline	23,4
No. 7 Main Haulage Inbye at aircrossing	26,5
Main return at Upcast Shaft	103,4
Pressure in Fan Drift 1145 Pa	

The main point of issue is the quantity of air entering Section 0 which was almost unchanged at 12 m^3/second. This is generally considered to be adequate for a Natal Colliery. The quantity passing through the last through roads was approximately 4,5 m^3/second in the last right hand through road and 3,8 m^3/second through the last left hand through road. The area of the last through road was 8,0 m^2, hence the air velocity in the road would be of the order of $\frac{4,5}{8,0}$

= 0,56 metres per second.

Under normal circumstances (line brattice installed in good condition in the faces and terminating some 3 metres from the face itself) such a velocity would have been adequate to remove any methane from the faces.

What was disturbing was the loss of air in the section itself. Out of a quantity of ± 13 m^3/second entering the section only 8 m^3/second was passing through the last through roadways. The reason for this discrepancy is explained later. Bratticing standards according to the Shiftboss were that face brattice should extend to within 1 metre of the faces. 4 days before the explosion occurred, evidence was given that the line brattice in each roadway was ± 10 metres back from the face.

Ventilation to this section (described as one of the best) was taken up the main intakes and split left and right into the section (see Figure 28).

Approximately 10-15 tubs of stonedust were delivered to the workings each day. This quantity proved adequate to prevent the propagation of a coal dust explosion.

Monthly coal production was 35 000 R.O.M. tons hence the application of stonedust would be as follows:

$$12{,}5 \text{ tubs} \times \frac{25 \times 0{,}84}{35\,000} \times 1{,}85 \times 1\,000$$

= 1,4 kg per ton mined.

This quantity is to be compared with the apparent 5,3 kgs per ton distributed at the No. 2 pit of the Durban Navigation Collieries in 1926 prior to the coal dust explosion.

d. The Incident

Figure 28 shows the area affected by the explosion. Figure 29 is another plan to a larger scale of the section where the explosion originated and Figure 30 is a plan showing the heading where the blown-out shots occurred.

The Inspector of Mines reported that at 08h30 on 26th May 1943 a telephone message was received from Natal Navigation colliery that an explosion had occurred about an hour earlier that morning in the area of Nos. 0, 2 and 6 Sections. This message was telephoned to the Inspector of Mines, Natal. He stated that:

"At the mine I was informed that the Manager was underground. The Colliery Engineer was requested to speed up the mine fan if this was possible without risk of damage and to make arrangements to send down brattice cloth, bricks, sand and cement in quantity.

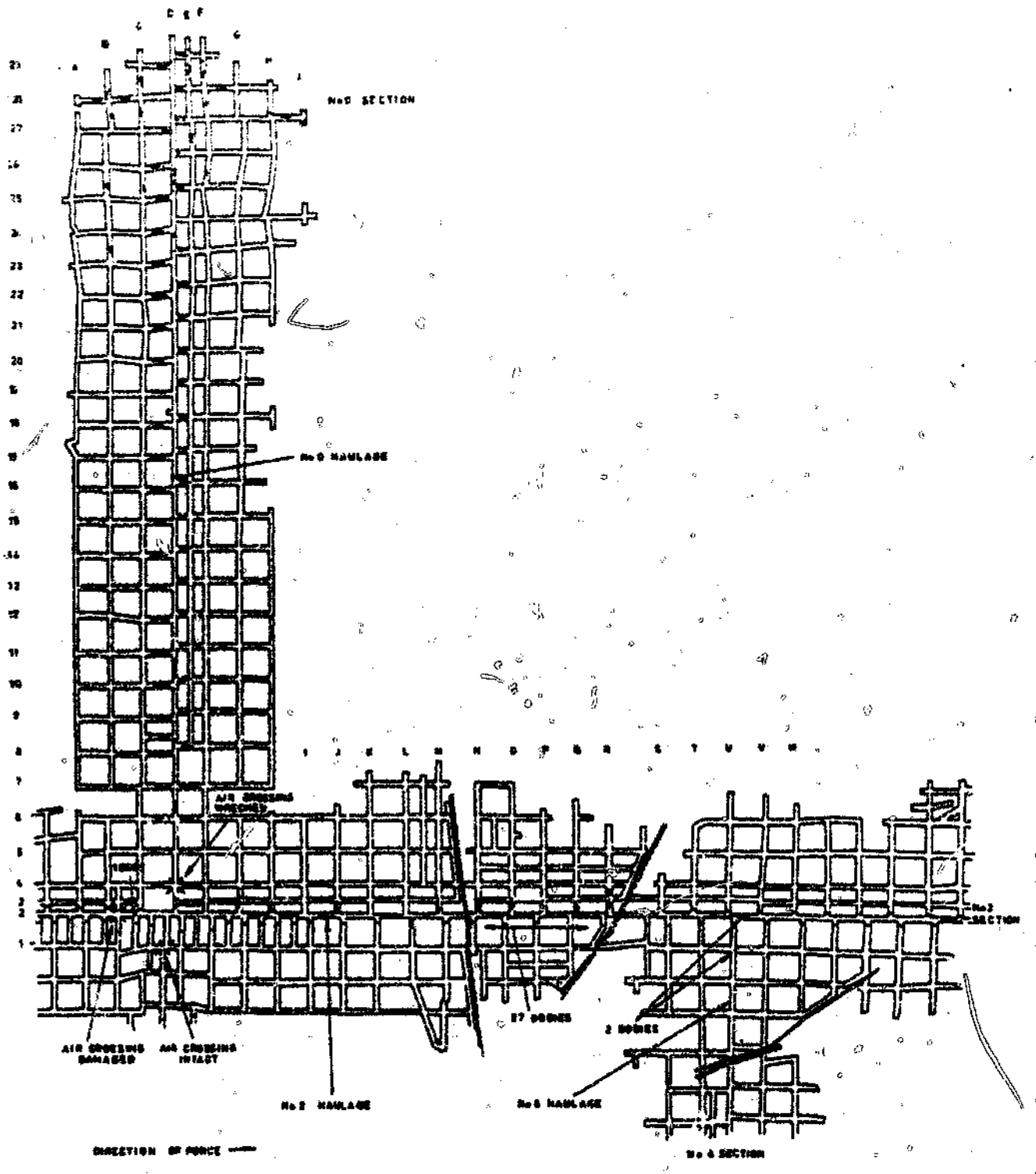


Figure 28

NORTHFIELD – AREA PLAN OF EXPLOSION

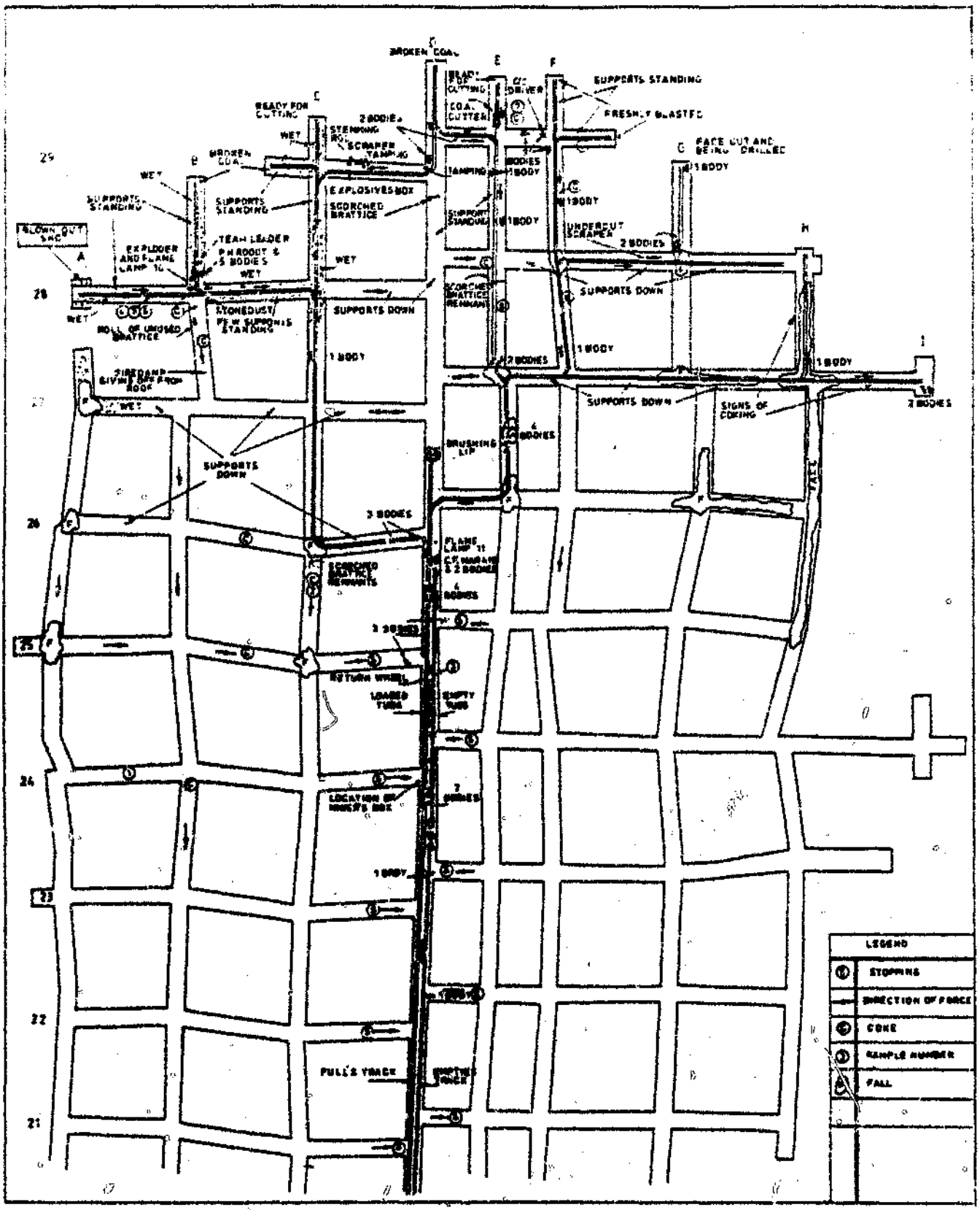


Figure 29

LARGE SCALE PLAN OF SECTION 0 NORTHFIELD

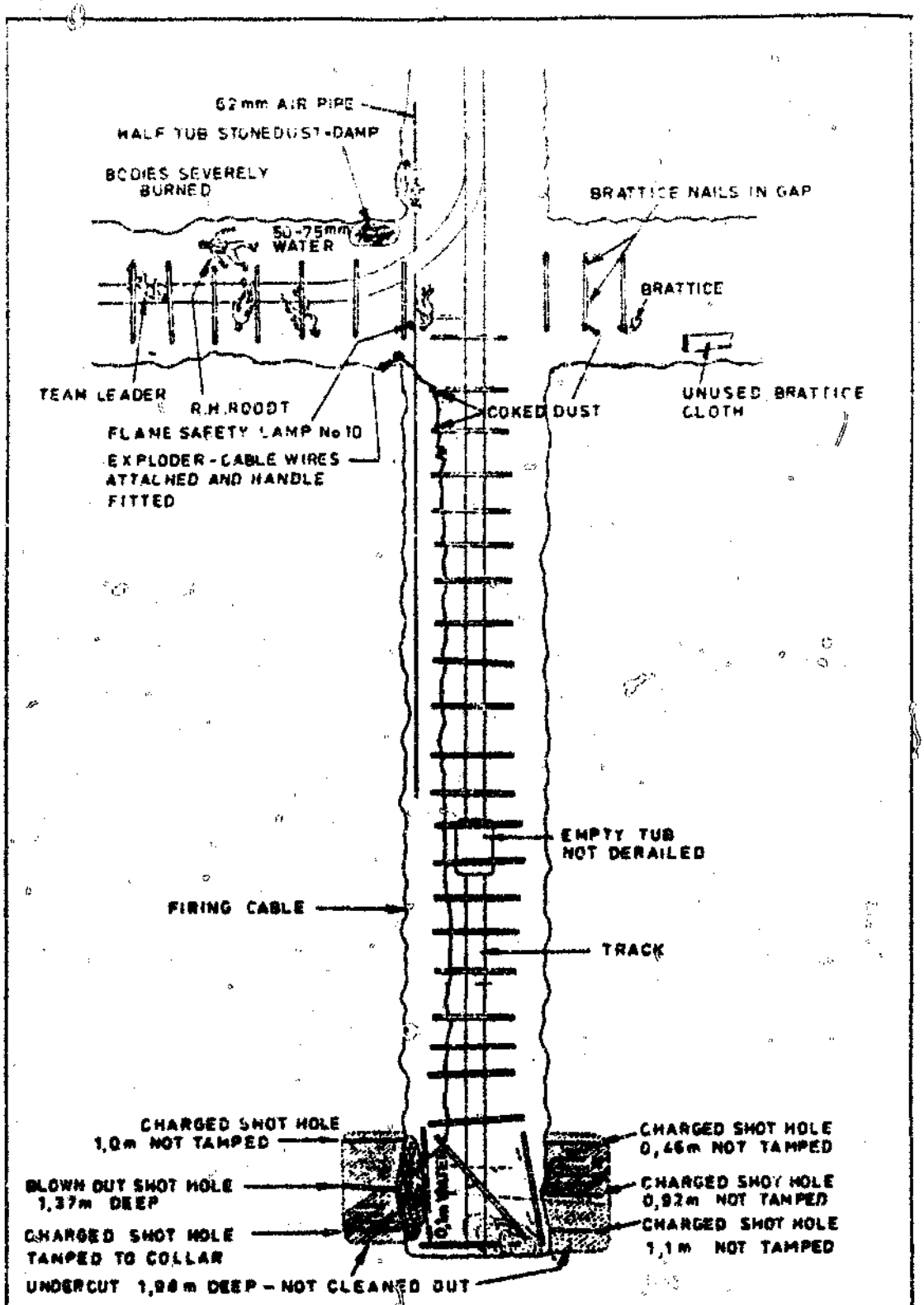


Figure 30
NORTHFIELD COLLIERY
DETAILED PLAN OF FACE WHERE EXPLOSION OCCURRED

At the shaft bottom I met a Shiftboss who gave me what information he could and I then went into the main return from the western area of the mine, through the doors off the main west haulage. Here was found a strong after-explosion smell - testing with a carbon monoxide detector revealed only a trace of carbon monoxide - less than .01 percent. Firedamp could not be detected on the flame of a safety lamp.

An attendant was stationed at the door with instructions to prohibit persons from using the return as a travelling way.

At 09h40 I reached the No. 0 Haulage Station in No. 2 Haulage. The air at this station was clear (of methane). Shortly before reaching the station, stretcher bearers carrying out those men who were alive, were passed on their way outbye. At the station itself tubs had been derailed and overturned, and the body of a man there appeared to have suffered considerable violence. A fairly considerable fall had take place at the station and the beams carrying the angle wheels for the haulage ropes along No. 0 Haulage had been dislodged from their position.

There was no sign at the station of the Manager or the Mine Overseer and I continued inbye along No. 2 Haulage for a distance of 120 metres where the canaries showed signs of distress. It was seen that stoppings on the left hand side beyond the station had been blown outwards from the direction of No. 0 section, and that foul air was emerging. I returned to the No. 0 section station and went in a distance of 91 metres along No. 0 Haulage where foul air was again encountered. Here too were signs that the force of the explosion had come from No. 0 section. It was seen that the air crossing carrying the return air from No. 2 section and from the right hand side of No. 0 section had been wrecked.

The Shiftboss arrived at No. 0 haulage station and I instructed him to withdraw workmen from the other sections of the mine and then to divert more air in along No. 2 Haulage.

Shortly afterwards the Manager and Mine Overseer appeared at the wrecked air crossing, and the Manager explained that he had effected a temporary repair of the air crossing over No. 2 Haulage which had been damaged, and had erected a brattice to prevent the intake air which was reaching No. 0 haulage station from short circuiting via the wrecked air crossing direct to the return serviced by the damaged air crossing.

It was quite clear that the force of the explosion had come from the direction of No. 0 Section and that there was more likelihood of persons still being alive in Nos. 2 and 6 sections.

Between N and R (see Figure 28) were found the bodies of 27 men and it was apparent that these men had fled out along the haulage and on meeting a deadly concentration of poisonous fumes had fallen one over the other in their vain attempt to escape.

At Q, 360 metres from No. 0 Haulage station, a prop was found across the road with "Come in this way, Ted" chalked on it. This direction was followed and in the workings to the right and a little further inbye were found two Miners with forty men. These men were led out and it was learned that they had with them all the workers in Nos. 2 and 6 sections who were still alive, except for one other who was badly gassed. He was found and was carried out to receive oxygen treatment.

These men owed their escape to the fact that the stoppings on the right hand side of No. 2 Haulage at L and Q collapsed allowing the bulk of the noxious fumes, which had reached this haulage, to escape to the return airway on the right hand side instead of being drawn in quantity to the working places in Nos. 2 and 6 sections.

There is no doubt that if these stoppings had not collapsed the death toll would have been considerably higher, and it is indeed unfortunate that some of the men from these sections had made a bid to escape when the fumes in No. 2 Haulage must have been at their worst.

It was apparent later that safe progress into No. 0 section could not be made until the wrecked air crossing had been rebuilt to carry, to the return, the foul air from the workings on the right hand side. This foul air was leaking through the brattices at the rear of the workmen.

By Monday 31st May it was possible to reach the brushing lip 90 metres from the face of the main road, and on Tuesday 1st June the section itself was explored. Details of what was found are shown on the plans.

No. 0 Section is in a panel of its own. Off No. 2 haulage three 4,57 metre wide roads lead through a barrier pillar into this panel, the centre road being the haulage and intake road which is from 3 to 3,2 metres high for most of its length. The other two roads form the return airways through the barrier pillar.

It would appear that the force of the explosion travelling outwards from the working faces of No. 0 section was checked by the restriction afforded at the barrier by the two return roads and sought release by blowing the brick stoppings on either side of the haulage road inwards.

The body of the Miner of No. 0 section, was found inbye of the haulage return wheel at the end of the double track. It appeared that he had been hurled head first under a tub loaded with broken rock from the brushing. A partly consumed packet of "aspro" was found in one of the pockets. It later transpired that he was suffering from toothache.

The Miner's initials for the date 26th May were not found chalked up in any of the working places in No. 0 section. His box, which had stood at D24 in the haulage, was shattered and only a few fragments of it could be found. The report book was not to be found and it is not known if a report on the condition of the section that morning had been made or not.

Nothing found in the right hand side was suspected as the possible cause of the explosion although in that area there were signs of considerable violence as large numbers of props had been knocked out and falls of roof had resulted. These falls were more numerous and of greater size on the extreme right hand side and outbye from the faces, suggesting that the explosion had gathered force here. Similarly nothing was found to suggest the cause of the explosion on the left hand side until the last working place was explored.

Figure 30 shows details of what was found there. This working place had been driven for a distance of 30 metres and two new faces, one to the right and one to the left, had just been started. These faces had each been undercut to a depth of 1,98 metres but the cuttings had not been cleaned out from the undercut. Both faces had been drilled, three shot holes in each. The centre shot hole in the right hand face had been fired and had blown out leaving a socket 1,4 metres deep, bringing down only about a ton of coal from the face. The other two shot holes in this face were standing charged with explosives; the one on the left appeared to be tamped to the collar, the one on the right had a length of 1 metre not tamped. Later the explosives contained in the holes were recovered and Figure 31 shows the details of these shot holes.

In the face to the left the three shot holes were charged with explosives but not blasted; from left to right the length of these shot holes measured 0,3 metres, 1,01 metres and 1,01 metres were all untamped.

Both faces were traversed to some extent by slip planes. The last 6 metres of the heading was wet and water had accumulated at the faces to a depth of 100 mm. The faces were wet and so also was portion of the inside of the blown-out shot hole.

Firedamp was being emitted from the floor, bubbling gently through the water from crevices. There was no brattice cloth

leading in to the face. Testing for firedamp showed the presence of 1.1/2 to 2% methane. The end of the firing cable was dangling from the strap over which it had been threaded at the face. The leads had obviously been connected to the shot which had blown out.

From the face the cable led up the road to the exploder which was round the corner. The leads were connected to the exploder terminal and the exploder handle was screwed in. Flame safety lamp No. 10 was lying on its side nearby. Examination of this lamp in the lamproom showed it to be undamaged, in safe condition and working order. The internal relighter functioned perfectly.

Nearby were lying the bodies of a Learner Miner and five workers. These bodies were badly burnt.

There is no doubt that this heading was the origin of the explosion and that the explosion was initiated by the blown-out shot which was fired by the uncertificated Learner Miner or under his direct supervision by one of the Assistants with him.

When the blown-out shot occurred it ignited an explosive methane/air mixture which had accumulated in that heading. This firedamp explosion was accentuated by coal dust from the sides and floor of the heading, which was raised in the van of the flame.

Observations made led to the conclusion that flame spread in some directions for a distance of at least 120 metres from the entrance to the heading. Coal dust did play a part in the explosion but it is thought not a major part. It seems that the naturally wet condition of the section plus the stonedusting which had been carried out did much to arrest the explosion and prevent its propagation. It would seem that an explosive mixture of firedamp and air was present in the roadway from at least 30 metres from the blown out shot. If the brattice doors across the tramming road to the left at D26 and D28 were torn or destroyed

prior to the explosion, there may well have been a considerably larger body of an explosive methane/air mixture lying in the left hand side of the section.

Careful examination of the heading where the explosion originated failed to reveal any sign that brattice had been erected to carry ventilating air into the face. The supports in the heading had not been disturbed and an empty tub 12 metres from the face had not been derailed. Altogether signs of violence in the heading were markedly absent.

It is quite certain that if a brattice had been erected down that roadway, even to within 10 metres of the face, there would have been found unmistakable proof that it had been erected there.

It is significant that a roll of new brattice 18 metres long, apparently unaffected by the explosion, was lying at the corner to the right coming out from the place. This brattice had not been in use.

The only possible conclusion is that no steps had been taken to direct a current of ventilating air into the end where the shot was fired. Nothing was found to suggest that workers were the victims of circumstances beyond their control in that a sudden outburst of firedamp occurred in that heading just before the shot was fired.

Normally if a properly charged and tamped shot hole is fired in undercut coal the coal is brought down without the shot blowing out even if the cuttings have not been removed from the undercut.

Figure 31 shows the disposition and amount of explosives and tamping in the shot holes on either side of the blown out shot. The deplorable manner in which these shots were charged would make them extremely liable to blow out if fired, mainly because of the insufficient use of tamping. The Mines and Works Regulations demand that in a fiery mine, shot holes be filled with tamping to the collar of the hole.

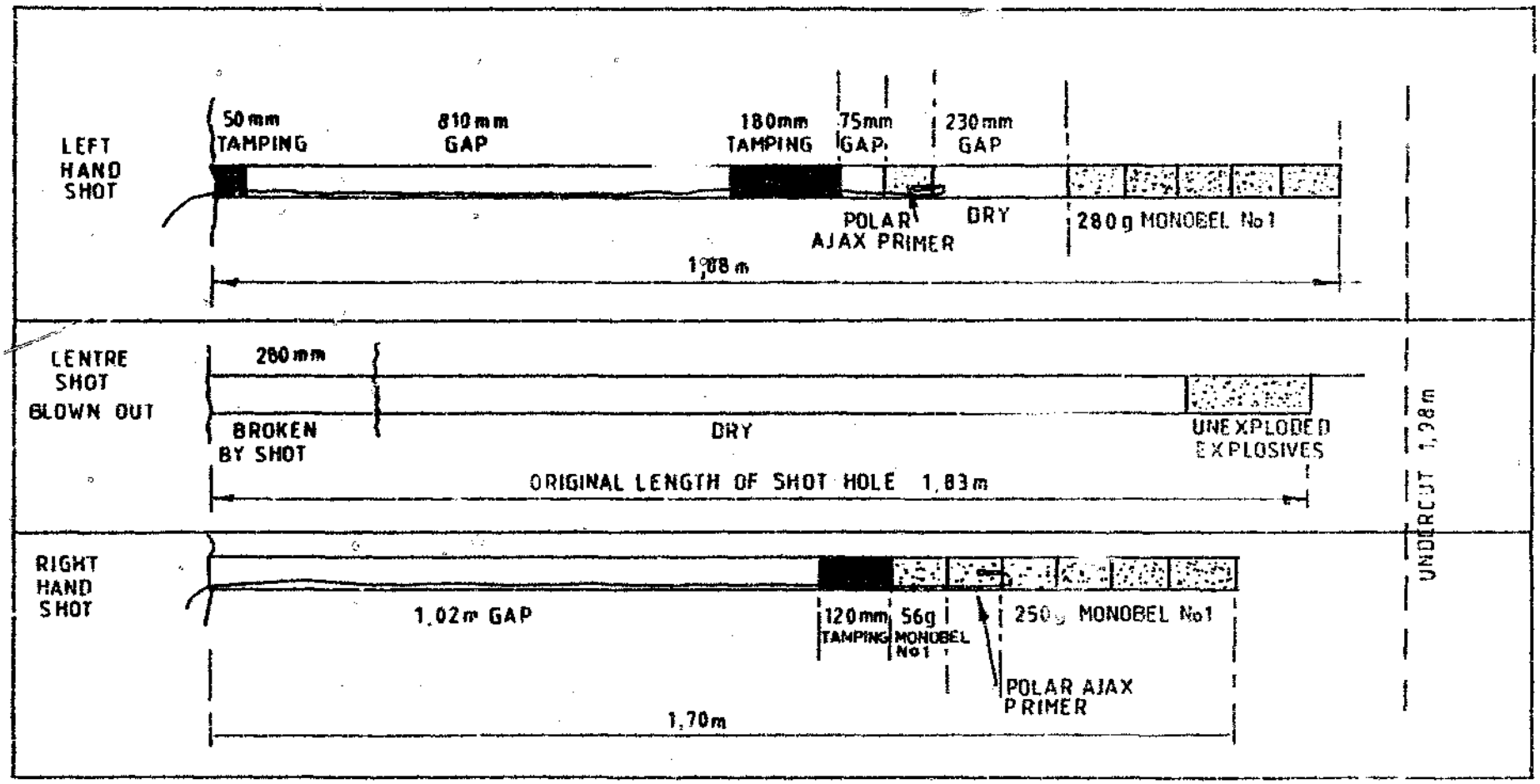


Figure 31
NORTHFIELD COLLIERY
DETAILS OF THE THREE SHOTHOLES IN THE FACE WHICH EXPLODED

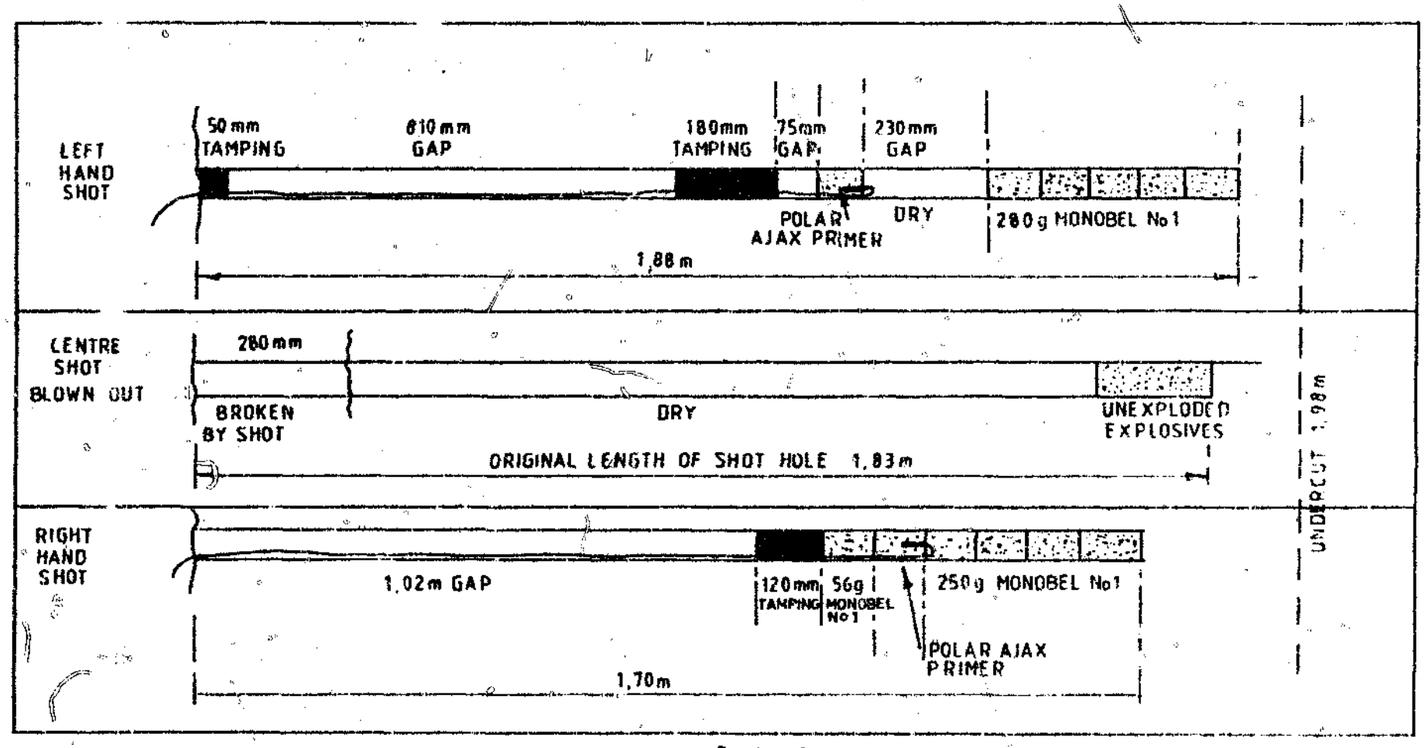


Figure 31
NORTHFIELD COLLIERY
DETAILS OF THE THREE SHOTHOLES IN THE FACE WHICH EXPLODED

Air gaps between cartridges, and the detonator so positioned that the primer cartridge is at the top of hole, are known to be causes of misfires, and could easily be the cause of setting a portion of the charge alight instead of detonating it, particularly if the shot had been standing charged for some considerable time.

It is possible that in the case of the centre hole not only did a blow out occur but also that explosives remained burning in the hole. This would make the ignition of an explosive mixture of methane and air virtually certain.

In the roadway between C and D (see Figure 29) a box was found containing explosives. The lid was unattached to the box and was lying half across it. The lock had been snapped to in the staple.

There was no shortage of tamping material in the section, as a further supply was found at D29 (see Figure 29) further along the road.

Samples were taken at the spots marked on Figure 29 (Plan of No. 0 Section). Where coke was found it was generally as a friable deposit not more than 5 mm and often less than 2 mm thick, and on the lower third of props. An occasional prop showed a few globules of coke spattered on to the side facing the direction of force, which globules were only visible when a layer of uncoked dust, deposited later, was blown away."

During the rescue and recovery operations in Section 0, which took approximately 5 days, methane concentrations of 5% were detected in the main drive as far back as the air crossing at the No. 0 section station.

During May 1943 gas was reported on two occasions in Section 0 and on the night prior to the explosion the night shift Shiftboss had reported gas present in the drive where the explosion

occurred. A disturbing feature of the evidence is that the night shift Miner failed to report the presence of gas on the left hand side to his Shiftboss at the end of the shift and, furthermore, stated that the Shiftboss had not visited his section that night (25th May 1943). The Shiftboss (supported by two witnesses in the section) stated that he had visited the section. Furthermore Miners were not kept informed of problems experienced on the opposite shift. The night shift Miner's blasting Assistant claims, in direct contradiction to the Miner, that he did not help the Miner tamp the 3 holes in the face where the explosion occurred. The Miner's arrogant and careless attitude is exemplified by his remarks under cross examination.

"What experience have you had as a miner? - You people should be glad that you have a man of my description for a Miner; a man who can look after things."

The Miner further stated under cross examination: "I think the best we can do is to forget this (the disaster) and to carry on. As far as I can see no one need worry and it is not due to a fault by anyone."

The brushing lip as shown on plan No. 29 was 100 metres back from the face of the main. Apart from the fact that this state of affairs prohibits the extension of the haulage and thus affects tramming efficiency, it militates against the erection of the permanent brick stoppings up to the second last drawing road thus seriously affecting the section ventilation to the working faces. The section Miner who was at the brushing lip at the time of the explosion had handed the blasting keys to the Learner Miner. The height of the main haulage at the brushing lip (2,1 metres) was clearly more comfortable than the faces themselves at 1,4 metres height.

The Learner Miner who blasted the shot holes had only worked on the mine for 5 weeks. His previous experience had been on gold mines.

There were 7 double shifted sections at the colliery. Two production Shiftbosses, one on dayshift, the other on night shift, were responsible to the Mine Overseer. Evidence indicates that the Shiftboss was unable to inspect all 7 sections in a shift. Those sections he was unable to visit were supposedly to be visited by the Mine Overseer. Another aspect of the supervision was that the night Shiftboss examined Section 0 without taking the Miner with him through the section.

This is exemplified by Court evidence: "When you (the Shiftboss) found gas in these places and then in the main you met the night shift Miner? - Yes.

What did you tell him? - I told him to go and extend the brattice in the last two places as there is a little gas in there.

And his reply was? - He said nothing except that he would attend to it.

You did not take him to the place? - No.

Could he understand from your explanation exactly where the two places were? - Yes."

The cross examination of the day Shiftboss reveals poor mining practices.

"The evidence is that a worker does the charging up, and the testing for gas is unsatisfactory, and an uncertificated man does the blasting; is there no means by which you as Shiftboss can keep check on this sort of thing? - It is not possible for the Shiftboss to remain in one section all the time, and as I have just said they never know when I will come into the Section.

It seems as if they did not particularly care about that according to the way they broke the regulations, and it is the

duty of the Mine Officials to keep some check on them? - I know that and we do our best to ensure it is done."

e. Conclusion

Three shot holes were charged up in a split off the left hand drawing road by the night shift Miner. The face had been undercut by the coal cutter but the cuttings had not been removed from the cut. This aspect led to an almost "solid" face. The roadway was developed for 32 metres past the last through ventilation road and no brattice had been erected in this road to remove methane which was issuing from a crack in the floor and slips in the roof.

On the following day shift, an uncertificated Miner blasted the inadequntely tamped centre hole which blew out igniting methane, causing an explosion. Evidence was that the section had not been examined at the commencement of the shift by the Miner in charge.

The brushing face was 100 metres behind the faces which added to the difficulty in ventilating the section notwithstanding the fact that adequate ventilation was being directed into the area.

Supervision appeared to be lax and stretched with one Shiftboss supervising 7 sections.

4.7 SUMMARY OF THE PRACTICES REVIEWED

1. There are inherent dangers in allowing apprentice and learner Miners to conduct blasting operations. Because of their inexperience in dealing with methane and its danger, coupled with a lack of knowledge generally of coal mining (including the use of permitted explosives), they will tend:

   - to take chances;
   - not to carry out appropriate tests for methane;
   - to charge and tamp holes incorrectly;
   - to ignore the need for two free faces and the cleaning out of duff from the undercuts;
   - to neglect the erection of brattices or ventilation ducting right up into the faces and so dilute and remove the methane emissions.

2. Senior Mine Management allow non-certificated Miners to blast and supervise section operations, either without a qualified Miner being present, or under such a person's supervision.

3. The mine structure vested a supervisory role and crucial responsibility for safe working conditions in White mine workers, who in many instances, did not live up to their responsibilities. This remark applies equally to Shiftbosses and Mine Overseers.

4. The attitude "Do as you please regardless of risk as long as you get coal out at a low monthly cost way of running the mine".

5. Poor training of Supervisors and Miners with special reference to blasting operations.

6. Incorrectly positioned shot holes in relation to the undercut and overburdened shot holes.

7. Shot holes drilled into the solid and over solid ribs of coal.

8. Incorrect positioning of the primer charge and explosives in a shot hole (top or collar priming versus indirect or bottom priming).

9. Precompression of explosives.

10. Use of non-permitted explosives in a fiery coal mine.

11. Charging up shot holes and then leaving them for the next shift to blast.

12. Allowing roof or floor brushing in sections to fall behind schedule and thereby delaying the installation of belt, haulage and electrical extensions.

13. The consequential effects of item 12 is that permanent ventilation appliances are not installed timeously and this affects ventilation efficiency in a section since temporary ventilation appliances tend to leak to a greater degree than permanent brick stoppings.

14. As a result of item 12 tramming distances from these faces to the tip point become attenuated - this has the effect of lowering production tempos and morale. Miners also tend to take chances in an endeavour to ensure that an adequate supply of loose coal is always available.

15. To allow work to proceed, especially blasting operations, in unventilated roadways is a recipe for a disaster.

16. Workings to the rise will always accumulate methane more readily than those on the level or dipping towards the face.

17. The statistic of kilograms of stonedust applied per ton of coal mined is not necessarily a guide to the effectiveness of stonedusting in preventing a coal dust explosion.

18. Incorrectly tamped shot holes and the use of the wrong tamping material.

19. Incorrect use of millisecond delay detonators.

20. Eventually the most important aspect of blasting accidents is the failure of Senior Mine Management to ensure that all levels of section supervision test for the presence of methane in the face where the shot holes are to be fired and all other contiguous faces.

21. Not sampling the mine at regular intervals for stonedust/incombustible dust analysis.

22. Senior Mine Management and Supervisors, during the course of their inspections underground, who see that work is not being carried out safely and yet do not "pull up" the culprits, turn a blind eye to the practices and allow the work to proceed as though nothing is at fault.

23. Supervisory effectiveness is never increased when production sections encounter a deteriorating position in poor ground conditions, increased methane emissions resulting from excessive slips and fault planes and burnt coal zones or increased water emissions.

4.8 PRECAUTIONS TO BE ADOPTED

1. The temptation, when Senior Mine Management are short of qualified blasting certificate holders, to use Apprentices to blast in sections is great. It is only a commitment to safety and sound discipline that will ensure that this dangerous practice is stopped. Vigilance from first line Supervisors is necessary to uncover these malpractices in the first instance and then stop them.

2. Proper and adequate training of Miners and Supervisors both on and off the job will ensure that they are able to appreciate the dangers attached to blasting particularly where methane is being given off freely. The importance of ensuring that adequate ventilation is delivered to each face cannot be stressed enough.

   Bridging training is particularly necessary for Miners who are promoted to Shiftboss level. All to often, men are appointed to the position of Shiftboss (and likewise Artisans to Foremen) without receiving any form of training in a Shiftboss' duties.

   A Learner Control Programme for Miners, Shiftbosses and Mine Overseers comprises the following main headings, and aspirant candidates should be required to complete the relevant course in not more than two months after appointment.

   Miners
   Planning and organising the production cycle
   Ventilation
   Supporting
   Cutting

Duff cleaning

Drilling

Charging and blasting

Loading

Trailing cables

Conveyor belt extension and equipment move

Load haul dump equipment

Production shiftboss manual

Shiftbosses

1. - To obtain an overall picture of the latest developments.
   - A means of communicating certain areas requiring attention.
   - Machine performance and availability.

2. - Cables
   - Tractors
   - Load haul dump units
   - Spare wheels
   - Oil

3. Roof:

(i) - Barring	-	Bar to the solid
- Support	-	Install additional support
- Roof control	-	Accurate drilling.
- Planes of weaknesses	-	Additional support
(ii) - Excessive overhanging coal	-	Bar, cut with coalcutter or drill and blast
- Direction of rib side	-	Refer to cutting in
- Barring	-	Bar to the solid
- Suspension of cables	-	Suspend according to mine standards

(iii)
- Slight overhanging coal – Cut the overhang with the coalcutter
- Excessive overhanging coal – Drill and blast overhang; do not cut face
- Direction lines – Sight through survey pegs
- Height excavation – Correct drilling of top holes
- Sockets and misfires – Depth of holes. Misfires; treat as per Regulations
- Depth hole burden – Drill holes to mine standard
- Depth of drill hole and direction – Measure the depth of cut and drill on line

(iv) Floor:
- Unnecessary equipment – Remove and store in correct place
- Floor coal – Drill and blast floor coal
- Excessive loose coal – Sweep coal up with loader
- Excessive water – Pump water
- Trailing cables – Suspend according to mine standards

4.
- Stonedusting
- Tramming roads and route
- Shuttlecar changeout points
- Shuttlecar anchor points
- Conveyor tip cleanliness
- Cable suspension
- Back-up machines
- Maintenance bay – spare oil, wheels
- Waiting place and first aid
- Explosive magazine
- Stores and material
- Brattices
- Water pipes

Mine Overseers
A concept of management
Planning
Cost control
Mechanised equipment statistics
Production potential in mechanised bord and pillar mining
Grade Control

It is the author's view that successful completion of such a course will help supervisors to carry out their responsibilities in a reliable manner.

2. The technical aspects of explosive and their use in breaking ground is not sufficiently covered in the training of all levels of Mining Supervisors and Mining Engineers.

Figure 32 depicts a properly charged shot hole in a fiery coal mine.

- The shot hole is 150 mm short of the end of the undercut (prevention of sockets and blown out shots).
- The shot hole has been cleared of any loose drill cuttings by running the charging stick up and down inside the hole.
- The hole has been bottom primed.
- The correct amount of explosive has been loaded into the hole.
- The clearance between the diameter of the hole and explosive diameter is a minimum to reduce the effects of coupling.
- All explosives are pressed together and not separated by coal dust or other foreign mater .

- The correct stemming material is used and is pressed against the explosive charges. Cushion blasting is not permitted.

- The undercut is cleaned of all cuttings thereby providing a space to allow the blast to expand.

- If clay stemming is used, the legal requirements are that the hole is to be tamped to the collar and this should be done. The author has witnessed many injuries to persons when an incorrectly and insufficiently stemmed shot hole has been blasted and clay stemming has been ejected from the blast at high velocity.

Experience will indicate whether a face has been properly stemmed or not simply by listening to the tone of the blast; a dull dampened sound generally indicates an effective blast - a loud bang usually results in a poor blast with the possibility of a blown-out shot.

4. Senior Mine Management should pay particular attention to detonation-driven air shocks in the air gap between a charge and its confinement. Explosives can become desensitised as a result of this problem, thereby leading to deflagration or unexploded particles remaining in the hole.

When explosives are detonated in a geometry where there is an air gap between the explosive charge and its confinement an air shock in the air gap travelling ahead of the detonating front can be generated. The resulting precompression of the explosive, also called the channel effect, has been studied by several researchers in the past; Johansson et al, (1958), (1), Goldbinder and Tysavic (1967), (2), Dubnov and Kohma (1966), (3), Udy (1979), (4), Nakano and Mori (1979), (5) Ranasak et al. (1982), (6) and Lownds and Plessis (1984), (7).

In practical rock blasting for example, long explosive column charges are often decoupled from the borehole wall by an air gap to reduce borehole pressure and vibrations on nearby structures and damage to the remaining rock wall when the final contour is blasted. The precursor air shock (PAS) travelling ahead of the detonation front in the air gap in the borehole will precompress the explosive ahead of the detonation front to a pressure of 10-100 MPa. If the PAS is reflected by some rigid surface normal to the flow in the air gap, the local pressure will be in the range 100-1000 MPa.

Figure 33 illustrates the PAS in the air gap between a cylindrical explosive charge and the borehole wall.

Depending on the type of explosive, the precompression caused by the PAS can have different effects on detonation propagation. Most explosives rely on an initiation mechanism, meaning that they need hot spots distributed in the explosive to propagate a detonation. These hot spots can be compressed if they are of the void type, for example air voids or gas filled micro spheres.

Depending on the type of explosive, the precompression can either decrease or increase the velocity of the detonation front. The case when D is reduced by the precompression can lead to an oscillating detonation velocity or detonation failure (dynamic dead-pressing). A precompression of 30-100 MPa will desensitise many commercial explosives relying on hot spots of the void type to such an extent that they cannot propagate detonation. If D is increased by precompression the propagation of detonation is not endangered but an oscillating behaviour can occur.

5. Dangerous practices which are practiced in fiery coal mines include:

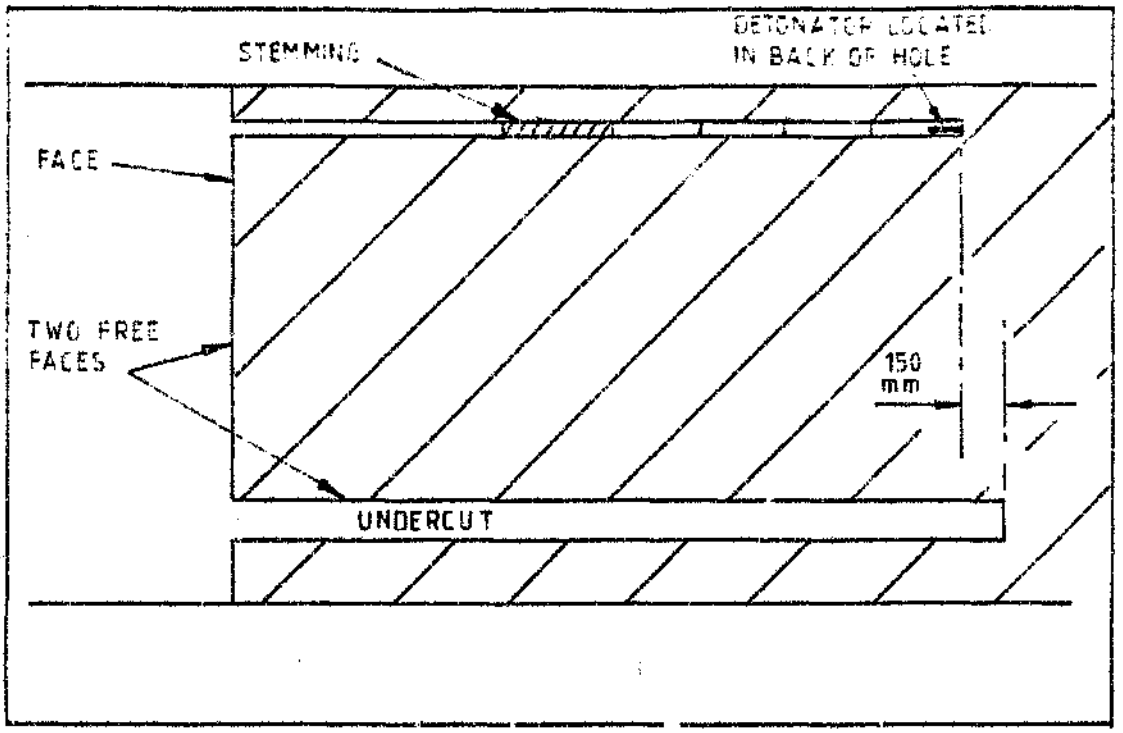


Figure 32
PROPERLY CHARGED UP SHOT HOLE

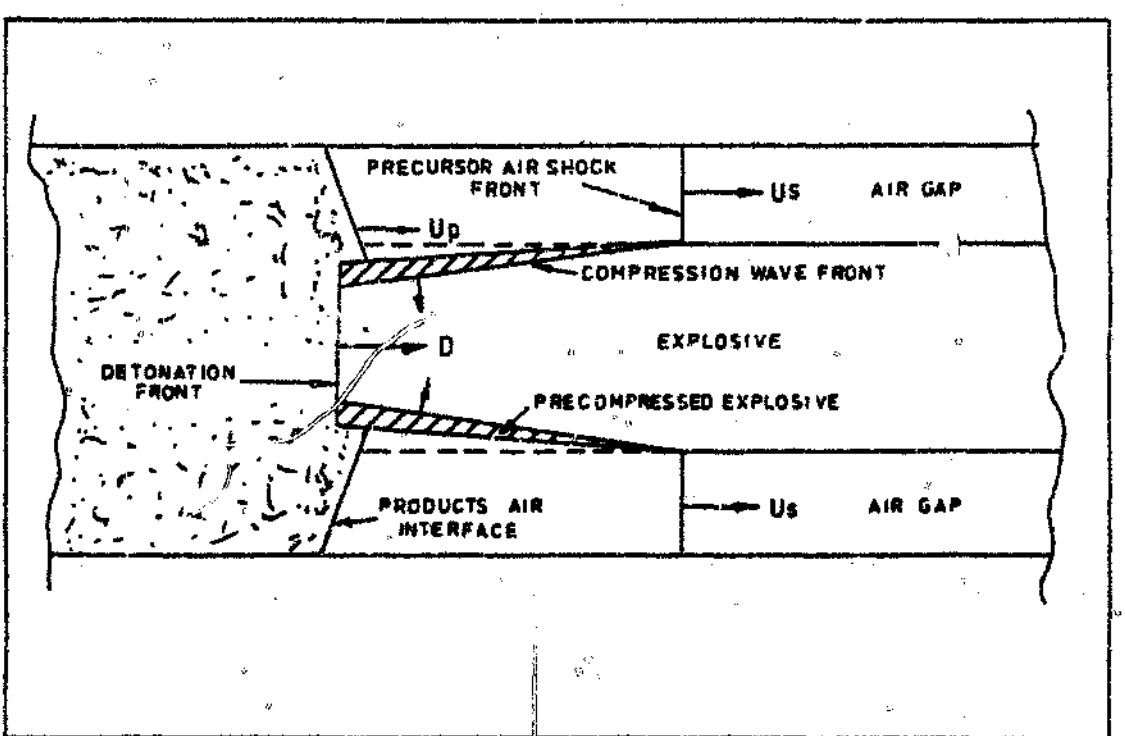


Figure 33
DETONATION DRIVEN AIR SHOCKS

- Breaking cartridged explosives in half and inserting the one broken half into a short shot hole so as to reduce the power of the blast.

- Mud blasting - placing an explosive charge on top of a large piece of coal or rock and covering this charge with a clay pack and then setting off the charge thereby breaking the coal or rock into smaller pieces. Hot gases and particles are likely to escape into a gas laden atmosphere and ignite an explosive mixture.

- Removing goaf line timber, steel pins in crawler gear and other obstacles by tying cartridges of explosives around the object can, due to the shock wave initially set up by the blast, lead to coal dust explosions. (Hlobane Colliery, No. 2 Pit, March and April 1952 - IMNA/232/52 and 245/52).

- Using coal dust as a tamping material can also lead to coal dust explosions as a result of blown out shots. The coal dust does not have the propensity to stem the force of the blast thereby allowing hot gases, particles and flame to escape from the shot hole. Evidence is that the initial shock wave raises a cloud of fine coal dust which is ignited by the explosive charge. (Hlobane Colliery, NO. 2 Pit. March and April 1952 - IMNA/232/52 and 245/52).

In all the above cases cartridges are detonated while in an unconfined state.

6. It is a recurring theme in this research investigation that insufficient attention is paid to the degree of supervision, quality and training of Shiftbosses and Mine Overseers. In both incidents referred to in this hapter the Shiftbosses were not only extended insofar as the areas they were required to supervise, but were also required to carry out blasting operations in sections.

This observation is supported by a letter sent to a Manager by an Inspector of Mines, a copy of which is recorded below.

"During visits by Inspectors of Mines at your Colliery, it has been found that the two Shiftbosses employed at your colliery cannot reasonably be expected to carry out the duties assigned to them under the Mining Regulations.

I, therefore, require, as provided in Regulation 161(1) the appointment of an additional Shiftboss, three in all.

I must also remind you that the practice of allowing a Shiftboss to undertake Miners' duties is illegal, except temporarily in case of necessity. This is not to be taken to allow the Shiftboss to take on a Miner's duties during absence on leave, and so on. Provisions for such a contingency must be made by the employment of a spare Miner.

Kindly sign and return the attached copy of this letter in acknowledgement of receipt."

As a rule of thumb the following is generally regarded as the upper limits of production which can be expected from Supervisors where safety plays a major ongoing role:

- Gassy pits noted for difficult conditions.

  ± 80 000 ROM tons/month production per Mine Overseer.

  ± 20 000 ROM tons/month production per Shiftboss.

- Pits which are not particularly gassy and where conditions are favourable (good roof and high seams of 2,5 metres).

  ± 120 000 - 180 000 ROM tons per month production per Mine Overseer.

± 60 000 ROM tons per month production per Shiftboss.

In gassy pits with low seam heights (± 1,4 metres for example), it would be advisable to increase supervision to the extent that a Mine Overseer is responsible for a monthly production of 50 000 R.O.M. tons per month.

When conditions underground change for the worse, it is incumbent on Mine Management, in the interests of safety, to increase supervision albeit temporarily.

7. In South Africa as recently as the 1950s, shot holes in coal faces, irrespective of the seam height, were normally detonated singly; the top holes being fired first, the top cut loaded out and then the bottom holes fired singly.

Gas released during the first round of firing could easily be ignited by the second, especially as gas detection was difficult amidst the dust, fumes and smoke in the aftermath of blasting. In addition, when firing shots singly, it was more often than not the case that the Miner would not enter the face to re-test for methane - blasting attendants would be sent into the face to reconnect the detonator wires to the blasting cable.

This problem was solved, to some extent, by the introduction of short period delay detonators (millisecond) to fire all the shots in one round simultaneously.

8. Charging

It will be clear that all the modes of ignition are rendered least likely by the choice of the appropriate explosive of no more than fully adequate strength at a weight well chosen for the burden; because, in this way, the initial temperatures are at a minimum and, further, there is not more than enough (although adequately so) explosive to do

the work. Hence, the cooling during expansion is at an optimum and the gases are reduced to the minimum practical temperature by the time they burst through to the atmosphere so that there is the minimum of energy available for ignition processes. Finally, the conditions described are those most conducive to the full completion of the chemical reaction virtually within the borehole so minimising the possibility of reacting explosive making contact with the mine air.

Needless to say, the preceding paragraph has assumed good stemming and freedom from breaks and partings.

9. Stemming

The function of stemming in the process is clear enough and, clearly also, a weak stemming is dangerous, because it does not inhibit the ignition processes outlined in the previous paragraph (in paragraph No. 8).

It is general knowledge that damp, gritty materials with some binder make very good stemming and that relatively greasy substances such as damp clay are bad. A revealing experiment made by Audibert in 1928 has a Gallic flavour. He put a grape at the bottom of the bore of a cannon mortar and separated it from an explosive charge by a buffer of stemming. When the stemming was clay the firing of the smallest charge crushed the grape but with a sand stemming of length appropriate to the charge, it was undamaged.

Water is a particularly effective anti-incendive material when put in the shot hole; it has been introduced by soaked porous matter, by infusion and, more recently, in plastic ampoules. In some circumstances it is effective as the sole stemming material.

10. Breaks and Partings

Breaks and partings across and along the shot hole respectively have long been recognised as dangerous. The impairment along the confinement of the shot is enough to introduce all the three modes of ignition. However, there is good evidence to show that firing into narrow gaps is particularly hazardous. This enhancement of hazard is likely to be due to the reduced attentuation that the shock and pressure waves will meet in such circumstances. If one thinks of a cartridge exploding into free space with all the products disseminated into the full solid angle, it is evident that the density of the products will fall off quickly with distance compared with what will happen when the discharge is confined to a break, and although this is but one factor, it is likely to be an important one.

It is well known that the ideal, indeed, the correct course is not to fire shots traversed by breaks or partings, but nevertheless it sometimes happens that shots are fired in these circumstances.

Pulsed infusion in which the shot holes are filled with water when the shots are fired is a first class safeguard because of the inhibitive effect of water on the ignition of methane. Unfortunately, the method seems not to be liked everywhere. Another effective method is to pre-inject the shot holes with a stiff foam.

Finally, the fourth-class explosives were developed for firing under conditions in which breaks and partings are to be expected and these explosives must pass a severe test to demonstrate their safety under such conditions. Incidentally, the most successful formulations have so far been based on the exchanged ion principle.

11. Deflagration

Deflagration is a mode of combustion in which an explosive which should detonate, may decompose in certain circumstances. As implied by the name, deflagration is relatively slow burning and is a highly incendive mode of reacting. It appears when an explosive is submitted to a certain range of temperature and pressure and yet fails to detonate.

Until recently, deflagration was very rare in the United Kingdom although not so infrequent in Europe. This difference in experience reflects different explosive practice because deflagration in modern explosives is virtually non-existent except in multi-shot delay firing, a desirable practice which has been common in Europe for some time but which is still uncommon in the United Kingdom, although this is changing.

Deflagration appears in practical use when relatively weak detonative explosives are used in conditions which render them less sensitive or which impede the transmission of detonation from one cartridge to the next. This description fits delay firing because this method requires the use of very safe explosives (which tend to be the weaker explosives) and of course the method of firing inevitably submits later shots to compressive forces from prior shots in the round; pre-compression of a cartridge reduces its capacity for detonation. However, it is probable that the major cause of deflagration is the presence of a gap between adjoining cartridges in the same hole and with the gap filled, or partially filled, with coal dust (Northfield Colliery disaster, 1943).

In Europe, the difficulty appears to have been overcome entirely by supplying the explosive in a continuous plastic sheath. The shotfirer cuts off a length containing the

number of cartridges required, fixes the detonator, lets the cartridges fall to one end of the sheath and then loads the resultant charge into the hole. By these means, the cartridges are maintained end to end and protected from both dirt and water and, according to report, the method is highly successful both in safety and in effective blasting. So far the method is not regarded with favour in the United Kingdom.

4.9 CONCLUSIONS

The development of permitted explosives (nitroglycerine granular type and the more recent emulsion explosives) has over the years reached a state where, when used correctly with millisecond delay detonators, will seldom, if ever, result in a methane ignition when detonated in an explosive atmosphere underground.

The problem arises when, as a result of poor training, insufficient experience, and a lack of attention to detail and discipline, Miners drill shot holes off line, longer than the depth of the cut, and then charge the hole with poor coupling of explosives, primers incorrectly placed and insufficient stemming. Frequently Mine Management pay little, if any, attention to the relationship between hole diameter and cartridge diameter which further aggravates the problem causing desensitisation of the explosives.

These omissions provide the recipe for a disaster and when carried out in the presence of methane lead to the unsafe practices referred to in this chapter.

Vastly improved training of Mining Engineers in the use of explosives underground will increase their awareness of the dangers attached to blasting. Improved supervision will ensure adequate ventilation to faces and considerably reduce the risks of an explosion as a result of shotfiring.

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mathematically quantified. It must also be realised that the intrinsic factors are by definition uncontrollable. Therefore if one wishes to prevent self-heating incidents it is necessary to establish the severity of the intrinsic factors and then design the extrinsic elements in such a manner as to avoid unsafe combinations.

The various factors involved are briefly summarised on Tables 2 and 3.

Table 2

Intrinsic factors favouring spontaneous combustion

Coal Constituency	Geological Environment
Low rank	Thick seams
Rich in reactive macerals	Highly faulted
High sulphur content	Poor caving characteristics
High alkaline content	High virgin coal temperature
High porosity	Outburst prone
Easily friable	Carbonaceous strata or roof

Table 3

Extrinsic factors favouring spontaneous combustion

Accumulations of broken coal
Stress distribution causing fracture
Low production rates
Limited ventilation through worked out areas
Heat from machines
Relative humidity differential between air and coal

These intrinsic and extrinsic factors highlight the inherent problem of spontaneous combustion in burnt coal zones in the South Rand and Natal coalfields where factors such as "easily friable", "highly faulted", "outburst prone" and "stress distribution" are all present. Eventually the coal ignites and provides an ignition source, when the methane/air mixture is within the explosive range.

5.4 INCIDENTS

The major incidents involving fires in the goaf or at the goaf edge may be listed as follows:

5.4.1 NUMBER AND DATE OF INCIDENTS

Glencoe Colliery. February 1908. Brattice cloth caught fire after a methane explosion caused by blasting. 77 killed and 8 injured.

Cambrian Colliery, June 1908. Almost identical to the Glencoe Colliery disaster. Nil injured. Great damage to mine.

Natal Navigation Colliery, 1912. Spontaneous heating of top coal in the goaf led to a fire which caused a series of firedamp explosions. No casualties.

Natal Navigation Collieries, No. 3 pit. 1917. Ignition of gas due to spontaneous combustion in goaf. No casualties.

Consolidated Collieries. October 1945. Ignition of gas caused by heavy caving in stooping section. 1 killed.

Northfield Colliery. November 1951. Spontaneous combustion in goaf. 3 killed and 2 injured.

Hlobane Colliery - No. 2 pit. March and April 1952. Blasting out goaf line timber. Coal dust explosion. 1 injured.

Northfield Colliery. March 1952. Gas concentration in goaf - ignition sources - matches. Nil injured. April 1952. Gas concentration in goaf - 1 killed and 5 injured.

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CHAPTER 5 IGNITION OF FIREDAMP AND COAL DUST BY SPONTANEOUS COMBUSTION INCLUDING HEATINGS IN THE GOAF AND MINE FIRES

5.1 INTRODUCTION

5.2 SPONTANEOUS COMBUSTION

5.2.1 Factors governing spontaneous combustion

5.3 MINE FIRES AT THE GOAF EDGE AND WITHIN THE GOAF AREA

5.4 INCIDENTS

5.4.1 Number and date of incidents

5.5 DETAILS OF INCIDENTS

5.5.1 Fires resulting from methane/coal dust explosions

(a) Glencoe Colliery 1908
(b) Durban Navigation Collieries 1965

5.5.2 Methane explosions in the goaf as a result of spontaneous combustion

(a) Northfield Colliery 1951

5.5.3 Methane and coal dust explosions as a result of blasting and electrical faults on the goaf edge of pillar extraction sections

(a) Hlobane Colliery No. 2 Pit 1952
(b) Cambrian Colliery 1956
(c) Belgray Colliery 1979

5.6 SUMMARY OF PRACTICES REVIEWED ABOVE

5.7 PRECAUTIONS TO BE ADOPTED

5.7.1 Bleeder roads at the furthest extremity of the goaf
5.7.2 Methane drainage from the goaf through to surface

(a) Results obtained from recent studies at Springfield Colliery
(B) Conclusion

5.7.3 Prompt sealing of fires
5.7.4 Section ventilation appliances
5.7.5 Fires
5.7.6 Supervision
5.7.7 Spontaneous combustion in the goaf
5.7.8 Systematic rehabilitation of airways

5.8 CONCLUSIONS

5.9 REFERENCES

5.1 INTRODUCTION

Pillar extraction or stooping has been widely practiced in South Africa since the turn of the century. The techniques used to achieve maximum extraction of the coal seams vary from the complex double seam stooping practices in Natal to the most modern method of rib pillar extraction and finally to longwall mining using powered supports. The goaf or waste is that part of the section where the pillars of coal have been partially or totally extracted and the superincumbent strata allowed to collapse into the resulting void. The goaf line is that line across the section where the faces being worked to recover the pillars lie adjacent to the goaf itself; this line may lie at 45° or across the section at right angles to the barrier pillar.

Pillar extraction methods account for 9% of South Africa's annual coal production.

Many serious methane and coal dust explosions have occurred on the goaf line and in the goaf itself as a result of fires arising from spontaneous combustion in the goaf and fires caused by methane explosions which explosions have been initiated by blasting, naked lights and electric faults at the goaf edge.

Unless proper precautions are observed in stooping sections the goaf area becomes a reservoir of methane gas which can be ignited in the goaf; or the methane is evacuated, by falls of roof or barometric pressure changes, into the workings on the goaf line.

By its very nature, especially when it is worked at a 45° angle to the line of retreat, the goaf line is difficult to ventilate effectively.

Frictional sparking caused by the caving process is also a cause of methane explosions but this has been dealt with in Chapter 2.

The causes of several serious explosions have been examined and then summarised. Examples of explosions as a result of mine fires not in stooping sections are dealt with also. Precautionary measures to be adopted to prevent explosions under these circumstances are then discussed.

5.2 SPONTANEOUS COMBUSTION

A study of methane and coal dust explosions in South Africa initiated as a result of spontaneous combustion reveals only six incidents; a small percentage of the total number of methane and coal dust explosions recorded. Yet spontaneous combustion in the goaf remains a danger for the following reasons:

- invariably the heating proceeds undetected;
- the goaf or waste areas are inaccessible and hence mine officials are unable to deal with the resultant fire;
- usually large quantities of coal or burnt coal are involved making it difficult to douse the fire with water - which in turn could lead to the production of dangerous quantities of hydrogen (water gas) (Morris 1987);
- much valuable time is sometimes lost due to poor roof conditions in attempting to load out the burning coal;
- ventilation of goaf areas is always difficult and in many instances the methane/oxygen mixture passes into the explosive range resulting in successive explosions. In addition ventilation may be just sufficient to provide oxygen but insufficient to remove the heat generated due to the oxidation process;
- spontaneous combustion is classified as a natural fire (as opposed to man-made fires). (Morris 1984).

Table I depicts the type of fires which have occurred on an area basis between 1970 and 1985.

Table I

Category	Witbank/ Ermelo	Natal	Vaal Triangle	Total
Spontaneous Combustion	7	25	38	70
Electrical	15	31	9	55
Methane	12	10	4	26
Cutting and Welding	1	-	2	3
Others	9	2	1	12
Unknown	2	1	-	3
Total	46	69	54	169

5.2.1 Factors governing spontaneous combustion

Generally spontaneous combustion occurs where large quantities of broken coal occur coupled with ventilation which is sluggish - in other words, sufficient oxygen is available to support the oxidation process but the flow of air over the coal favours the accumulation of heat rather than removing it.

Spontaneous combustion of coal is the cause of mine fires and is related to a coal oxidation phenomenon. This complex reaction is exothermic. If the heat of oxidation is not sufficiently dissipated, the coal heats up. And since the rate of oxidation increases exponentially with temperature, the heating rate accelerates. If a steady state is not reached between the heat produced and the heat dissipated, the initial self-heating will lead to spontaneous combustion and give rise to a mine fire.

Wade, Phillips and Gouws (1987) in a paper entitled "Spontaneous combustion of South African coals" have stated:

"The degree to which these conditions are satisfied depends on intrinsic factors, such as the coal constituency and the geological environment, together with extrinsic factors associated with the mining and ventilation method. The various component factors are well known but are as yet not

5.3 MINE FIRES AT THE GOAF EDGE AND WITHIN THE GOAF AREA

Mine fires which occur in the goaf or on the goaf edge may be classified as follows:

- flames from blasting operations;
  - explosive flames
    - blown out shots
    - shotholes holed through into goaf
  - detonator flames
- flames and sparks from faulty electrical gear including auxiliary fans with faulty motors;
- fires generated from methane and coal dust explosions which subsequently set alight
  - brattice cloth
  - timber
  - loose coal
- incendiary sparking (this has been dealt with in Chapter 2);
- lightning and stray currents (this has been dealt with in Chapter 3);
- matches and faulty flame safety lamps.

Natal Cambrian Colliery. July 1956. Methane concentration at goaf edge ignited by faulty auxiliary fan. 2 killed.

Northfield Colliery. June 1962. Methane ignition in goaf as a result of blasting. 10 killed and 12 injured.

Durban Navigation Collieries. September 1965. Coal caught fire after methane ignition in face. Nil injuries.

Northfield Colliery. February 1969. Explosion in goaf of stooping section. Nil injured.

Balgray Colliery. April 1979. Blasting fenders on goaf edge and resultant fires. Three successive methane explosions. 2 killed and 9 injured.

Springfield Colliery. February 1981. Methane ignition at goaf edge. 2 killed and 11 injured.

Ermelo Mines. October 1984. Methane ignition in goaf - lightning and stray currents. 6 killed and 18 injured.

Springfield Colliery. June 1985. Methane ignition in goaf. No injuries.

5.5 Details of Incidents

A discussion and analysis of all the incidents referred to in item 5.4.1 would result in a lengthy dissertation and the author has chosen to examine only 5 of these incidents which are considered most relevant to the project.

5.5.1 Fires resulting in methane/coal dust explosions

(a) Glencoe Colliery 1908

The following extract is taken from the Natal Government Gazette No. 3726A dated 22nd December 1908.

"Description of the colliery.

The Glencoe Colliery is owned by the Glencoe (Natal) Collieries Limited and is situated about 4,2 kilometres east of the Hattingspruit Station on the Natal Government Railways with which it is connected by a private line.

The maximum number of employees was seven hundred and sixty two. The colliery is worked on the "pillar and stall" system. The seams are intersected by three dykes running in a northwesterly and southeasterly direction which do not, however, throw the seams. The two shafts are three hundred and fifty seven feet (109 metres) in depth, the downcast shaft being twenty one feet ten inches (6,4 metres) by seven feet (2,1 metres).

The upcast shaft is fitted with a single deck cage for use in case of emergency and is connected by a fan drift with a Capel Fan, fifteen feet (4,6 metres) in diameter, the capacity of which is stated to be 120 000 (58 m^3/sec) to 150 000 (74 m^3/sec) cubic feet per minute at 100 revolutions a minute.

The two seams are separated by a band of sandstone varying in thickness from two (0,6 metres) to five feet (1,52 metres). The following is an analysis of a sample of coal taken by the Coal Technical Committee and published in the Natal Government Gazette in Government Notice No. 70, 1906:

No steps were taken by watering or otherwise to minimise the risk which is always present where fine dust exists, and it is evident that the danger which arises from coal dust in mines was either overlooked or not appreciated. This situation is identical to that which existed at the Universal Colliery at Senghenydd in 1901 and 1913 prior to the two coal dust explosions, the latter being the most disastrous incident in British coal mining history.

Ventilation

According to the evidence, immediately prior to the explosion, the mine was ventilated as follows: the quantities of air being those taken at the last measurement made before the explosion on January 14 1908:

The Main North Level was the main intake from the North, North western and North eastern districts, the quantity of air entering this level being 88 230 cubic feet a minute (43 m^3/sec).

The Western district was supplied by a split carrying 22 080 cubic feet (11 m^3/sec) along the Main West, splitting at the end of this road; one split travelling round the faces northward and joining the air from the northwestern district near the end of Ogilvie's Road.

The Eastern district was ventilated by a split from the Main North, carrying 33 600 cubic feet (16 m^3/sec) along the Main East, splitting at the end of that road - one part travelling along the faces to the south and thence along the barrier dyke to the upcast shaft.

The remainder of the air in the Main North Level - 32 650 cubic feet (16 m^3/sec) - travelled northwards to the first stalls outbye the dyke. Splitting there, part travelled westwards round the faces and joined the air from the Main West near the end of Ogilvie's Road returning with it to the upcast.

as had been arranged between the Manager and himself. A commencement of one dam about 0,46 metres high was subsequently found.

About 19h00 a severe explosion took place which caused the death of the whole party of five Europeans and forty Blacks. This explosion was, in the opinion of your Commissioners, due to an accumulation of firedamp in the Main North heading, ignited in a similar manner to the previous explosions, the force of the explosion being accentuated by accumulations of gas in other parts of the Northern Districts of the mine.

The most serious explosion took place at about 08h30 on the 14th February, by which Mr. Muir, the Inspector of Mines, six other Europeans, and twenty Blacks lost their lives, and, in the opinion of your Commissioners, originated from an ignition of firedamp at the seat of the previous explosions. Ogilvie's explosions had broken down the ventilation and caused further falls of roof. This, taken in conjunction with the time that had elapsed - thirteen hours - rendered a much larger accumulation of gas possible. The explosion of gas extended to the coal dust, and was thus propagated over nearly the whole of the mine.

Responsibility - Finding

Your Commissioners find:

That on the morning of the 13th February 1908, there was an accumulation of gas in the face of the Main North Heading, when the Miner Phillips fired the shot which caused the first explosion.

Phillips must, or ought to have been aware that the shot fired on the 12th February, had

a. misfired; or
b. chambered, or fissured the coal,

and met Ogilvie coming up, returning together up the Level. When they were a little inbye the second aircrossing a still more violent explosion happened, which shook the aircrossing. This was about 14h50 and the men then went to the surface."

Some 12 hours after the first explosion, which had ignited the brattice cloth in the North Main section (Phillip's section), a decision was taken to seal off the affected area. The Mine Overseer, Ogilvie, and 44 other employees entered the mine at 20h00 (13th February) with a view to erecting the seals when a violent methane explosion killed the entire party.

The evidence then describes in detail:

- the call for assistance from neighbouring collieries:
- the re-establishment of ventilation appliances which had been destroyed by the explosion as far back as the Main West and East haulages (see Figure 1);
- the fear of another explosion;
- the need to rescue Ogilvie's party;
- the instruction not to attempt the exploration in the neighbourhood of the explosion in the North Main;
- serious accumulations of methane and also coal dust blown into heaps on the haulages by the blast of the successive explosions.

Finally the decision was taken at 07h00 on the 14th February 1908 that Mr. Muir, Inspector of Mines, and a party of 33 persons should enter the mine to repair stoppings between the second and third East and West roads (see Figure 1). The fate of this party is described in the evidence which also highlights the devastation of a coal dust explosion.

Gas continued to be given off by the borehole until the heading was opened up at the end of May 1908. This is the first recorded incident in South Africa in which a borehole was drilled from surface to drain off methane from the workings.

An almost identical set of circumstances occurred at the Cambrian Colliery at Dannhauser in June 1908, and in this instance an attempt was made to seal off the fire. This failed and the whole mine was ravaged by a great coal dust explosion - fortunately all the workmen had been withdrawn from the pit, and no lives were lost. The force of such explosions can be gauged by the remarks of the Inspector of Mines in Natal on the 4th October 1956.

"In 1908 a violent explosion at this same mine blew the cage into the headgear and a second explosion some 20 minutes later blew a fully loaded car of bricks from the shaft bottom (which was 213 metres below the surface) up the shaft and clean through the bottom of the cage on the surface. This was quickly followed by five other explosions."

(b) Durban Navigation Collieries. 1965

An incident at the Durban Navigation Collieries on the 6th September 1965 resulted in a fire which was caused by a methane explosion in a section roadway.

At about 08h00 on the 6th September 1965 whilst two men were drilling a bottom hole, the eighth in a round, on the face of No. 2 Road in Section 25 at the Durban Navigation Collieries, an ignition of methane took place (Figures 2 and 2a). Both Drillers and a Ventilation Attendant who was also in the end at the time, managed to escape without injury and reported the occurrence to the Miner in charge who, together with the Shiftboss, returned to the heading and found a fire burning near the bottom of the face on the left hand side.

General:	Percent
Moisture	1,15
Volatile Hydrocarbons	18,75
Fixed carbon	66,69
Sulphur	0,97
Ash	12,82

Organic constituents:

Carbon	76,27
Nitrogen and Oxygen	5,29
Specific Gravity	1,389

The mine is worked in sections, the coal from the faces being brought to the Main North and South Levels, and thence to the downcast shaft.

Endless rope haulage is used in the Main North and First East and West Roads. The rope was not, however, carried on rollers on the floor (an important issue as will be seen later).

The mine, with the exception of a few isolated places, is dry.

Conditions of the mine

Methane gas was present in considerable quantity in various parts of the mine.

From time to time the roof had been broken down by the pressure of gas and occasionally boreholes had to be put in to drain the gas from the upper seam.

Although the mine may not be considered to be exceptionally dusty, there is no doubt that a considerable quantity of dust was allowed to collect in the main haulage roads, which owing to the traffic and the haulage rope dragging along the floor, was reduced to a fine powder.

The Southern portion of the mine was ventilated by a main split carrying 48 310 cubic feet a minute (23 m^3/sec), which split at the Slope Road.

Causes - Finding

Your Commissioners find:

That on 12th February, a shot was fired on the West side of the Main North Heading, which failed to do its work, but chambered or fissured the coal, causing a cavity in which gas accumulated.

That the shot fired on the morning of the 13th February was charged with three cartridges of "Carbonite" and was intended to remove the coal which had not been brought down on the previous day; and that under the circumstances the charge was excessive.

This shot was fired by a low tension electric detonator fuse, and ignited the gas and dust in the chamber or fissures, the explosion of which was communicated to a further quantity of gas in the heading. Your Commissioners are of the opinion that under these circumstances an explosion would most probably have occurred with any other explosive.

The explosion set fire to the brattícing in the heading which had the effect of stopping the ventilation through the dyke. The fire spread from the brattice to the props and coal. The heat of the fire caused the roof to fall which formed a dome-like cavity in which gas collected. A series of explosions of firedamp, ignited by the fire, followed, each one increasing in violence, the sixth and last of this series which occurred about 14h50 damaging the second aircrossing.

Between 17h20 and 18h00 it was decided to build off the fire, and a party under Mine Captain Ogilvie went underground shortly after 18h00 to carry out the work - the Mine Captain taking with him plans showing the positions at which the dams were to be erected

and that under these circumstances, not only was the charge of three cartridges excessive, but that no shot ought to have been fired.

Your Commissioners are of opinion that had the ..ground Manager at the Glencoe Colliery carried out his intention - shown by ordering material to be sent into the mine - promptly built off the fire, further explosions would have been prevented and that he erred in assuming that the fire had been extinguished by the fall of roof, the conditions pointing to an opposite conclusion.

The Manager of the Glencoe Colliery was engaged on other work outside the management of the mine, which was apparently acquiesced in by the Directors. The evidence disclosed laxity of control and supervision. From his own statement, it appeared that he had not visited the Main North Heading for a period of two months prior to the explosion, and, taking into consideration the conditions which existed in that heading, and the danger which is always present in gassy mines after a dyke has been cut through, your Commissione find that the Manager neglected his duties."

Figure 1 shows a plan of the northern area of the underground workings of the colliery and the direction of the force of the methane explosions which in turn resulted in a coal dust explosion. Ventilation quantities are shown on this plan.

The serious situation which existed at 10h00 on the morning of the 12th February 1908 in the face of the North Main haulage is described in the evidence quoted below - Ogilvie's fatal assumption that the fire was out is inexplicable from the evidence given. Apparently the collapse of the midstone roof hid the fire from their view and it was on this fact alone that the judgement was taken that the fire had been extinguished.

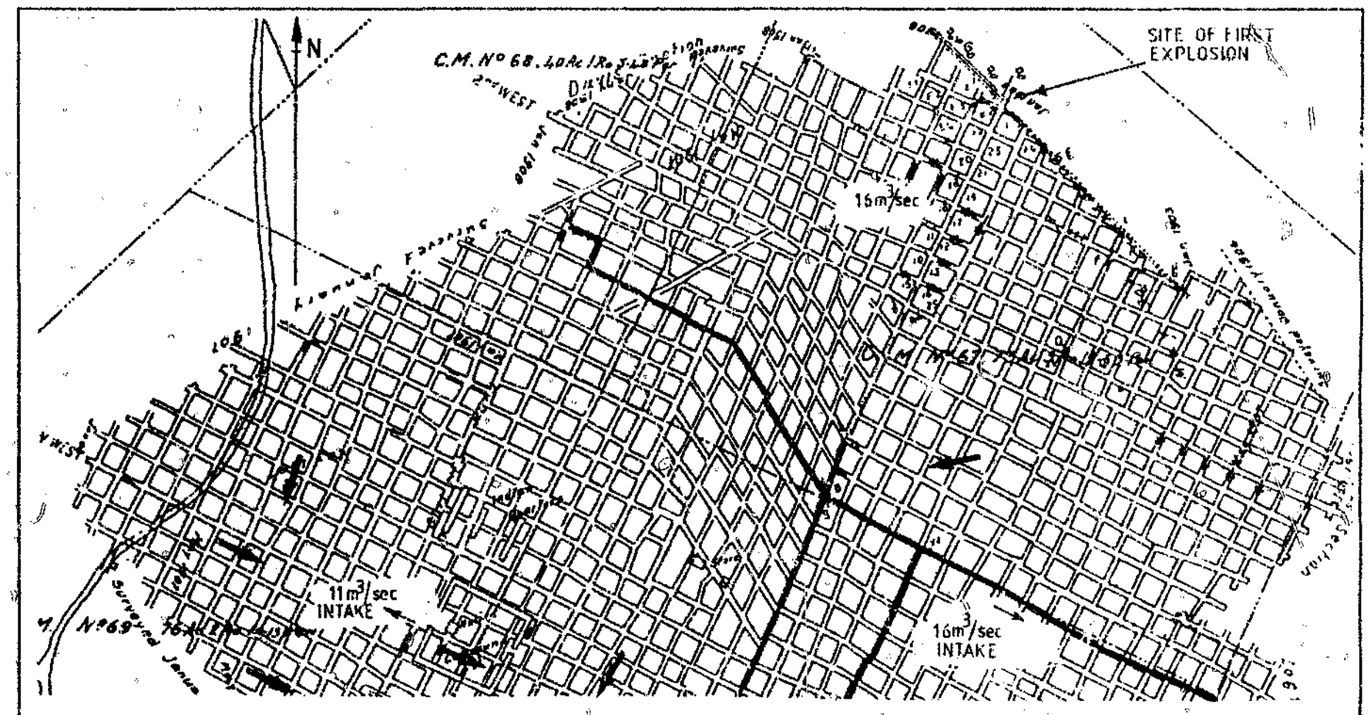


Figure 1

GLENCOE COLLIERY — PLAN OF UNDERGROUND WORKINGS — SHOWING DIRECTION OF FORCE OF EXPLOSION →
DIRECTION OF VENTILATION →

Ogilvie, Clark and Davis remained to watch the fire, it being then about 10h00. It is stated that the fire was blazing in the face, the brattice and timbers burning, and here and there small flames on the face of the coal. While they were watching, the sandstone parting between the upper and lower seams fell, about twenty three metres from where they were standing. The upper seam and its roof then commenced to fall, and continued to do so for some time. This fall hid the flames from their view. Ogilvie said that they had better return, and they retired eight stoops length - about two hundred and forty metres - down the level. Ogilvie expressed the opinion that the fall had extinguished the fire, and Clark made two journeys up the level "to see how things were going on". After his first journey he reported that "he could not see anything wrong" but he stated after his second journey he reported to Ogilvie that he did not think the fire was out. Ogilvie then proceeded to the surface where he told the Engineer not to send down any more "dagga" as the fire had "put herself out". This was about 11h00. About 09h00, or soon after, instructions had been given by Ogilvie for "dagga" to be sent into the mine, in case it might be required to build off the fire.

This poor judgement is substantiated by further evidence.

"About ten minutes after Davis had returned to the North Level, a small explosion took place, which slightly burned some Indians who were engaged in the timbering work. It was agreed that Davis should go and bring all the Blacks in his section out. This was done. Clark and Thompson retired two stoops length to Phillips' cross heading. While they were there another small explosion took place. At this time Ogilvie had gone towards the shaft. Clark, Thompson and their men then retired to the second East and West, which is also known as "Clark's Road", when another explosion making a loud report took place. They then retired a further distance of three stoops. By this time all the Blacks in the Northern parts of the mine had been withdrawn. A more violent explosion occurred, and the party retired down the Level

"A party, under Mr. Muir, consisting of Thomas Crossan, John Borland, James Reid, Alexander Campbell, J.E. Whitacre, J. Lindsay and about twenty seven Blacks, went underground soon after 07h00.

At about 08h15, an explosion of great violence took place. A dense cloud of smoke, dust, stones, and other material was emitted from the downcast shaft, described by Mr. Heslop, who saw it from the St. George's Colliery, as being several hundred metres in height, black and very dense.

As described by Mr. J.B. Pender, the Engineer of the Glencoe Colliery, there appear to have been three emissions from the shaft; the first, to about twice the height of the chimney of black smoke and dust; a second of a brown grey colour, accompanied by stones, sticks and other things. This had somewhat died away when the most violent emission occurred, the whole lasting approximately for three minutes.

One of the cages in the downcast shaft, weighing 1701 kg was at the time hanging eight metres from the bottom of the shaft. This was blown up to the pulley wheels at the top of the headgear, which was greatly damaged, the wheels being broken and the gear displaced. The cage had also cut through the beams to which the safety appliances were attached. The weight of the cage and rope together was 2082 kg.

At the time of the explosion four Blacks were carrying another Black on a stretcher down the Main North, near the first aircrossing. They were carried by the explosion to the shaft, where they found themselves, on recovering consciousness. After some search, they were able to find the signal wire which was pulled and the "bell" was heard at the surface."

During the recovery operations, which took 6 weeks, a borehole was drilled from the surface into the North Main heading, close to the face, to draw off gas which was accumulatingin the area.

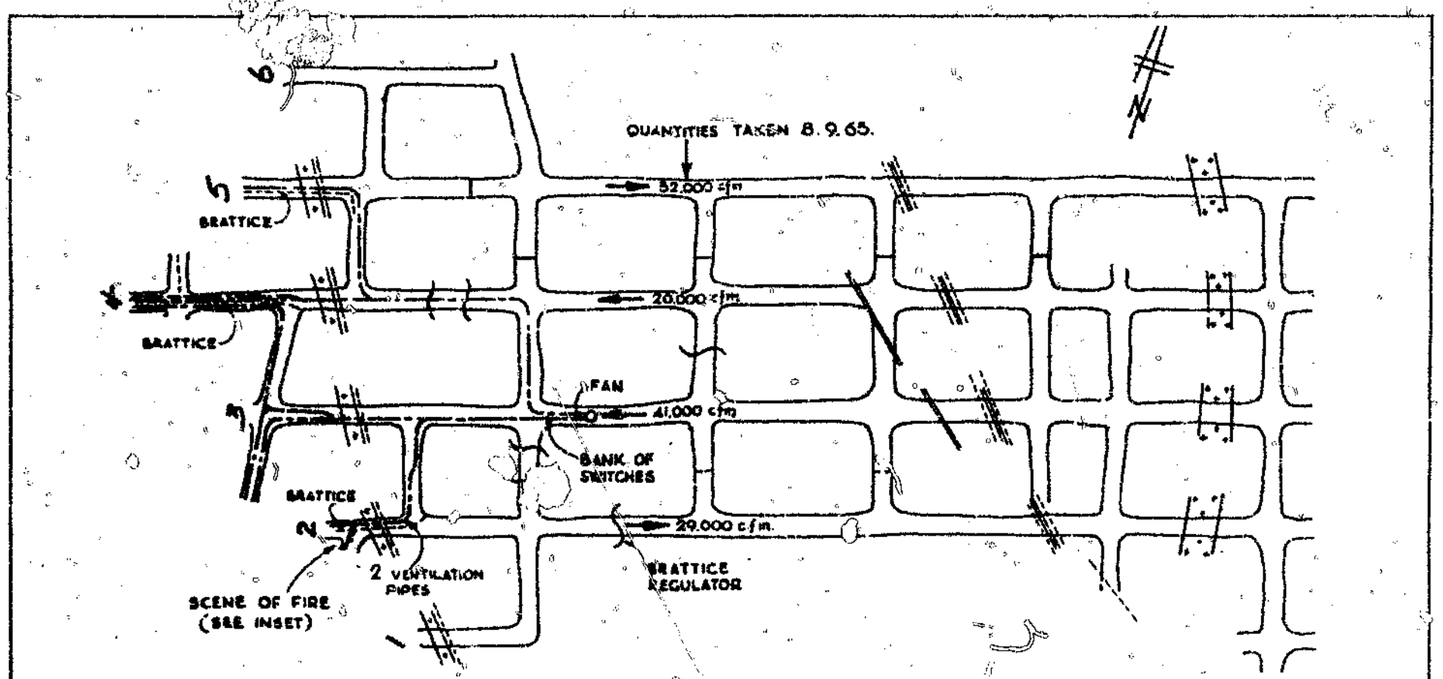


Figure 2

PLAN OF WORKINGS DURBAN NAVIGATION COLLIERY
SHOWING SCENE OF FIRE

(For sectional views see figure 2a)

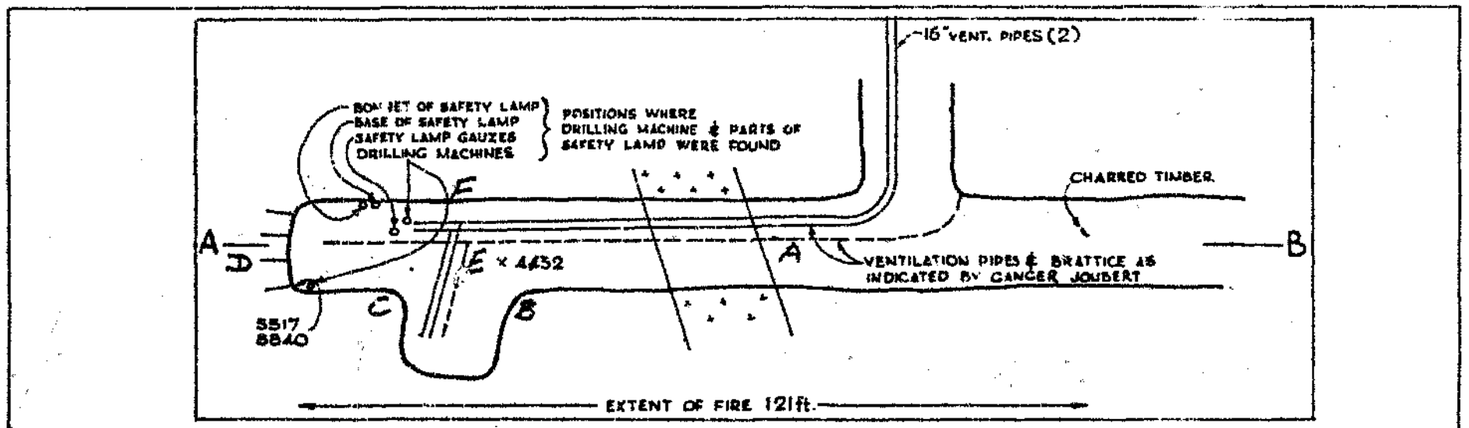


Figure 2a

SECTIONAL VIEWS OF WORKINGS AT D.N.C

see also figure 2

They attempted to extinguish the fire with water, stonedust and fire extinguishers and when they were unable to do so they notified the Mine Overseer who in turn notified the Underground Manager. By the time the Underground Manager arrived on the scene the fire had taken a firm hold and the coal was burning for some distance back from the face. Four proto teams were called out and the fire was finally extinguished at about 14h00 on the same day.

The investigation into the fire revealed that about 1.1/2 hours before the ignition both the Shiftboss and Miner had inspected the heading and had declared it to be free from gas. They also stated that the ventilation, which comprised two ventilation pipes through which $\pm$ 2,8 m^3/second was being forced to the face, was adequate. In addition the return air from No. 3 Road was coursed into No. 2 Road by means of bratticing making an alleged total of $\pm$ 4,7 m^3/second with a velocity in the region of 0,3 metres/second.

At the time of the ignition the Driller's safety lamp was suspended about 1,4 metres above the floor some 6 metres back from the face and according to the witnesses the ignition started from this vicinity. The lamp had been left unattended for some 30 minutes after the Drillers had allegedly tested for gas. After the fire had been extinguished the disintegrated remains of the safety lamp were found amongst the rubble on the floor. Also found amongst the rubble were two coal drills - the one which had been in operation at the time of the ignition and a second one which had been brought into the heading shortly before the ignition.

Charred remains of the neoprene ventilation tubing and brattice cloth were also unearthed which seems to indicate that some air had been ventilating the end.

Extensive coking of the coal was found to extend some 27 metres back from the face and charring of headboards extended a further

12 metres back. Had the fire not been brought under control so rapidly it could have entered the main return airway with disastrous results as sealing off would have been extremely hazardous. It was indeed fortunate that an adequate supply of water under high pressure was available so close to the seat of the fire.

During the inspection in Loco a strong emission of methane was found issuing from a drill hole in the middle of the face on the left hand side. Gas had been encountered in the particular road some two weeks previously shortly after the face had passed through the dyke. This was cleared without much difficulty.

The cause of the ignition could not be accurately determined and could have been due to a faulty drill cable, safety lamp becoming overheated or some other unknown cause.

The drill cable was tested after the ignition and found to be in order and as no contraband could be found it appears as if the safety lamp was the cause of the ignition. In addition it is fairly certain that the ventilation was not as good as was claimed and as the roadway was rising the gas accumulated until an explosive mixture was reached. Meanwhile the unattended lamp, in which gas was probably burning, heated up until the flash point of methane was reached with the subsequent explosion.

A disquieting feature of the construction of the Wolf safety lamp was brought to light after the ignition. A lamp was deliberately dropped causing damage to the two pillars. An examination of this lamp revealed that the rivetting of the pillars had become loosened causing a small leak between the gasket and glass. This was not noticeable from an external examination and an apparently safe lamp was rendered unsafe.

No casualties or fatalities resulted from the fire.

5.5.2 Methane Explosions in the goaf as a result of spontaneous combustion

(a) Northfield Colliery. November 1951

The area concerned is situated on the North-West side of Northfield Colliery (refer Figure 3) and its closing down meant the loss of ± 2 000 000 tons of high quality coking coal; 950 000 tons in pillars and 1 050 000 tons in situ.

The goaf fire in this area and the associated explosion caused the loss of the lives of two Blacks and one White and also severely injured two Indian bricklayers. This disaster took place in November 1951. The section involved was No. 4 situated North of the Main West Haulage and which was almost fully developed into 30 metre x 30 metre pillar centres, the panel measurements being approximately 300 metres x 300 metres and enclosed by barriers. Pillar extraction had commenced in the top right-hand corner of No. 4 panel. Three pillars (Figure 3a) had been extracted in the bottom seam only, when signs of heating manifest itself. This apparently caused no concern for work continued in the area.

From all accounts ten section workers were gassed on one occasion and the following day a further twelve were gassed. The section was then double-shifted with a view to rapid extraction; the objective being to smother and arrest the heating. This had no effect and the situation became worse. A decision was taken to seal off section 4 at the first dyke which was outbye at pillar 20. Much time was lost due to the recovery of material from the section and also the preparation of sealing sites and the section was sealed at three positions marked F.G. and H on Plan 3A. The main and right hand companion stoppings were new walls 0,5 metres thick but the left hand companion wall was part of an old stopping that had been used as a regulator. This was filled up with brickwork and plastered over to effect a seal. No production took place from the area during this time.

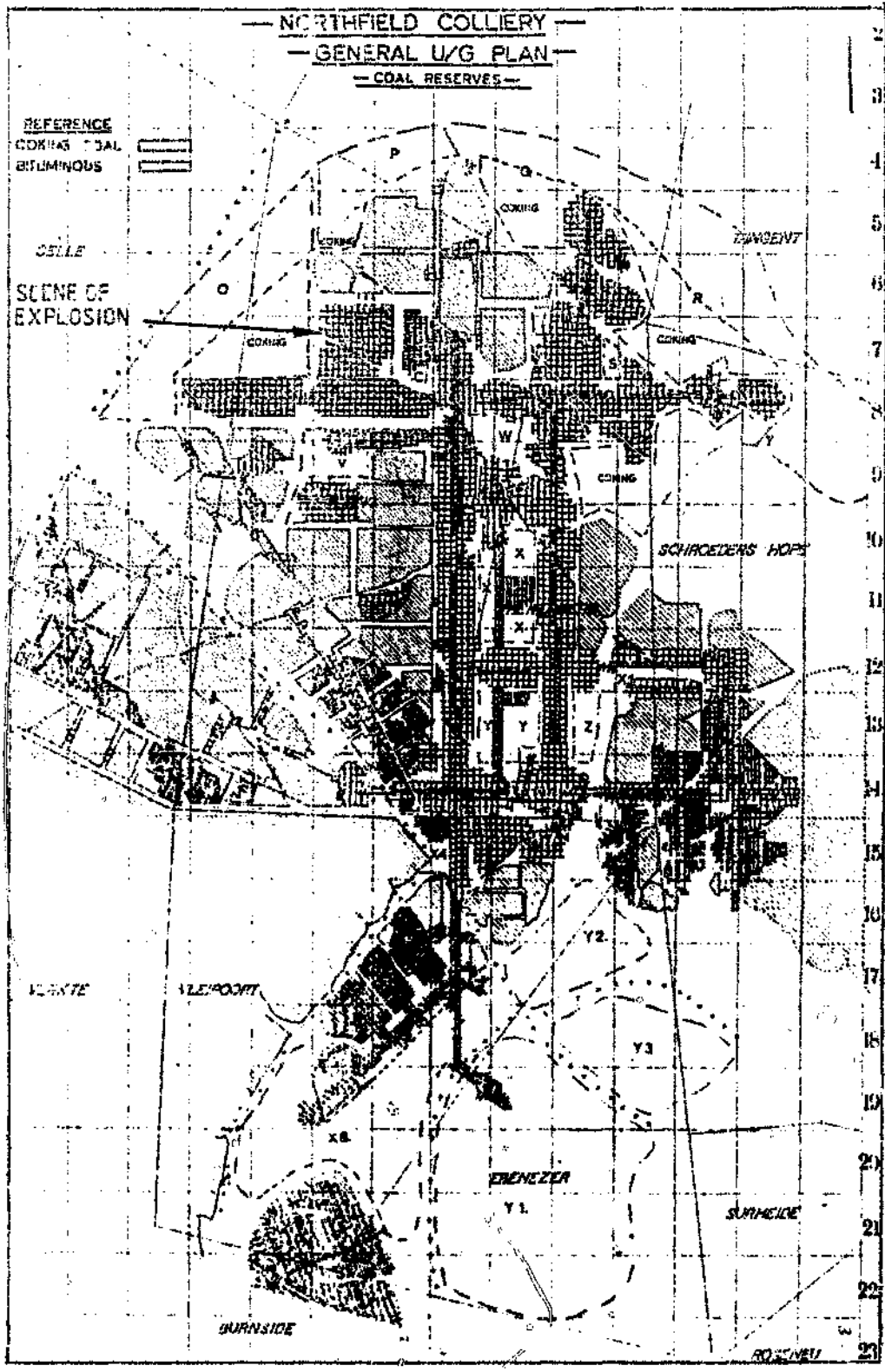


Figure 3

GENERAL UNDERGROUND PLAN NORTHFIELD COLLIERY

see also figure 3a

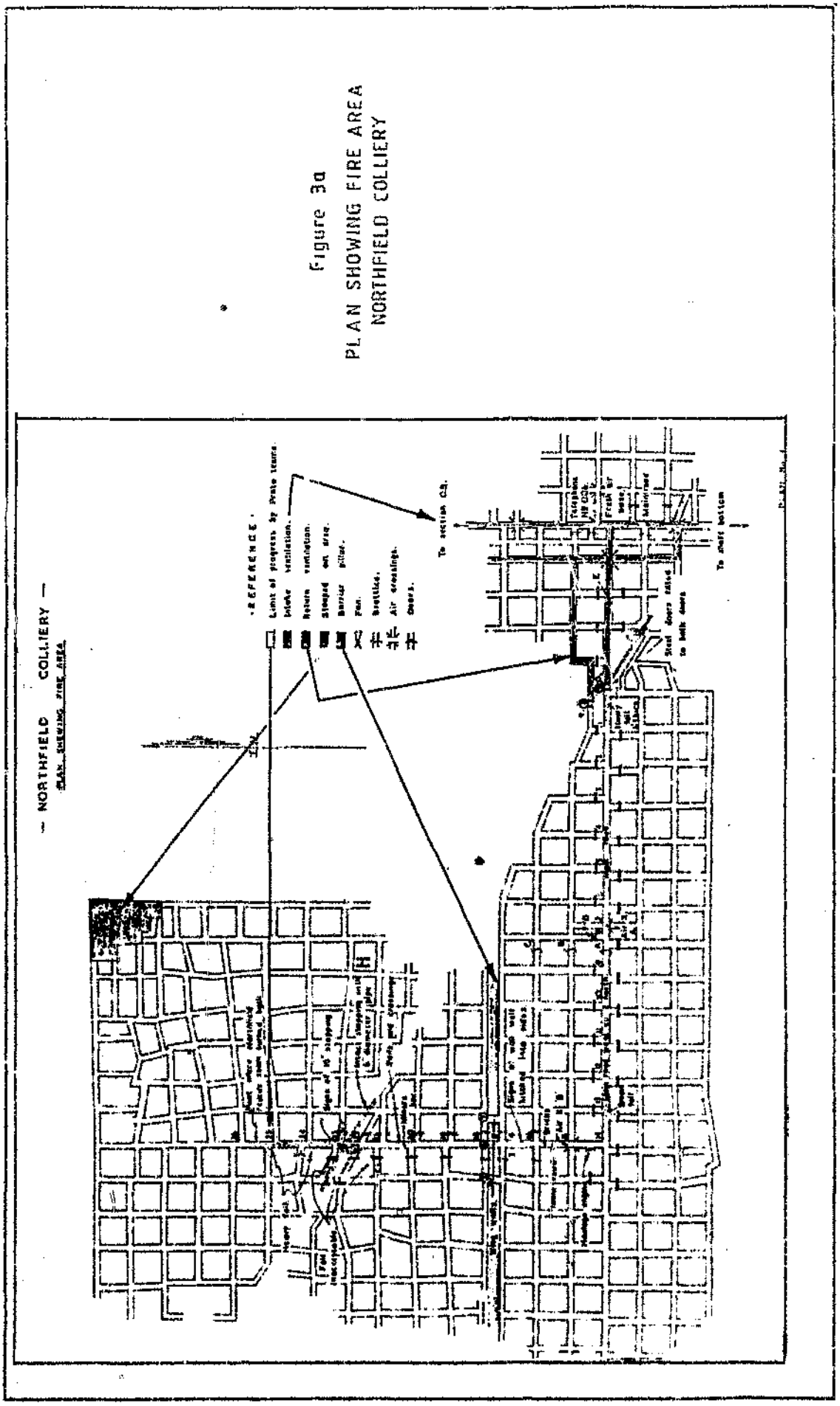


Figure 3a
PLAN SHOWING FIRE AREA
NORTHFIELD COLLIERY

The following day an inspection was carried out of the stoppings with the view to strengthening the left hand companion stopping. Whilst the Shiftboss and two helpers were making this inspection, an explosion occurred behind the stoppings, killing all three outright. The stopping on the left hand companion and main were blown out.

A second attempt was then made to seal off the area in the three roadways through the east and west barrier (marked 5, 6 and 7 on Plan 3a at Pillar 17). This again failed, as another explosion occurred, injuring the three bricklayers.

The Mines Department then intervened and made the decision to erect three explosion proof stoppings at points 8, 9 and 10 marked on Plan 3A i.e. approximately 120 metres along the west haulage; the stoppings were to be built in a large dyke. Details of the stoppings are shown on Figure 4. No further explosions occurred and the gob fire was eventually extinguished. In September 1963, the area was successfully re-opened and pillar extraction commenced.

5.5.3 Methane and coal dust explosions as a result of blasting or electrical faults on the goaf edge of pillar extraction sections

(a) Hlobane Colliery No. 2 Pit. March 1952

On the 6th March 1952 in a pillar extraction section in the coking coal seam at Hlobane No. 2 pit a timber man Titus Khumalo suffered 2nd and 3rd degree burns to his face, neck, arms, hands and back as a result of a coal dust explosion which was ignited by the illegal blasting out of 7 goaf edge timbers using 60% Gelignite non-permitted explosives tied to the timbers. Subsequent evidence of the Miner in charge of the section was that it was a common practice to blast out goaf edge timbers by drilling holes in each timber and inserting a detonator into the hole and blasting the timber out. The goaf edge was dry and dusty, some 25 mm of fine dust and coal lying on the floor. No

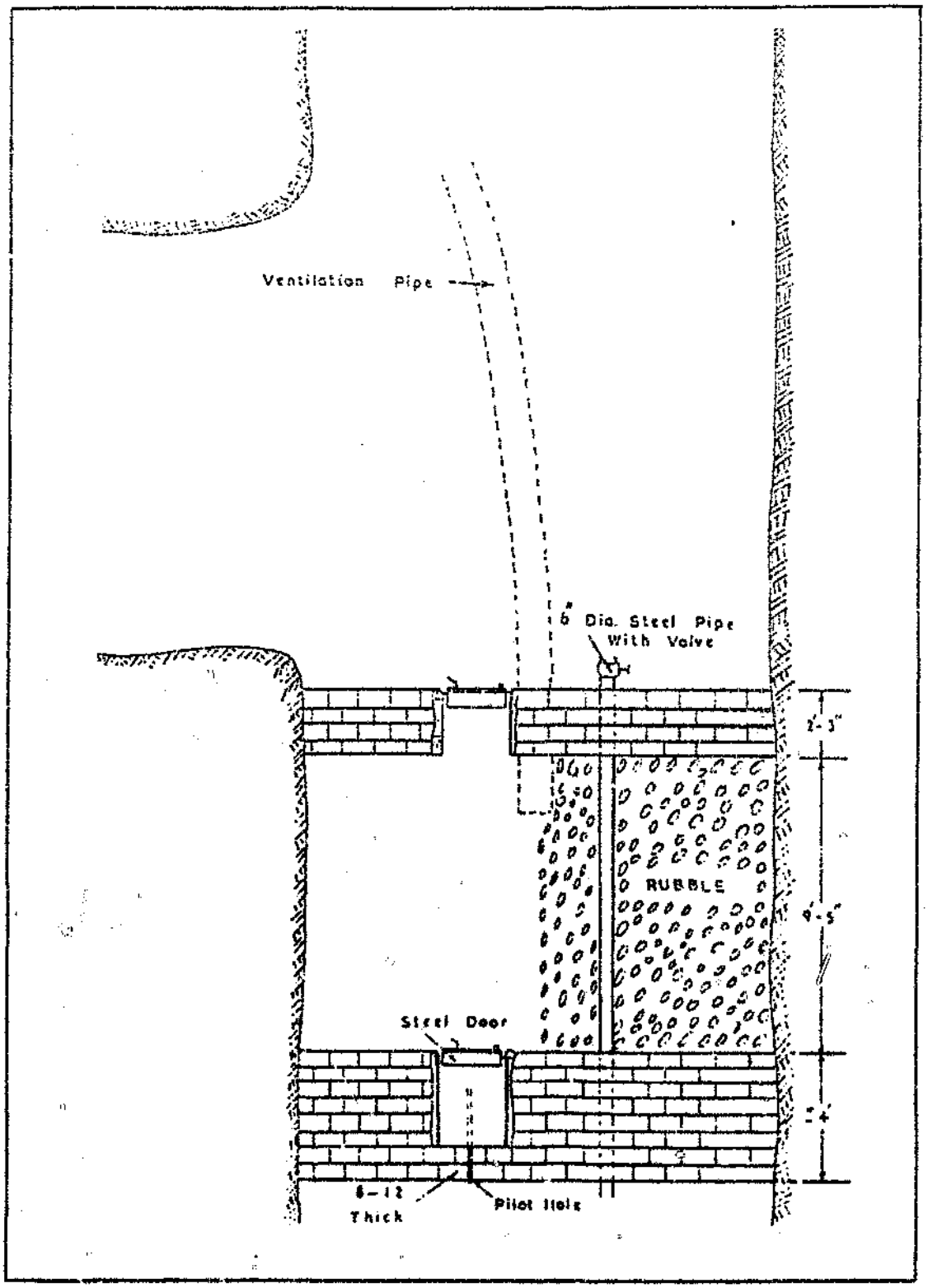


Figure 4

DETAILS OF EXPLOSION PROOF STOPPING

methane of any significance was ever detected in this seam and the mine was non-fiery. In addition Blacks who were not qualified miners, were performing blasting operations. Following a similar accident in the same section a month later, in which a man was killed and five were injured, the following letter was submitted to the Inspector of Mines (Pietermaritzburg):

"Mr. J. Ferguson, Manager, produced a statement sworn before a member of the South African Police, by Zwelingima Dyantji, to the effect that Ganger van Staden did not carry out the blasting associated with the coal dust explosion but that this was done by a Black who used coal dust to tamp the holes.

Mr. Ferguson further stated that he is not prepared to let the matter rest here.

What line of procedure do you want adopted, taking into account the information you have had from the Government Mining Engineer in this connection?"

Signed Deputy Inspector of Mines: Dundee. 21st April 1952.

The Miner Zwelingima Dyantji was injured in this coal dust explosion.

Figure 5 depicts the goaf line and the scene of the accident which was not reported to the local Inspectorate until, three weeks later, an almost identical incident occurred killing 1 face worker and causing severe burns to 5 other workers. It was only after this accident that the first incident was reported.

Some pertinent details of evidence from the Inquiry are quoted:

The Deputy Inspector of Mines, Natal, stated:

"While I was investigating an accident in which certain persons were burnt in No. 1 section of Hlobane No. 2 Colliery on the 2nd April 1952, I was told that there had been a similar occurrence in No. 1 section some three weeks previously.

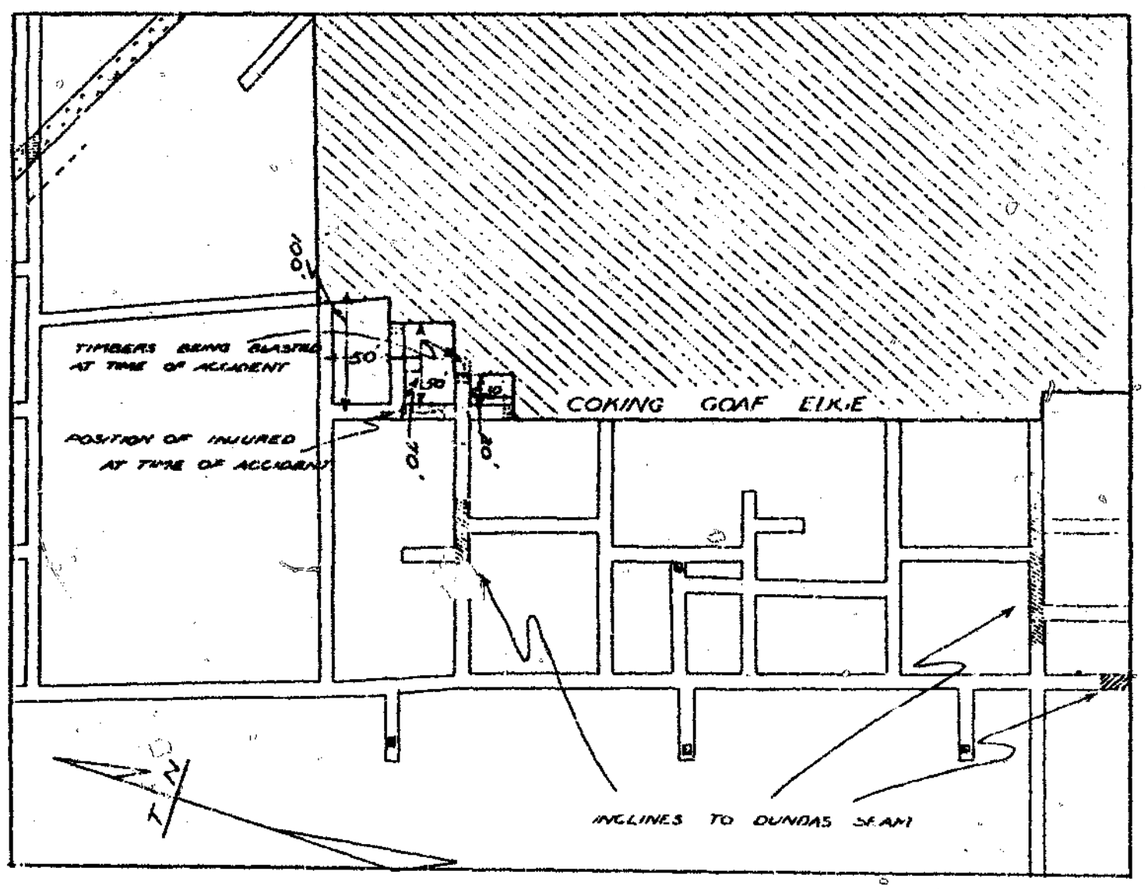


Figure 5

PLAN OF SECTION AND GOAF LINE OF ACCIDENT

On the 4th April 1952 the Sub-Inspector of Mines and I inspected section 1. I tested for firedamp at various places using a flame safety lamp and the Mines Department M.S.A. Methane Detector. I also observed whether there were any areas of the section which were dry and dusty. I found two lifts which were approximately in the vicinity of the position where a worker was injured on March 6th, 1952, according to Figure 5, the plan submitted by Mr. Nelson. These two lifts were dry and dusty. I tested for methane in five apparently freshly drilled holes in these lifts. I could detect none with the flame safety lamp but the M.S.A. detector showed methane in every hole.

The dust on the floor in the two lifts was mixed with fine coal and was about 30 mm to 50 mm in thickness. Some of this dust could quite definitely be raised into a cloud by the concussion of shotholes being fired.

There was no indication of stonedusting being or having been carried out in the coking seam area."

The injured Timberman stated:

On that day (of the accident) Luthebe and I were engaged in drawing timber from a working place.

We tied explosives around seven sticks (timber supports). The least number of shots which we tied around any one stick was six. Around some we tied 8 and others 10 shots.

The Gang Leader then arrived. He went straight and lighted the fuses of all the charges. At the time he lighted these fuses, we retreated to a set of points which were further back. After the Gang Leader had lighted the fuses, he also went back.

Several shots went off inside and whilst they were going off, I fell. I felt hot air burning me which resulted in my falling to the floor. The clothing I had on caught fire. Aaron Khumalo

came and put out the flames of my burning clothing. I was around a corner about 25 metres from where the nearest shot was fired. The place was dry. I did not see any stonedust in that area."

The acting Shiftboss in charge of No. 1 section on 6th March 1952 stated:

"I visited the place where Khumalo was later injured. Everything was in order. The place was dry and Loaders were still loading all the fine coal on the floor. This coal was from the blast. No stonedust had been distributed. This was not at that time done on this mine (stonedusting). When timber has to be drawn I report to the Manager or Mine Captain who supervise it.

There was no indication of an explosion or ignition of coal dust to be seen. I have not had any experience of these matters. I noticed no indication of burning. I did not notice any charred particles of coal adhering to timbers or any indication of coking. I thought that an electrical short circuit had occurred and was responsible for the accident as I was told the injured person was sitting on the rails. There was no electric cable in this place or was there an electric winch. I could find no short circuit. 60% Gelignite blasting cartridges were in use in No. 1 section on 6th March 1952.

Dust is removed from the floor in No. 1 section by the loaders who clean it up with their shovels as the face advances. No brooms were used at that time."

The Acting Mine Manager stated:

"On the 6th March 1952 Hlobane No. 2 Colliery was not a fiery mine; it was declared a fiery mine by the Inspector of Mines in his letter I.M.N. 52/11/18 of 5th April 1952. On 6th March 1952 I was under the impression that exemption from the requirements of Regulation 65 had been granted and this is the reason why no stonedust was applied in any of the mine workings. I had an

electrician sent down to check the electrical circuit for faults. He reported that he was quite certain that the injured worker was not burnt by electricity."

From the foregoing evidence, it is possible to form the following conclusion. The accident was directly due to the instructions given to two Timbermen and a Gang Leader by a Learner Miner to draw timber props on the goaf line by blasting the props in an area which was dry and dusty. Acting on their instructions, the Gang Leader blasted seven props using excessive charges of 60% Gelignite. The evidence does not disclose how many of the seven charges exploded before the ignition occurred, but it is quite certain that the first charges to explode raised a sufficient concentration of fine coal dust which was ignited by a succeeding shot As far as can be ascertained, the inflammation only extended over the dry and dusty portions of the goaf and adjoining roadways concerned. This area was limited and the remainder of the section workings, being in a damp or wet state, precluded propagation of the flame. The inflammation was not attended by violence to any considerable extent. No inflammable gas had been been detected in this mine before or since the accident except in drill holes and then by means of an M.S.A. methane detector and only in small percentages. The Timberman was burnt at an approximate distance of 25 metres from the point at which the charges were fired. A solid coal pillar of dimension 30 metres x 17 metres existed between himself and the charges.

Although the Acting Manager, in an attempt to defend himself for not reporting the accident, ascribed the burns to either electrical short circuit or the inflammation of atomised gelignite, all other possible causes than ignition of coal dust by the agency of explosives can be ruled out.

Owing to this accident coming to the notice of the Mines Department some 29 days after its occurrence (official report was received 33 days after occurrence), the scene could not be inspected but a sample of coal dust taken from the same seam in

the vicinity, exploded six times and inflamed twice in eight consecutive tests to which it was subjected in the Wheeler apparatus.

No stonedust had been applied in the accident area or for that matter in any other area in the mine notwithstanding the fact that the Inspector of Mines, Natal, in his letter of exemption from the requirements of Regulation 65, I.M.N. 52/2/33 dated 9th June, 1942, stipulated that exemption from the provisions of that Regulation were granted with respect to those portions of Hlobane No. 2 Colliery where the amount of moisture precluded the possibility of coal dust being raised as an explosive cloud. In a modification of the above exemption, I.M.N. 52/2/33/1 signed in Pietermaritzburg on 5th March 1952, the Inspector of Mines made the same stipulation but, as this modified exemption could not have reached the mine by March 6th (the day of the accident), the earlier exemption is considered applicable and legally binding.

The only dust sweeping in the mine prior to the accident was carried out by means of shovels. No brooms were used.

(b) Natal Cambria Colliery. July 1956

This accident, which resulted in the loss of two lives, provides a good example of the dangerous conditions which arise with high methane concentrations along the goaf line, insufficient ventilation and possible faulty electrical equipment coupled with negligence on the part of two Miners. Subsequent methane explosions imperilled the lives of senior officials and rescue brigademen intent on rescuing the two men in the section. The dangers of switching off power and then subsequently re-instating the power together with poor training and communication is highlighted. Most of the evidence of the two miners is unlikely to be true.

Evidence given at the Inquiry by the Mine Overseer (Acting Manager) is as follows:

"I am the Mine Overseer of the Natal Cambrian Collieries Limited, Cambrian Colliery.

On the 22nd July 1956 I was acting as Manager, as the Manager was away on leave. I am not the holder of a Mine Manager's certificate. I was appointed to act as Manager from the 2nd July 1956 to the 31st July 1956.

On the 22nd July 1956 (Sunday) it was reported to me that an explosion had occurred in Section 9 and that two employees had not come out of the mine. Two Miners had come out to surface. Figure 6 depicts the scene of the accident.

I proceeded to the pit head immediately arriving there at about 12h30. The Miners reported to me that they had examined the auxiliary fans in Section 9, found no gas at the fans and started them up and then proceeded with the two Blacks out of the section. When they reached the gate end switch in No. 9 haulage they noticed that it had tripped. The Blacks proceeded up No. 9 Branch Main, the Miners switched on the gate end switch and soon afterwards an explosion occurred. The two Blacks did not return from No. 9 Branch Main.

I submit herewith a plan showing No. 9 section where the accident occurred (Figure 6).

The one Miner complained of an injury to the chest and the other made no complaints. They lost no shifts as a result of injuries.

I then notified the Rescue Station and reported the matter to the Mines Department.

At about 14h15 accompanied by the Assistant Inspector of Mines and two proto teams I went underground and proceeded to No. 9 section.

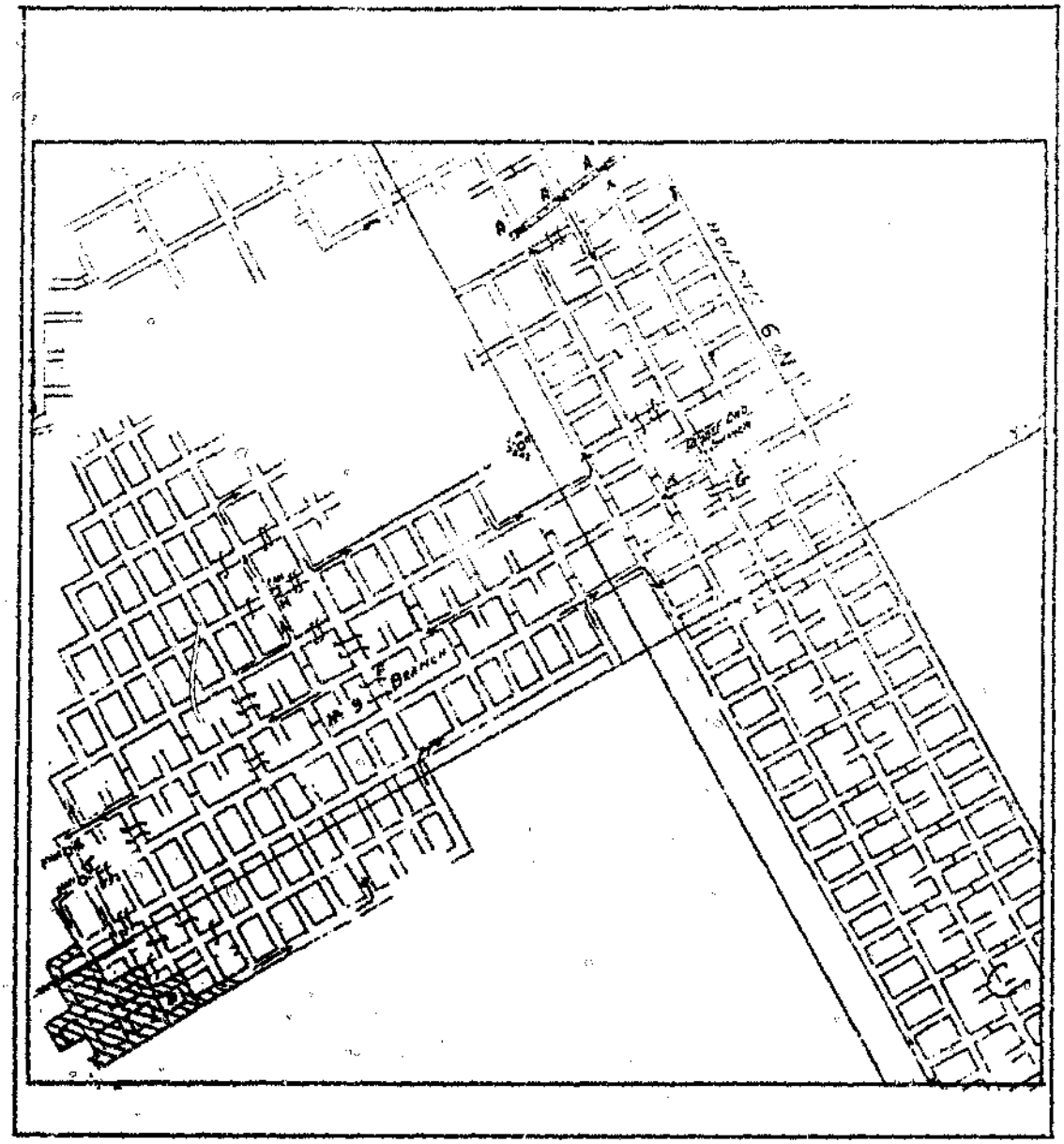


Figure 6

PLAN OF SECTION - CAMBRIAN COLLIERY

We proceeded to the intersection of No. 9 Haulage and No. 9 Branch Main in fresh air. Periodic tests were made in the return airways and in No. 9 right hand companion we found .04% carbon monoxide and smoke (400 P.P.M. of carbon monoxide).

At the intersection of No. 9 Branch and No. 9 Haulage I left the Inspector and a member of the proto team, while I, with another rescue man, proceeded up No. 9 Branch in fresh air.

We proceeded about 110 metres when we heard a noise like the uneven running of a fan. We then decided to return to fetch the rescue team when I felt a rush of wind from inbye. I presume that this was caused by an ignition of firedamp. We retreated to the intersection. The time then was about 17h00.

The possibility of entering Section 9 to rescue the Blacks was then discussed and I decided to seal the area and leave them in the section.

I consider it unlikely that the two Blacks survived the first ignition. I was told that the first ignition was violent and there were indications of this in No. 9 Haulage for a distance of about 600 metres outbye of the intersection. This could be judged by the fine layer of dust along the haulage. In addition there were heavy concentrations of carbon monoxide in the return air from this section.

I consider that a fire had been started by the first ignition and there was danger of further ignitions.

I later discussed my decision not to rescue the Blacks with the Manager of Ballengeich Colliery. He agreed with my decision. I reported to the Manager, who was in Durban, and he also agreed with my decision.

I then made arrangements to build stoppings in the dyke at the points marked A (Figure 6). The initial seals were completed at

13h45 on the 23rd July 1956. There were heavy concentrations of carbon monoxide in the right companion and dense clouds of smoke and the seal in this road was built by the proto team.

Later the stoppings were reinforced and production was recommenced on 27th July 1956.

We had no trouble with gas in No. 9 section prior to the explosion. It was suspected that trouble may be experienced there because this was a gassy area of the mine. Gas had never been reported from the section prior to the accident.

Gas had been encountered in the areas marked B and C on the plan.

Development in No. 9 was done on the bottom seam. Stooping operations had been commenced on the left hand side of the section and both seams were being extracted in the normal manner. I suspected that gas may be given off from the upper seam.

It was necessary to put auxiliary fans in the section to boost the ventilation. Originally two fans were put in the section and the third was put in because of the gas. The third fan was put in on the 21st July 1956.

Gas was reported at the point D on 19th July 1956. When I said that gas had not been detected in the section, I meant that prior to 19th July 1956 gas had not been detected. The gas was not cleared. I went to the section on 20th July 1956 and on 21st July 1956. On 20th July 1956 I found 4% fire damp on the goaf edge. On the 21st July 1956 it was about the same. I then put in a third fan marked E on the plan. It did not clear the gas but about half an hour after starting the fan the concentration was reduced to 2.1/2%. I left the place fenced off.

The accident happened on a Sunday. On that day earth continuity tests were to be carried out and the fans would therefore stop. The Miners were sent down to start the fans. One of the two Miners is the Miner in Section 9 and he knew about the gas.

The other Miner went underground on a fire patrol which is done every Sunday. The Miner in Section 9 went down because he knew of conditions in Section 9 and to learn the routine carried out during the fire patrol. The two Blacks went as their assistants.

I was told that the current was off from 06h00 to 10h00. The quantity of air during this period in Section 9 would be decreased as a result of the fans being stopped. I would expect gas in the goaf. It is possible that the concentration of methane could increase to somewhere in the explosive range during the period the fans were stopped. I would not expect gas to reach the fans. The main fan on surface was running and there would be a certain amount of ventilation in the section.

I have no idea what caused the ignition. Stooping operations had been in progress for a couple of months. There were no indications of heating in the goaf. It would not be necessary for the Miners to do any blasting. There is a possibility that the ignition was caused by the fans.

The 3,7 KW Meco EF4 fans were all electrically driven. They are all flameproof. The fans were in order. I know of nothing else which could have caused an ignition.

Flame safety lamps were issued to the Miners before they went underground. Two lamps were issued and to my knowledge they are still underground. These lamps were Nos 39 and 21. The Lampman reported to me that they were not returned to the Lamproom. The Lampman informed that these lamps were sent to the pit head for the miners. Two electric cap lamps are also missing. I do not know the numbers of these lamps.

No statement was made to me for what reason the two Blacks returned to the section.

On 22nd July 1956 I noticed that the gate end switch was in the on position.

I do not think that coal dust played any part in this ignition.

The Banksman usually challenges persons entering the mine for contraband. I do not know if he challenged persons on the 22nd July 1956.

The waiting station is approximately at the pointed marked F. Shortly before the second ignition I noticed that the gate was closed.

By the Manager:

On 21st July 1956 I examined the brattices in the section. At all of them there were slight leaks from intake to return. There was no recirculation of air taking place.

By Court:

There is a possibility that the gate end switch had tripped, but that the handle remained in the "on" position.

No further questions."

Figure 6, a plan of the section, depicts the ventilation arrangements, position of fans, gate end switches and the final seals erected after the explosions. Bleeding air over the goaf to remove methane was not practiced at this colliery. Apart from the fact that there was insufficient ventilation in the section (fans were put in to boost the section ventilation) the leakage of ventilation through the brattices in the section could have been high especially at brattice G on the main haulage.

The intake to this ventilation district was 15900 cfm (7.5 m^3/second) as shown on Figure 7 which depicts the affected section in relation to the main haulages.

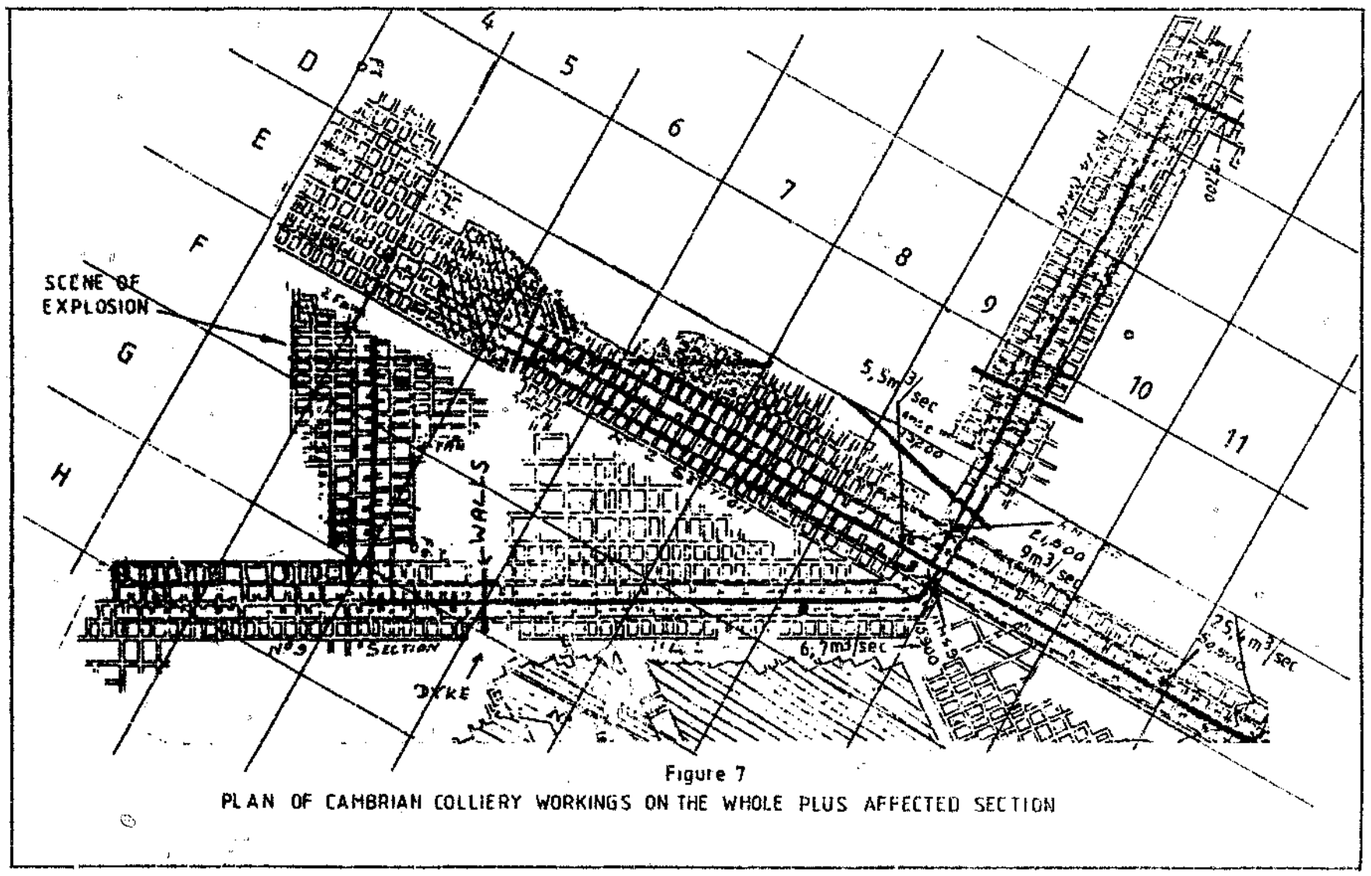


Figure 7

PLAN OF CAMBRIAN COLLIERY WORKINGS ON THE WHOLE PLUS AFFECTED SECTION

The evidence of the electrician is important as far as the explosions were concerned.

The Electrician stated:

"I am the Underground Electrician employed at Cambrian Colliery.

By Court:

On 22nd July 1956 I went underground at 06h00 for the purpose of carrying out earth continuity tests. It is necessary to cut off the power to carry out these tests. I cut off the power at 06h00 and restored it at 09h55 after completing the tests. The fans would not start after restoring the power. It would be necessary to go to each fan to switch them on. There is a switch at each fan. The gate end switch would be on.

The gate end switch controlled all the power supply to the section which includes a coalcutter, drills and fans and a winch. There are also two electrically driven pumps. These were not running on the day of the accident.

It is not likely that the gate end switch would trip as a result of a fault at the fan. In such a case the switch at the fan would trip.

I did not enter Section 9 on 22nd July 1956. I only went as far as the transformer situated at 600 metres outbye of the section.

I left this transformer at about 08h45 and returned to surface. On my way out I saw the two Miners. They were on their way in. I do not know where they were going. I saw them at about 09h15 in the Newcastle Main. We did not speak about work. I asked them what the weather was like on surface. There were two Blacks with them.

On 21st July 1956 I installed a fan in Section 9. This fan was then in order. On 17th July 1956 I tested the electrical equipment in Section 9. I found no faults. I have no knowledge of any electrical faults in the section immediately prior to the accident. A fault between the fan switches and the gate end switch could cause the gate end switch to trip."

Question:

"I had permission to isolate before I went underground. There is only overload protection on the gate end switch. It is not provided with a no volt release.

No further questions."

No attempt was made to prevent recirculation at the three fans as evidenced by the acting Manager.

"The fans H, E and J were open in the roadways. There was no brattice cloth to prevent recirculation of air taking place. Fan H had a tube about 30 metres long attached to it. Fan E had a tube about 15 metres long. Fan J had no tube. It is possible that some recirculation of the air was taking place. The general circulation of the air in the section was, however, improved by the installation of the fans.

I purposely took no steps to prevent recirculation at the fans because, in the case of a fan tripping, the flow of air would be restricted to that which would pass through the fan itself. The disadvantages of recirculation were overcome by the improved flow of air through the section."

The installation of auxiliary fans in a section will not increase the quantity of intake air - the only advantage to be gained is that air entering the section can be delivered to remote faces through tubing more effectively than face line brattice.

Evidence gathered verbally many years later revealed that a sub-Inspector of Mines had declared the fan at site to be flameproof provided that it was used in intake air where no recirculation would take place. The fan was apparently not flameproof. The section was never re-opened and the two bodies remain entombed to this day.

(c) Balgray Colliery. 1979. Two killed and 9 injured

Balgray Collieries (Pty) Limited, a wholly owned subsidiary of Natal Anthracite Colliery Limited, operated a colliery in the Utrecht district of Natal. The colliery, an adit mine, produced a low volatile anthracite from the Gus seam.

At the time that the explosion occurred, the colliery deployed two conventional mechanised sections and two single shift handgot sections to produce an average of 40 000 run of mine tons per month from a seam with an average thickness of 1,5 metres.

Mining was by the bord and pillar system with seven headings being developed at 23 metre centres for a panel length of approximately 750 metres. Bord widths were 5,2 metres. The pillars were extracted after completing the development of a panel.

The colliery was classified "fiery" but prior to the explosion had not had a history of methane emissions. From time to time small quantities of methane had been detected during the course of normal mining operations.

By 23rd April 1979, the 102 panel had been developed 730 metres inbye of the 101 panel conveyor drive and pillar extraction had commenced. Four rows of pillars had been extracted.

On the 23rd April 1979 pillar extraction was in progress in Section 102 as depicted in Figure 8. Details of the ventilation

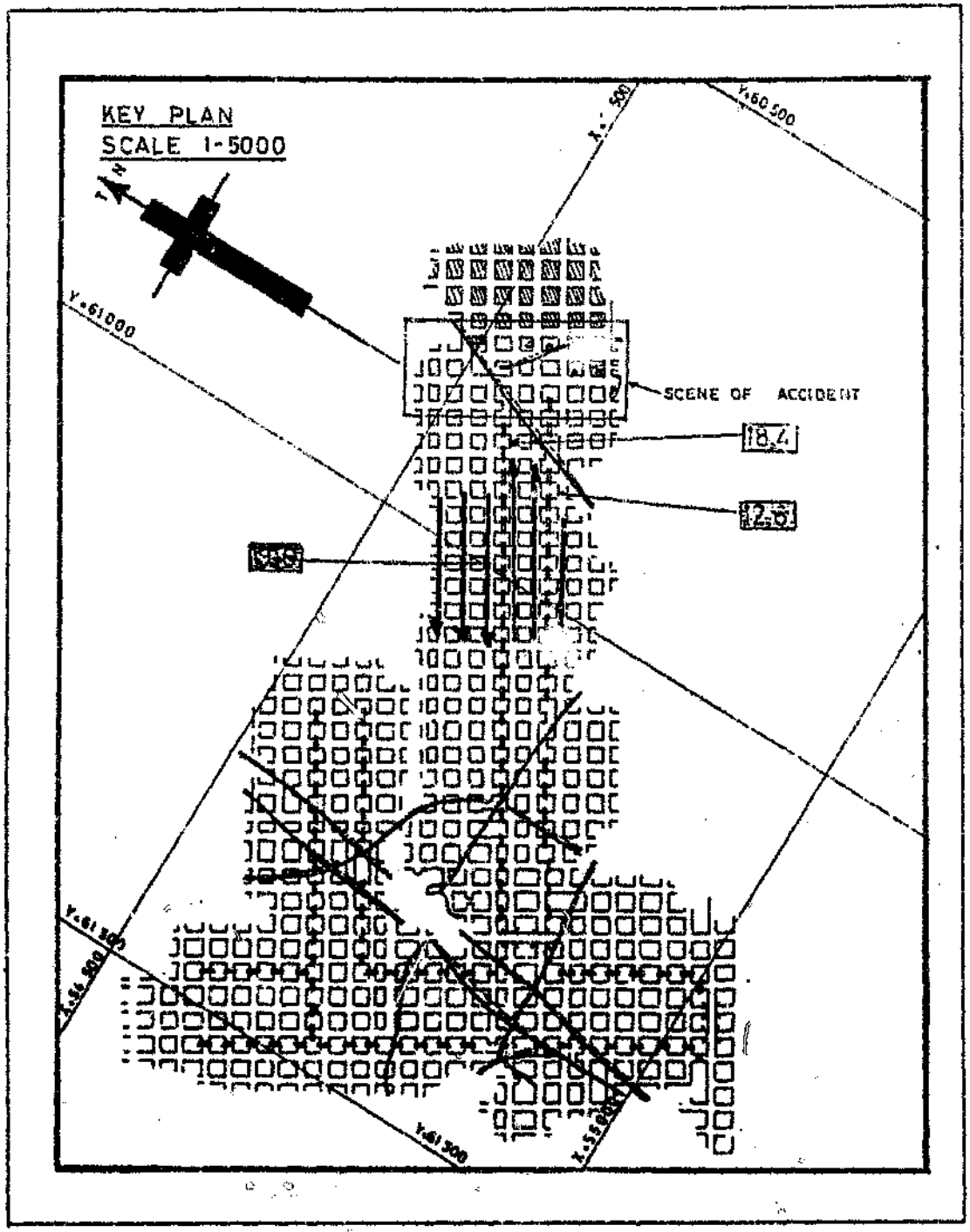


Figure 8

SECTION 102 BALGRAY COLLIERY

into this area is shown. The practice of bleeding ventilation over the goaf was not practiced presumably since the colliery was not regarded as a "gassy pit".

Pillar extraction was by the well-proven method of pocket and fender which had been successfully used at Springfield Colliery in the Transvaal since 1970. Coal was won from the section using conventional equipment consisting of a mechanical loader, two shuttlecars and one coalcutter.

Figure 9 shows a detailed plan of Section 102 at the time of the explosion. The evidence of the Inspector of Mines who arrived in the section an hour after the ignition is quoted since this provides a lucid picture of the area just after the explosion had occurred.

"I am the senior Inspector of Mines stationed at Dundee.

On 23rd April 1979 I was notified of an explosion at Balgray Colliery, at about 12h00 whilst I was holding an enquiry at Umgala Colliery.

I arrived at Balgray at about 13h00 and proceeded underground with the Surveyor to 102 section. There I met up with the Manager and acting Mine Overseer. These persons were busy rearranging the ventilation brattices which had been destroyed in the blast. There were no eye witnesses to the accident at the time of my underground inspection. All the crew, including the Miner and the Shiftboss had gone out of the mine.

I have seen the plan Figure 9 tendered by the Manager and agree that it represents the conditions found in the section except that on the south side last road the methane reading was 1,2% and not 0,8% as shown on the plan. Similarly on the North side the methane reading was 1,4% and not 1,2% as shown. I took these readings myself, against the roof.

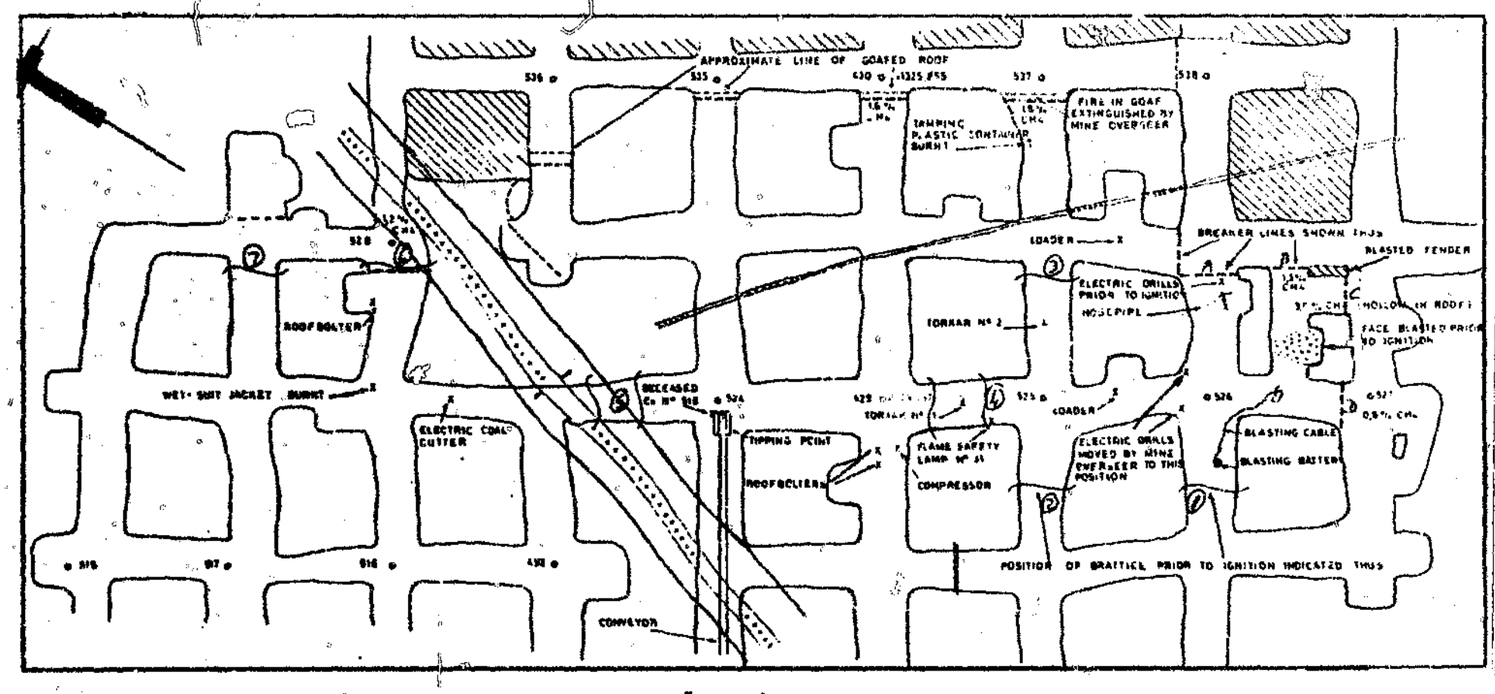


Figure 9

DETAILED PLAN OF BALGRAY SECTION
AT THE TIME OF THE EXPLOSION

I examined the brattices in the section. At No. 1 position I saw a rail against the roof, from which a brattice could be hung, but there was no brattice and I was told that this brattice had been blown away by the blast.

At No. 2 position I cannot remember what was there.

At No. 3 brattice there were signs that the brattice had been scorched.

At No. 4 brattice position section personnel were busy erecting brattices and the Mine Overseer informed me that these brattices had been rolled up (presumably prior to the explosion).

At No. 5 brattice position new brattices had already been erected but there were pieces of scorched brattice lying in the direction of the tipping point on the haulage.

At No. 6 brattice position the brattice had been badly burnt and blown out. A new brattice had not yet been installed.

At No. 7 brattice position the brattice was also badly burnt and blown out.

It was pointed out to me by the Manager and Mine Overseer where the Miner was at the time of the explosion i.e. near the position marked blasting battery (exploder). The exploder had been removed but the blasting cables were still in position. We then went into the places which had been blasted just prior to the ignition. It was pointed out to me that a fender and a split had been blasted simultaneously. There were no signs of a violent explosion in that place because an empty explosives carton was lying there and it had not been damaged or burnt.

In the next road marked "electric drills prior to ignition" the split had been cut and some holes drilled in the face. Some large pieces of coal had fallen from the face, obviously after

the blast. The Mine Overseer informed me that he had removed the electric drills as well as flame safety lamp No. 31 from this place. He had moved the electric drills back along the road to the position indicated and the safety lamp to the vicinity of No. 4 brattice.

I examined the flame safety lamp No. 31 and found that the striker mechanism was not working, there was no spark at all and it seemed as if the flint was missing. The pillars were badly bent.

In the vicinity of peg 537 it was pointed out to me by the Mine Overseer that he had found a fire burning there. He added that the fire had been extinguished using fire extinguishers and water hoses. I felt along the area indicated and the area was cool. There was no sign of fire nor any smell of fire. A Draeger carbon monoxide test was made and gave only a slight trace. A plastic bag which had contained water tamping ampoules was found in the vicinity. The bag was burnt except for the ampoules which contained water. It was clear to me that there had been a lot of heat there.

We proceeded along the section in a northerly direction. At the place near No. 6 brattice there were signs that the force of the explosion came from the goaf and travelled in a westerly direction. A burnt wet suit jacket was found in this vicinity. This jacket was badly burnt. The condition of No. 7 brattice has already been described. We then walked back towards the tipping point. From the direction in which the brattice had been thrown it was clear that the force of the explosion travelled from north to south. It was pointed out to me where the now deceased had been thrown against the tail end conveyor. The structure had blood stains and bits of tissue on it.

The intake air was measured and it was found that 18 m^3/s was entering the section. I also asked for an Electrician and he examined all the cables on the machines. All the cables were

found to be in order except for one cable on a roof bolter that was not in use at the time of the accident.

I asked the Mine Overseer for the fireman's report for the day shift of the 23rd April 1979 and he said that he could not find it.

I also went through the records in the lamproom and found that the Miner had signed for four safety lamps on that day. Three of these lamps were back in the lamproom and appeared to be in good order. The fourth lamp was No. 31 which was found underground.

By Court:

"At No. 4 brattice position personnel were re-erecting the brattices. It is possible that they (Manager and Mine Overseer) added one or two more brattices. These brattices were not scorched and I accepted that these were the brattices which had been rolled up at the time of the ignition as stated by the Mine Overseer.

The striker mechanism of No. 31 safety lamp turned but no spark was obtained. It is possible that the flint had been used up."

Once ventilation had been re-established in the section and methane had been cleared from the goaf, a decision was taken by mine management to establish bleeder roads to ventilate the goaf before proceeding with any further pillar extraction. This plan of action is shown on Figure 10. The logic behind this decision was that the goaf contained large quantities of methane and that this danger should be removed by ventilating the goaf.

Normal work was resumed in the panel on the 24th April 1979.

Table 5.1 shows the analysis of gases on the 24th April after work had resumed:

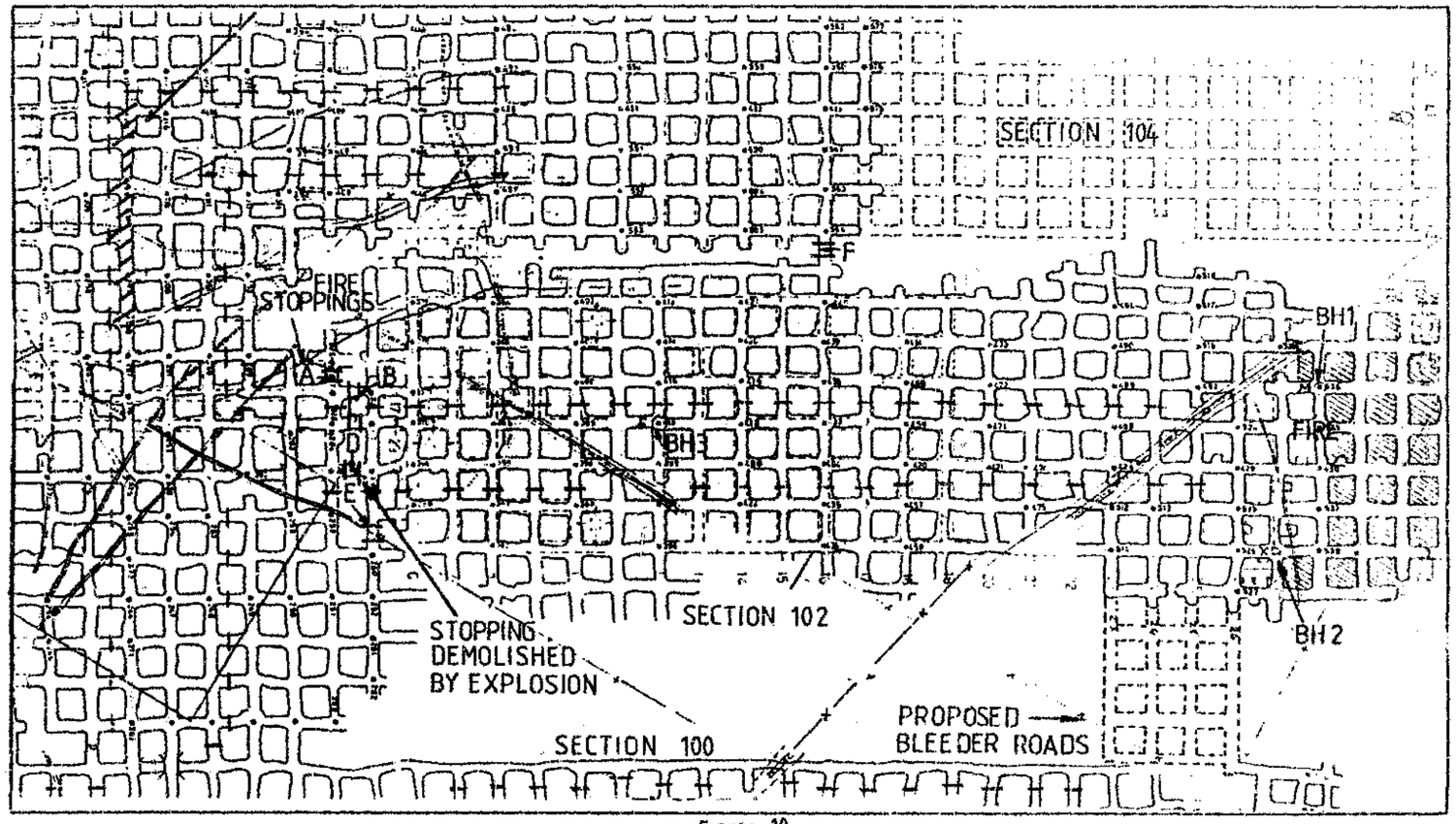


Figure 10

BALGRAY SHOWING PLAN TO INSTAL BLEEDER ROADS

Table 5.1

Initial results were:

	CO	CH4	
		Roof layer	General body of air
Left hand side returns (air routed through goaf)	100 ppm	1,8%	1,0%
Right hand side returns	100 ppm	1,4%	0,6%

After 8 hours results were:

Left hand side returns	10 ppm	1,0%	Nil
Right hand side returns	Nil	Nil	Nil

However, at 18h40 and 19h00 on this day two further explosions took place in the goaf - some 31 hours after the first explosion. No blasting operations were in progress and it is concluded that:

i) An undetected fire in the goaf as a result of the first explosion had caused these two explosions, or

ii) Frictional heating during heavy goafing had been the cause.

Two interesting points arise at this stage. Firstly, coal lying in the splits had caught fire after the first explosion and these fires were extinguished with fire extinguishers in the section. Secondly the Mine Overseer reported in evidence that on the 29th March 1979 and 30th March 1979 heavy concentrations of methane (in excess of 5%) were detected in the right hand side goaf of section 102 - the scene of the first explosion. The Mine Overseer cleared this gas himself that night. He reports that:

"I found +5% on the roof just behind the breaker lines; the layer was about 15 cm thick along the roof.

I had to install a temporary support and brattice to clear the gas. The gas took about 2.1/2 hours to clear. The section continued working afterwards.

The following day (30th March) gas was again reported in the same area. The Ventilation Officer and myself found the gas. I repaired the brattices which I had installed the previous night and the gas cleared. I again found 5% methane against the roof in the form of a layer.

This gas accumulation was discussed with the Manager and it was decided to hole through one of the roads which had previously been stopped. This was done and no further accumulations of gas were found. The Ventilation Officer and I discussed the possibility of a bleeder road on the right hand side (of the section) but as the section narrows down to one return through the dyke we did not think that it was a feasible idea. I cannot remember discussing bleeder roads with the Manager. I do not know if the Ventilation Officer did or not."

Following these three explosions a decision was taken to seal the section with 6 stoppings as shown on Figure 11. The installation was completed at 23h55 on the 26th April 1979.

At 13h10 on the 27th April 1979 a further explosion (the fourth) occurred in panel 102. Several incidents of technical interest now occurred:

a. Initial sealing of the panel and gas analysis

Figure 11 shows the 6 numbered stoppings which were installed.

It was also decided to seal the panel as quickly as possible and thus only the inner walls were to be built initially and the outer walls and sandbagging were to be subsequently completed. On completion of the inner walls, all personnel were to be withdrawn from the mine for 24 hours. An essential requirem...

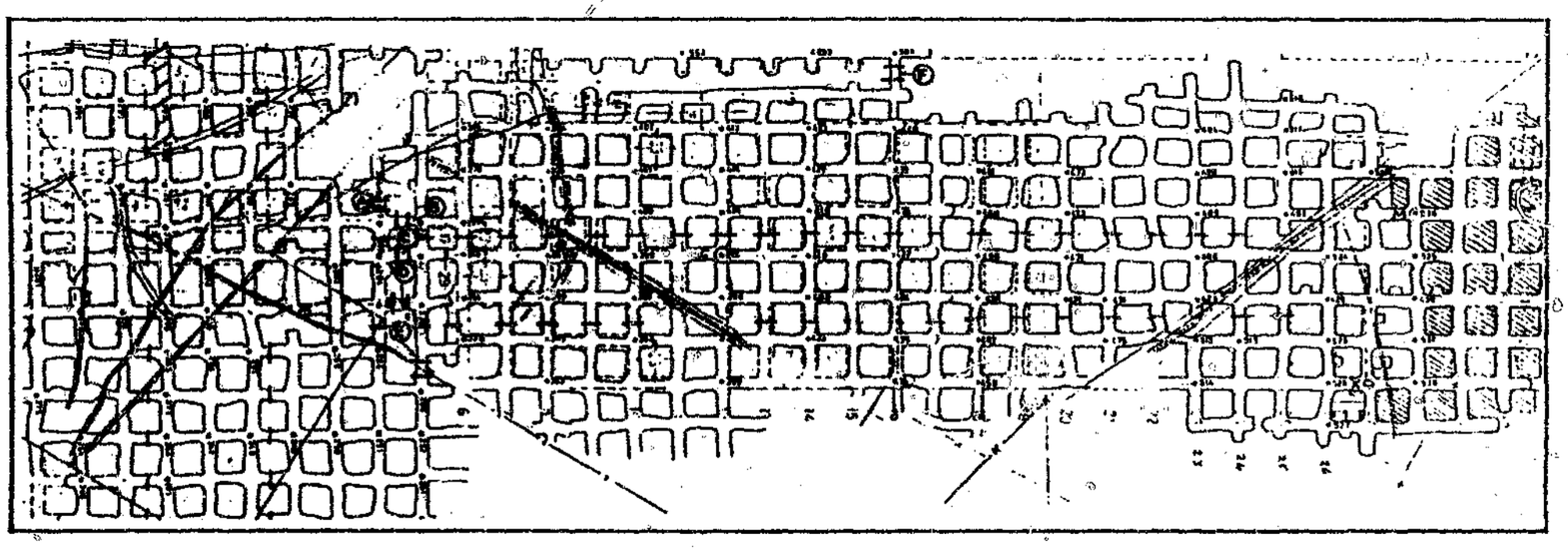


Figure 11
PLAN TO SEAL OFF SECTION AT BALGRAY
WITH 6 STOPPINGS
AFTER NEXT 2 EXPLOSIONS

in the sealing of the panel was that any gas build up had to be delayed until the last possible moment, before effecting the final seal.

In this particular instance it was necessary to dilute the methane being generated in the goaf area by maintaining a flow of air through the goaf while the sealing operation was in progress.

The construction of the stoppings was planned to be completed in the following sequence - A B F D E and C. 800 mm x 800 mm x 25 mm thick steel doors were to be built into stoppings C and E. To maintain the flow of air into the panel, the doors were to be kept open during the sealing operation and were to be closed simultaneously when all the walls were completed.

b. Gas analysis equipment

The decision having been made to seal the panel, the Chief Inspector of Mines, Dundee, was advised at 10h40 on Wednesday 25th April that the Chamber of Mines mobile gas analysis equipment was required at Balgray.

At 17h40 on the 26th April the gas analysis unit from Witbank arrived at Balgray and was commissioned and made ready for use.

The gases contained in the air returning on the left hand side of the section during construction of the inner walls did not exceed:

CO	-	14 ppm
CH4	-	1,27%
O_2 minimum	-	20,6%
CO_2	-	0,00%

The importance of having the gas analyses unit available at the colliery cannot be over emphasised:

- Results of analyses of air samples are available within 20 minutes of delivery to the unit.

- Early indications of a build up of dangerous conditions allows action to be taken to minimise the danger to work parties associated with the emergency.

c. Drilling of borehole from surface into the goaf of Panel 102

The planning committee decided that the Chamber of Mines 200 mm diameter rescue drill be obtained to drill a hole into the goaf area. The primary aim was that air samples would be taken thus firstly providing management with details of the gas mixtures in the goaf and secondly providing an early warning system of dangerous gas build-ups in the goaf which could lead to further explosions. It had been assumed (correctly as was later established) that one or more of the seals had been destroyed during the fourth explosion and the immediate objective was to replace it thus allowing production to be resumed. The author was present at the colliery during these discussions. What was not realised at the time was that the hole could also be used to evacuate dangerous quantities of methane from the goaf. When the hole did finally penetrate the goaf area the hot gases blew out of the hole at a high velocity and it was not necessary to use a standbye fan to exhaust these gases.

Figure 12 is a graph of the gas samples taken at the borehole after the damaged seals had been replaced on the 30th April 1978. This was a crucial day since the decision had been taken to replace the blown out stopping. The percentage methane in the goaf was approximately 60% and the oxygen percentage was at 4,5 - a mixture clearly outside the explosive triangle (after Coward). The author was present and in charge of the building operations. Of concern to the team at the building site ("D" on Figure 11) was the gas analysis at 13h00 (an hour before the final sealing) which is shown below:

Table 5.2 Analysis of air sample at building site

Oxygen	15,9%
CO	0,22%
CO_2	0,48%
CH_4	7,5% (this had increased from 5,5% at 08h00)
H_2	0,0%

The methane/oxygen mixture is within the explosive triangle (after Coward) but it was judged that, since the methane/oxygen mixture in the goaf, where the suspected fire existed, was well outside the triangle that work should proceed on the final seal.

Figure 13 depicts the type of seal that was finally erected on the 6 roadways shown on Figure 11. This seal was considered explosion proof.

After the last brick had been placed in position, the crew hastily withdrew from the site, travelling in Kersey scoops or trailers to surface, closely followed by the proto team who were on standbye at the fresh air base. The judgement proved correct - no further explosions occurred. A new fire ratio has subsequently been developed by Morris which, inter alia, may be used to determine when an explosion behind seals is imminent. The ratio developed is $\frac{N_2}{(CO + CO_2)}$ versus time and Figure 14 is an example of the use of this formula in a sealed off area.

From Figure 14 it may be stated that at the time indicated between points 1 and 3, the fire is active, whilst from 3-4 the area had been sealed and the fire area was becoming extinct. The subsequent rise between points 4,5 and 6 corresponds to a resurgence in the activity of fire due to the holing through of the coal pillar. After sealing, the result becomes apparent if one observes the smooth flow from 6 to 7. Points 6 to 12 inclusive, recorded over a six hour period, indicates a definite cessation of activity within the fire area, which may at this stage be termed extinct.

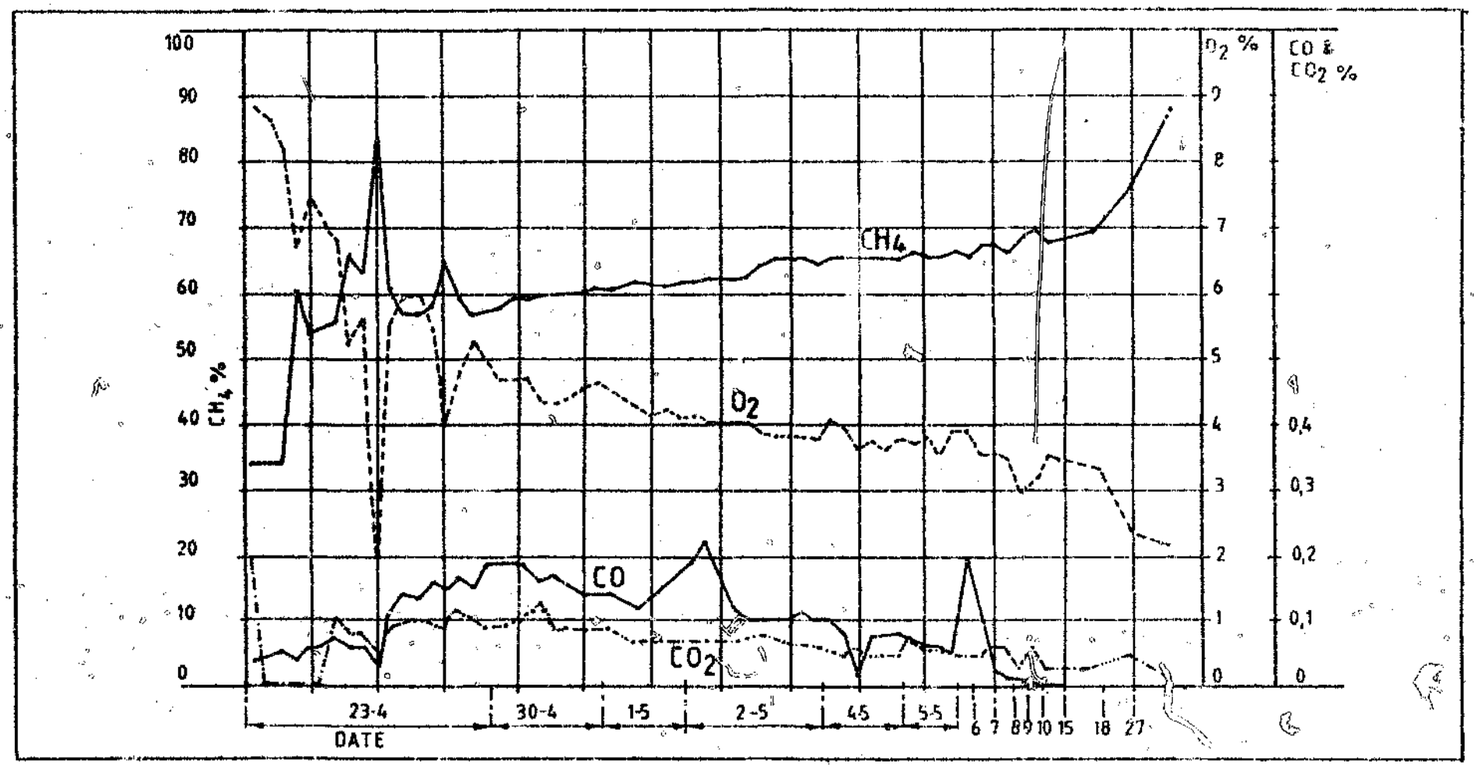


Figure 12

RESULTS OF GAS SAMPLES TAKEN AT SURFACE BOREHOLE
BALGRAY COLLIERY

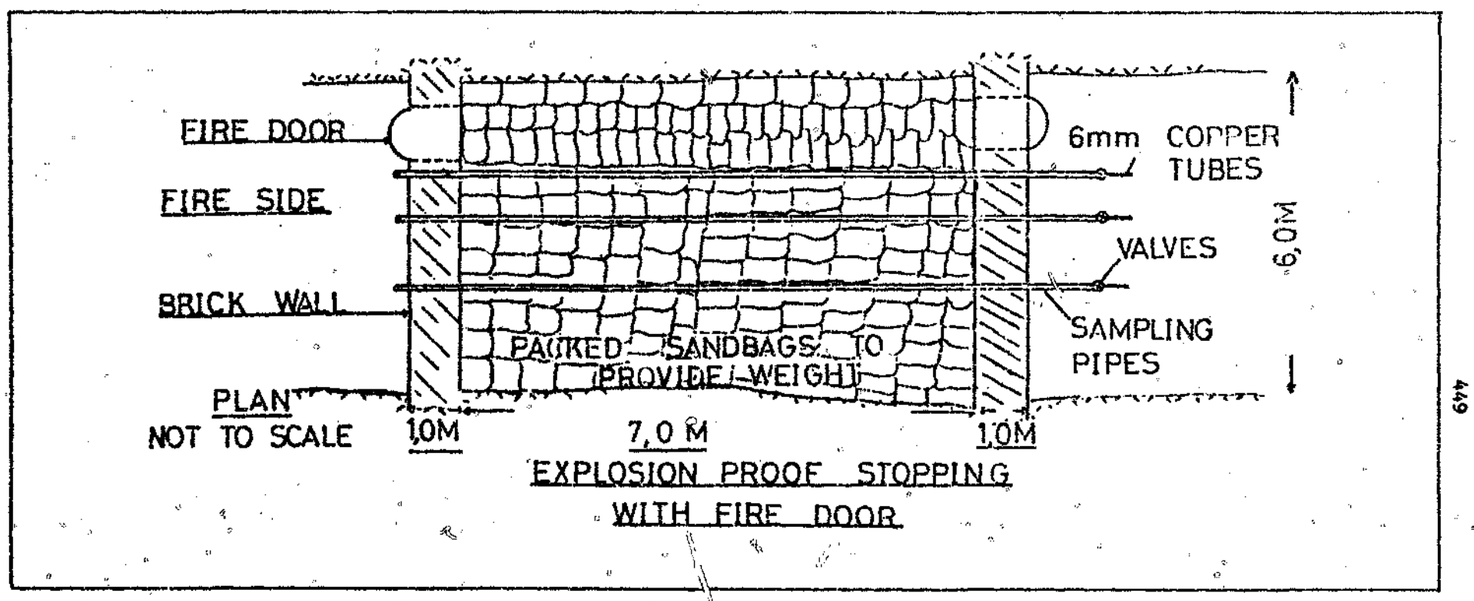


Figure 13
TYPE OF SEAL USED
BALGRAY COLLIERY

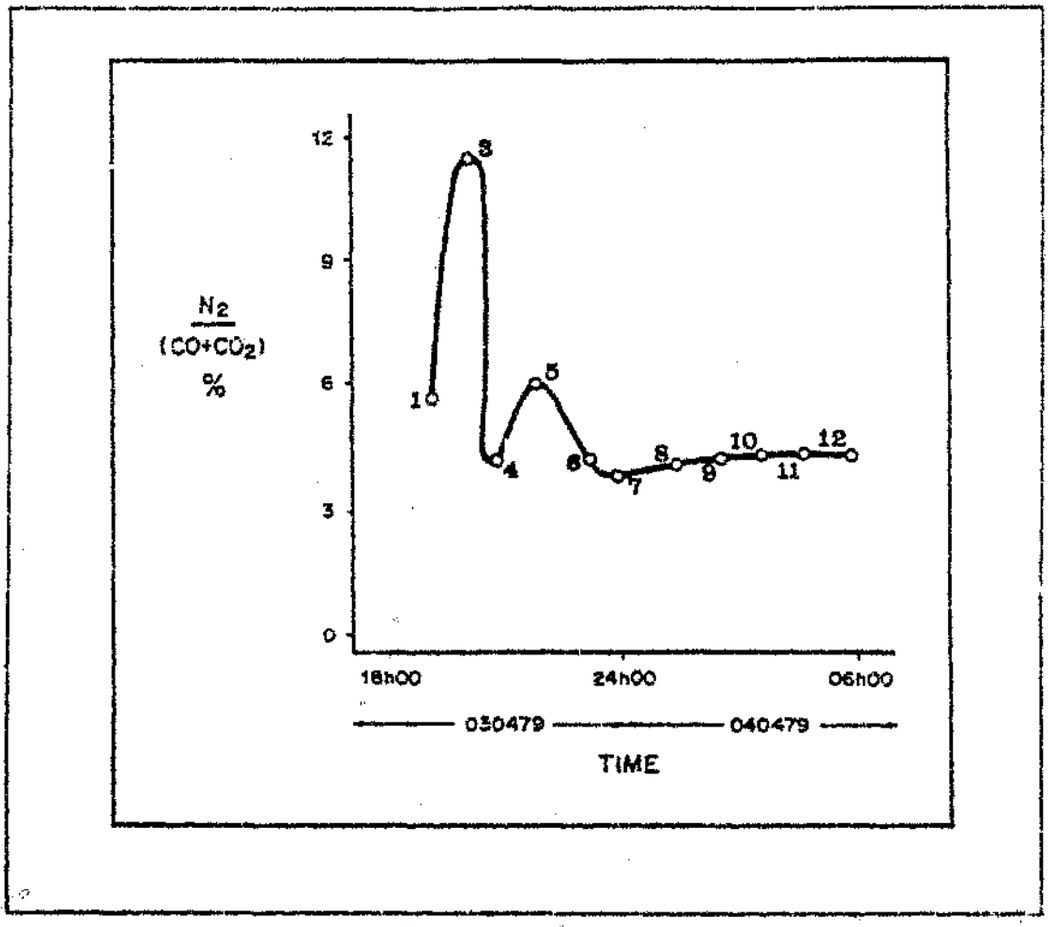


Figure 14

EXAMPLE OF THE USE OF THE FORMULA IN A SEALED OFF AREA

MORRIS FIRE RATIO

5.6 SUMMARY OF THE PRACTICES REVIEWED ABOVE

The incidents described in item 5.5 deal mainly with explosions in secondary mining (stooping). Unsafe practices which arise in these Inquiries which are of importance to Mining Engineers and serve as a point of departure for the formulations of "precautions to be adopted" are summarised below:

- Delays in sealing off fires or dealing promptly with this aspect in some other manner. "Better is a good retreat than a bad stand". It is sometimes beneficial to seal off a fire area at some distance from the source of the fire and not directly adjacent to it (see Figure 7). When a fire has been detected, for instance, in the goaf, then attempting to smother it by increasing the pillar extraction rate is not a safe alternative.

- Assuming that fires have been extinguished when they, in fact, continue to provide an ignition source. Thirteen valuable hours were lost at the Glencoe Colliery in which sealing-off operations could have commenced as a result of incorrect judgements that the fire had been extinguished. At Balgray Colliery it was the opinion of Officials that there was no fire in the goaf after the first explosion - a fatal opinion.

- Senior officials engaged on other work outside the management of the mine. In addition Management often make decisions and statements on matters without taking cognisance of technical aspects. After the coal dust explosion at Hlobane No. 2 pit in 1952 the acting Manager was of the opinion that the injured suffered burns as a result of "atomised gelignite explosives".

- Laxity of control and supervision. The author questions how Mining Officials from the Manager down to the Shiftboss could continue to allow a Miner and his workers to overload shotholes, use coal dust as tamping and remove goaf line timber by blasting these supports using non-permitted 60% gelignite. In evidence, under oath, the same junior Officials indicated that no goaf line timber was withdrawn until the Mine Overseer/Section Manager had been notified.

- Officials failing to direct their full attention to areas where problems (gas, poor roof and faulted ground and dykes) have arisen. This is described as a lack of a sense of proportion - getting the priorities wrong.

- Diluted supervision; insufficient officials appointed to supervise difficult conditions.

- Miners in charge of sections which are too large in complement of people or in area to be properly controlled and managed.

- Working sections with no qualified Miner in charge.

- Use of inflammable brattice cloth (still in use today).

- Erecting temporary brattice cloth in splits instead of erecting permanent brick stoppings.

- Lack of stonedusting in sections.

- Lack of building material at a particular site to promptly seal off an affected area.

- The occurrence of sheets of whinstone (dolerite) in the strata overlying the seam which forms an impervious layer and tends to retain gases within the coal seam as opposed to allowing the gas to escape to surface through breaks in the

strata (this will be dealt with in the disaster at the Durban Navigation Colliery No. 2 Pit in October 1926).

- Extraordinarily high methane emissions in pillar extraction sections from unworked coal seams which overlie the seam being stooped - this gas finds its way into the goaf; major subsidences may initiate explosions due to frictional sparking in this case.

- Spontaneous combustion in the goaf area as a result of snooks being left in the waste; excessive loose coal lying on the floor. Competent roof immediately overlying the stooped pillars may not totally smother the crushed snooks with the result that large voids may be present in the goaf.

- Dry and dusty conditions existing at the goaf edge in splits - no stonedust applied to the area along the goaf line. In many instances the dust on the roadway floors was measured at 50 mm thick and contained 82% combustibles (Hlobane No. 2). No attempt was made to sweep the areas and load out the dust. Neither was the dust allayed by watering down which was a prerequisite to being granted exemption by the Inspector of Mines from stonedusting (Hlobane No. 2).

- Untrained and uncertificated persons in the section charging up shotholes and blasting; removing goaf line timbers by attaching sticks of explosives with primers to each timber and then blasting the timber supports out; use of non-permitted explosives in the above practices; overcharging shotholes.

- Blasting goaf line fenders without the prior testing for the presence of methane.

- Allowing diverse breaches of the Mines and Works Regulations to occur in sections:

  i) failure to notify the Inspector of Mines of reportable accidents.

  ii) Failure to report a coal dust explosion to the Inspector of Mines.

  iii) Failure to appoint a Mine Overseer to assist in the management of the mine.

  iv) Failure to report breaches of Regulations to the Inspector of Mines.

- Use of contraband (matches) to light cigarettes while working in splits in the goaf.

- Faulty and non-flameproof electrical gear installed in the vicinity of the goaf.

- Duly appointed responsible officials not carrying out their duties and leaving the examination of the gas-filled goaf to untrained incompetent persons.

- Insufficient attention paid to ventilation appliances on the goaf edge resulting in the build-up of dangerous concentrations of methane in the vicinity of the goaf edge; leaking brattices; brattices rolled up to allow the easy passage of mobile machines.

- Dangerous quantities of methane on the goaf edge were not dealt with in a positive manner.

- Loose coal igniting as a result of a methane ignition.

- Failure to install bleeder roads and gas drainage holes from surface in pillar extraction sections. In all the Inquiries examined which deal with some form of secondary mining there was an absence of methane drainage in the goaf resulting in a build-up of methane inside the goaf and at the goaf line - in many incidences the ventilation appliances in the sections and in some instances the air quantities delivered to the area were inadequate to remove the methane and render it harmless.

- Lack of brattice curtains at auxiliary fans.

- Blown-out or overcharged shotholes (see Chapter 4).

- Lack of adequate fire-fighting appliances in a section where a fire breaks out (water supply at adequate pressure and quantity and fire extinguishers). The fire which developed after the explosion at the Durban Navigation Collieries was quickly doused using water and fire extinguishers.

- Roadways which have penetrated dykes are more than likely to be poorly ventilated and, paradoxically, to encounter excessive methane emissions or blowers. These two aspects lead to a dangerous condition in the form of an explosive methane/air mixture. Furthermore this danger is exacerbated by poor roof and sidewall conditions.

- Methane explosions send out shock waves which blow coal dust in roadways and haulages into heaps, thus increasing the danger of coal dust explosions.

- A coal dust explosion in a colliery may render it inoperative for anything from 6 to 20 weeks due to the total destruction of all ventilation appliances. In some instances the colliery could be closed permanently (Wankie Colliery 1972).

5.7 PRECAUTIONS TO BE ADOPTED

The preceeding sections have provided details of several documented and researched incidents. Based on these, the following conclusions and observations can be listed.

5.7.1 Bleeder roads at the furthest extremity of the goaf

In order to prevent the occurrence of methane/coal dust explosions on the goaf line and inside the goaf two critical objectives should be met. These are:

- To adequately ventilate, by coursing, the goaf line so as to remove dangerous methane concentrations.
- To simultaneously ventilate the goaf in such a manner that no dangerous quantities of methane accumulate in this region.

The alternative to these recommendations would be to practice pillar extraction without bleeder roads and the case studies investigated have indicated that the goaf area tends to become a reservoir for methane which finally finds its way to the working faces. The gas remaining in the goaf area could also lead to the build up of an explosive mixture.

There are several alternatives available for the bleeding of methane from individual panels - total extraction or incomplete extraction. Total extraction of a production panel involves the "second mining" of all possible pillars in the panel. Air must be able to flow through the goaf that remains after the extraction is completed. In many cases, after extraction is completed, the roof in the goaf falls tightly, compacts itself, or even cements itself together in such a manner that it will not allow air to pass over the goaf. The practice of bleeding requires that a differential air pressure be established that

will cause air to flow from the goaf into the bleeder returns.

The one alternative to total panel extraction is incomplete panel extraction, which is the practice of systematically leaving a row of pillars to provide an air passage through the panel. Incomplete panel extraction should not be confused with partial pillar extraction, which is the practice of extracting individual pillars in such a manner that caving of the main roof is not achieved.

Figures 15, 16, 17, 18 compare the four methods of total extraction.

The third alternative considers leaving the two barrier roads and the brick stoppings intact and returning the bleeder ventilation along these roads to the main return. This method is only used when bleeder roads are not available at the end of the panel where pillar extraction commences. The one disadvantage of this system is that roadway crush may damage brick stoppings thereby allowing bleeder ventilation to short circuit and not clear the goaf of methane concentrations. This method has the advantage that if the goaf 'packs tight', and reduces the bleeder road air quantities, brick stoppings marked "A" for instance can be removed thus increasing air flow to the goaf.

Figure 15 shows details of bleeder road ventilation where the goaf does not 'pack tight' and air passes over the area for the duration of pillar extraction in the panel.

However, it is often the case that the goaf eventually 'packs tight' and reduces air flow through the bleeder roads to a quantity of air some 40% of the target quantity. When this situation occurs the method shown in Figure 16 is used whereby the bleeder road to the adjacent unstooped panel is successfully moved to follow the stooping line.

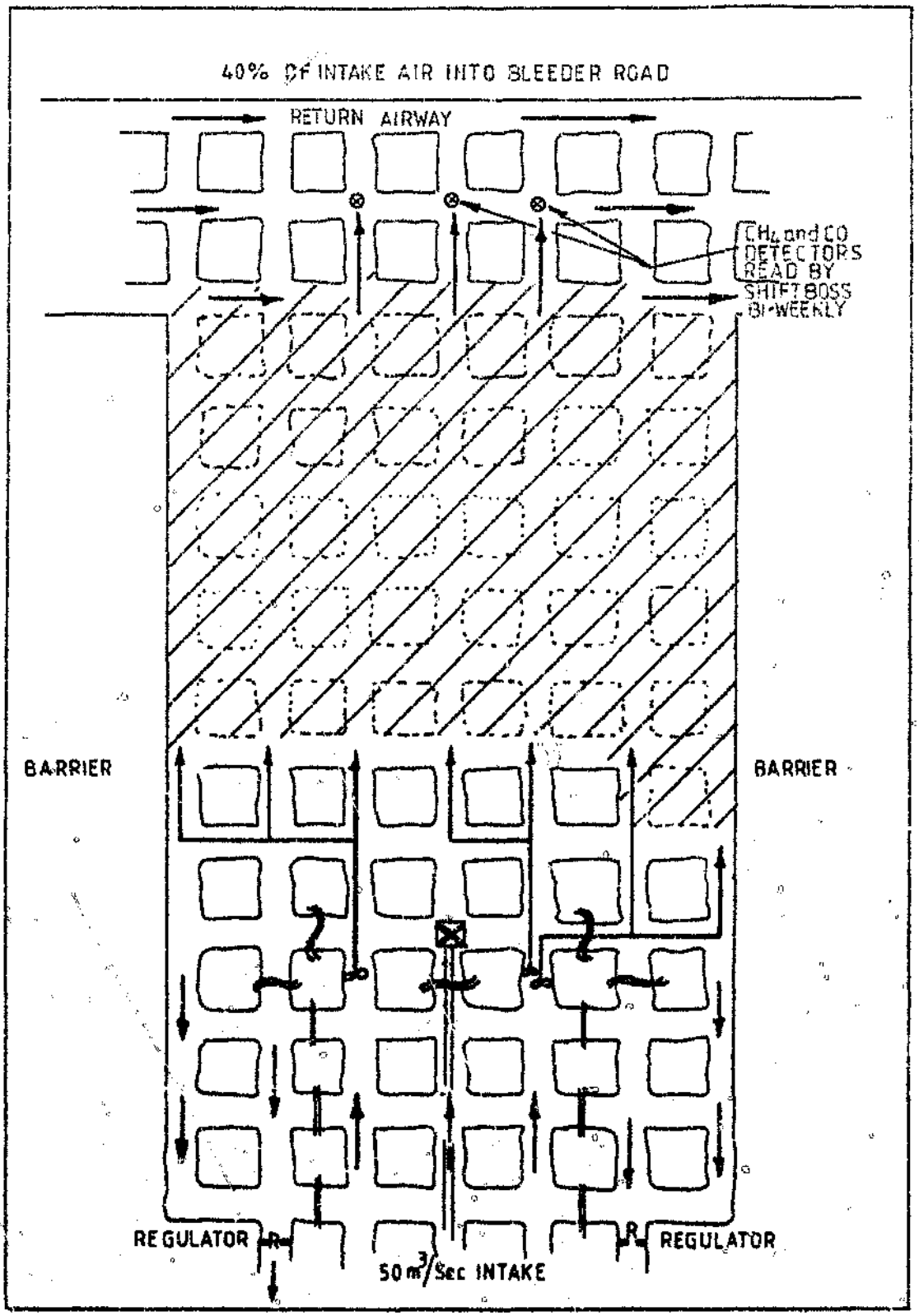


Figure 15

BLEEDER ROAD VENTILATION WHERE GOAF DOES NOT PACK TIGHT

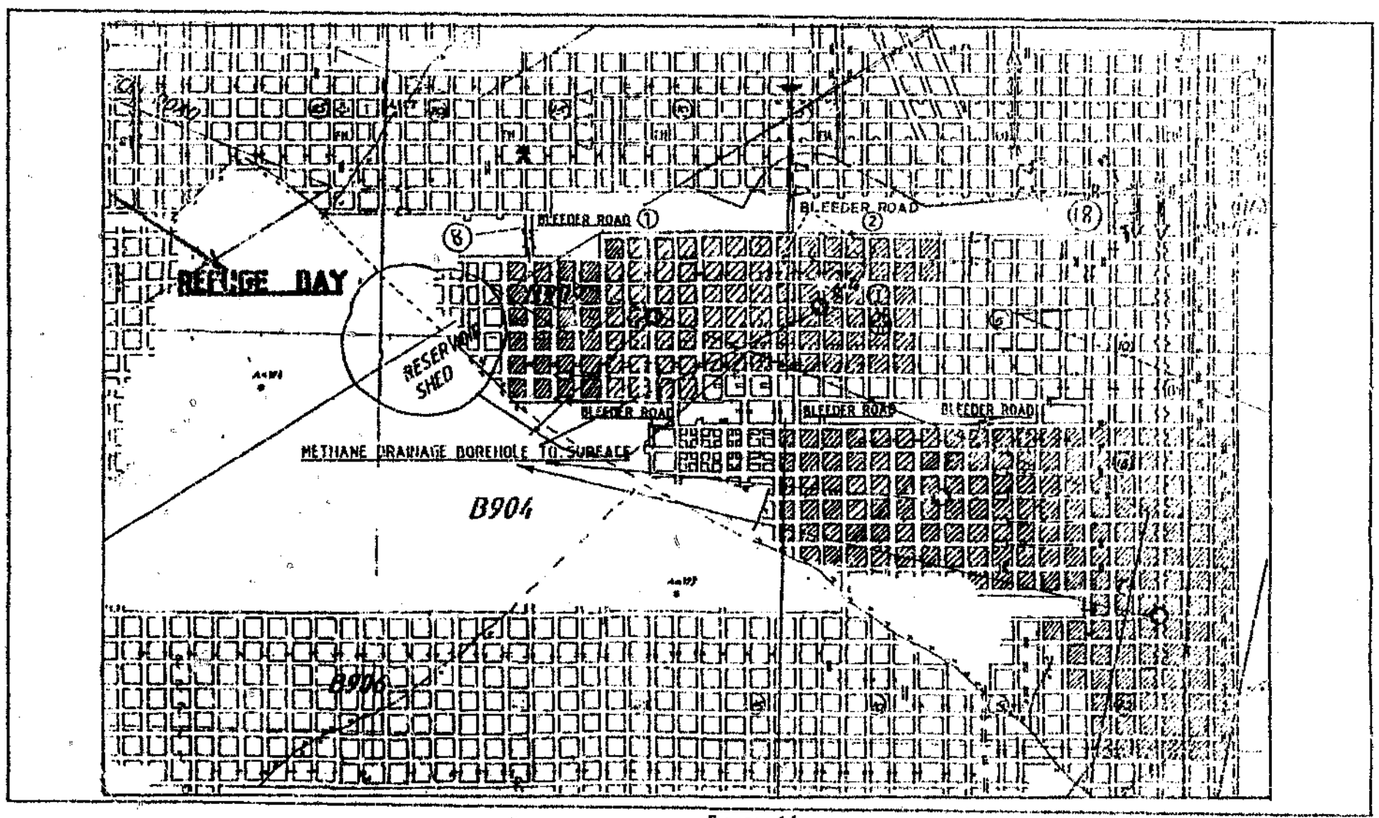


Figure 16

BLEEDER ROADS – PILLAR EXTRACTION – WITH BLEEDER ROADS
DEVELOPED INTO ADJACENT PANEL AS STOPING
PROCEEDS

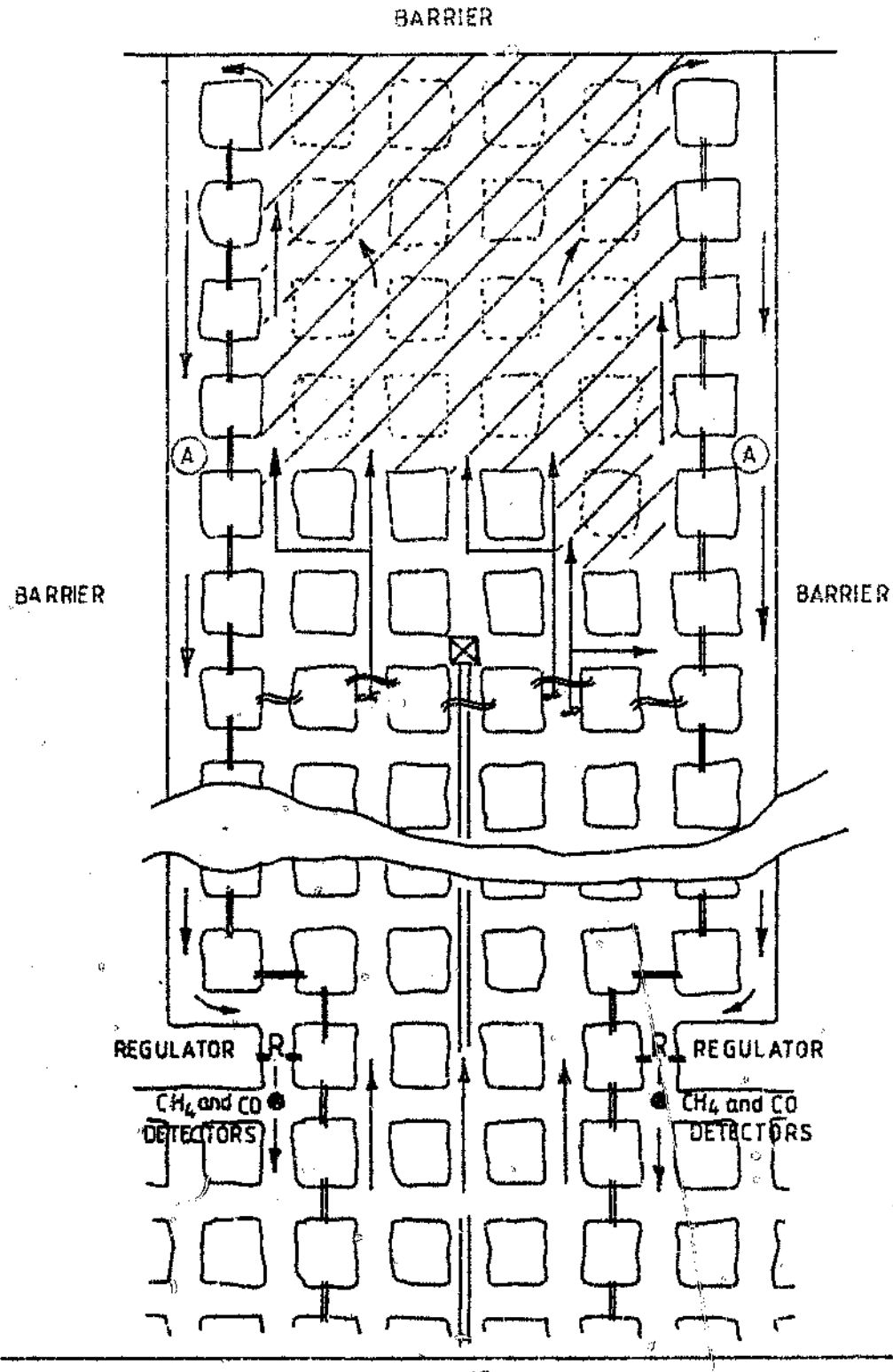


Figure 17

BLEEDER ROADS – PILLAR EXTRACTION
LEAVING BARRIER ROADS AND STOPPINGS
INTACT

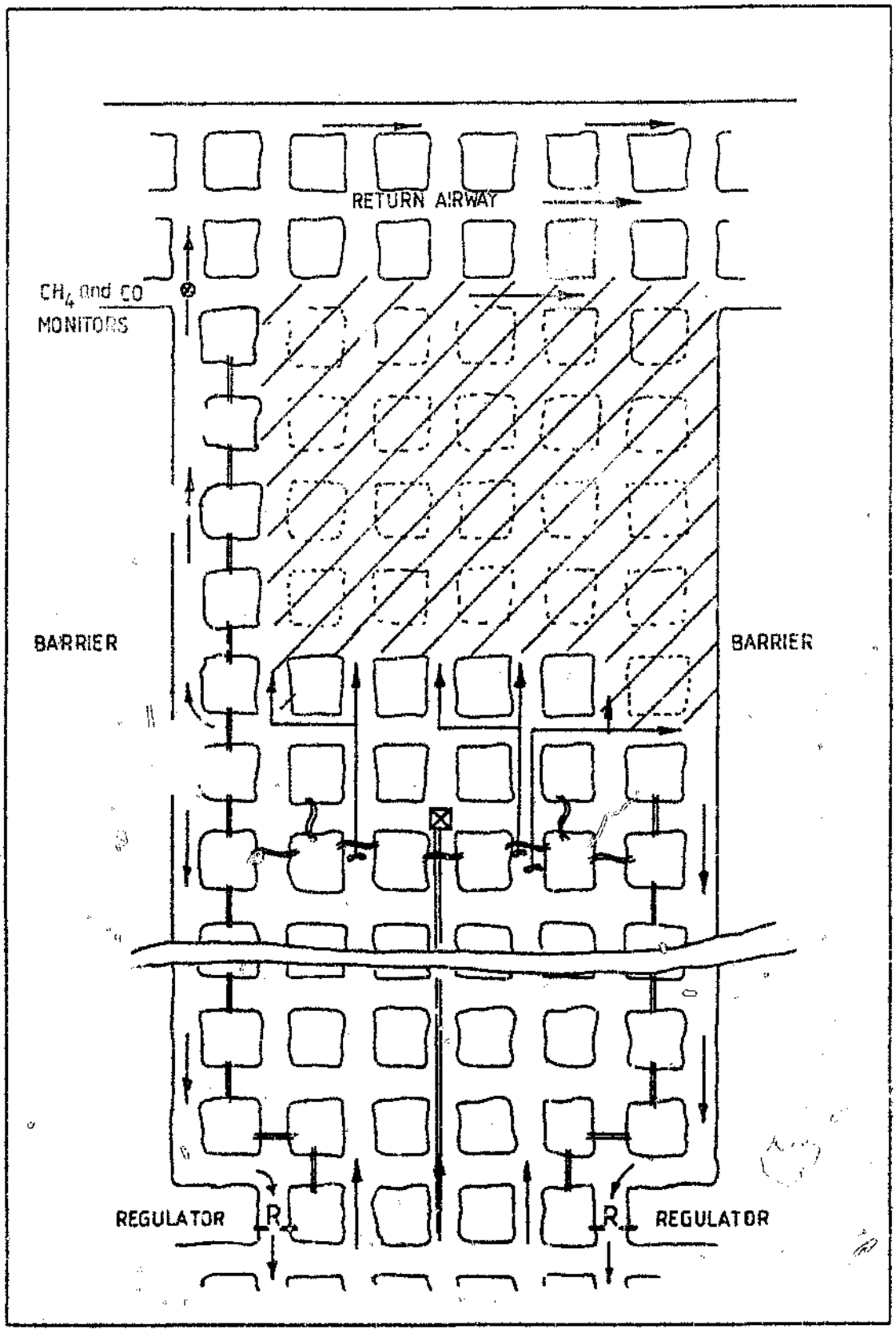


Figure 18

BLEEDER ROADS - PILLAR EXTRACTION
LEAVING A ROW OF PILLARS TO PROVIDE
AN AIR PASSAGE THROUGH THE PANEL

Experience in the South Rand coalfield particularly in the area where the overlying strata above the seam consists of a competent sandstone is that, apart from one recorded instance, all goafed areas eventually 'pack tight' and limit bleeder road ventilation.

Intake quantities of approximately 40 m^3/second should be available for pillar extraction sections of which 45% or some 18 m^3/second should be bled over the goaf.

Figure 19 shows a typical ventilation arrangement for pillar extraction.

5.7.2 Methane drainage from the goaf through surface boreholes

Notwithstanding the use of bleeder roads the removal of methane accumulations in the goaf is not totally guaranteed as is proved below.

The experience at the Balgray explosion in 1979 when a 200 mm diameter borehole was drilled from surface to the goaf indicated that it was not necessary to use a fan on surface to evacuate the atmosphere in the goaf; the goaf atmosphere, when the drill hole reached the goaf, blew out of the borehole - against the mine fan pressure - similar to blow-outs experienced in gas wells.

In addition to bleeder roads 75 mm diameter boreholes are drilled from surface to the goaf at the Durban Navigation and Springfield collieries to drain excess methane to surface and it is recommended that this practice be used in all secondary mining methods (including longwalling). This method is extensively used at the Island Creek Coal Company in West Virginia, U.S.A. where high methane emissions are experienced in their longwall panels.

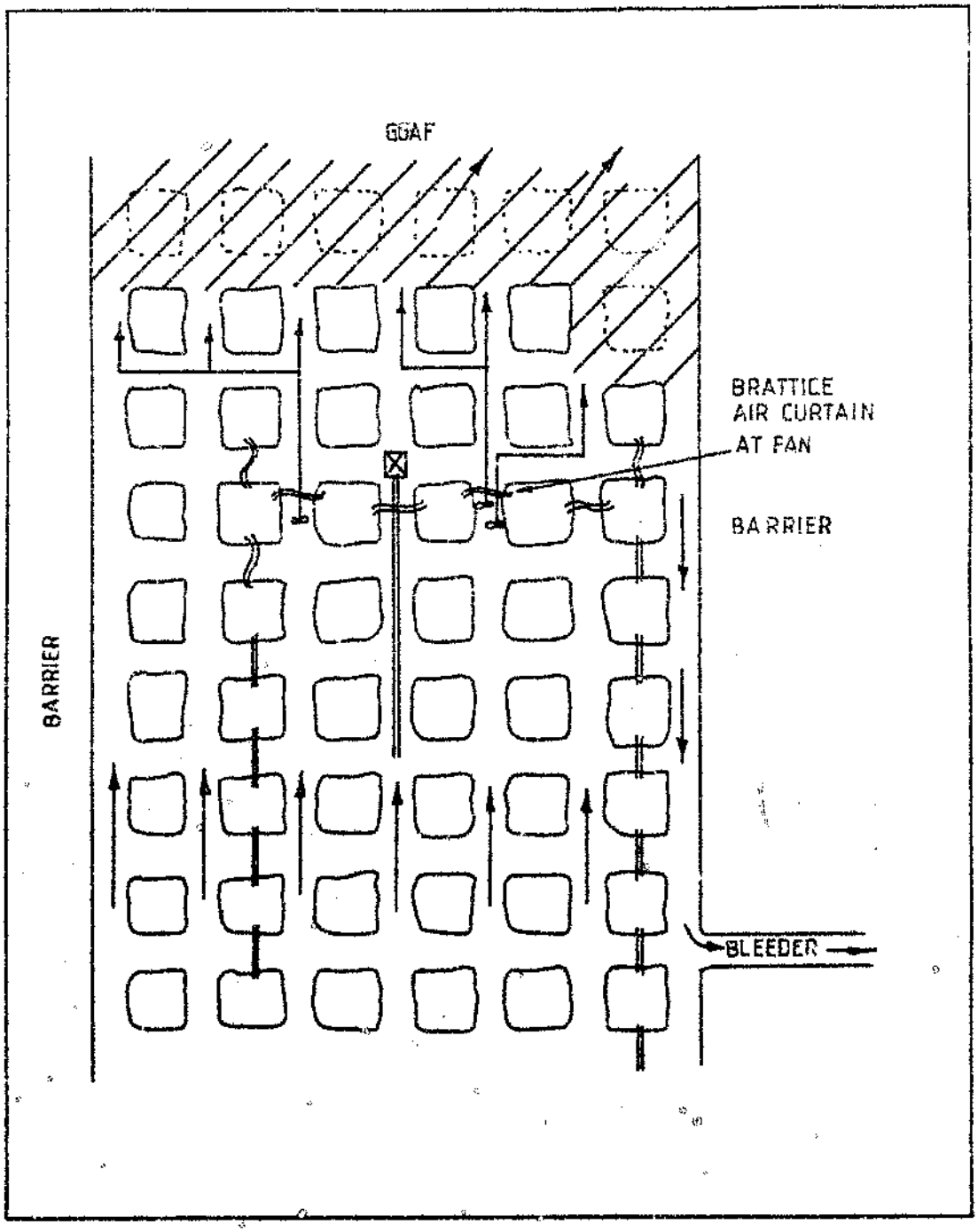


Figure 19

TYPICAL VENTILATION ARRANGEMENT
FOR PILLAR EXTRACTION

Skow (1964-1979) states that when virtually all the coal is extracted and the roof allowed to fall, a zone of tightly compacted rubble (goaf) is created. Air does not flow easily through this rubble, and large quantities of methane may flow into the goaf from the fractured strata above the coalbed. The large volumes of methane that accumulate in the goaf areas frequently cause severe ventilation problems during mining of adjacent areas. When the barometer falls, the methane-laden air in the goaf expands and may enter portions of the mine where a spark source is more likely to be found. Even if sealed, goafs are not completely airtight and can leak methane-laden air into the active portions of the mine.

To assist the conventional ventilation systems in some gassy mines, vertical boreholes are drilled from the surface into the overburden ahead of mining. When the mining face passes the hole, roof subsidence causes cracks in the overburden, and methane that normally would be released into the goaf area flows to the borehole and is drawn to the surface. Such holes can remove as much as 2800 m^3/day of methane from a mine and reduce methane emission underground by more than 50%. Bureau of Mines tests have shown that a "short hole", one that terminates well above the coal, is just as effective as a hole drilled within a few feet of it, is less expensive to drill, and discharges a gas having a higher methane content.

Flow rates from goaf ventilation holes tend to drop rapidly, for example, from 28000 m^3/day to only 2800 m^3/day over a period of one year. The concentration of methane also decreases with time, from as much as 100% to 50% or less within several months. Despite these factors, gas drained from goafs can be used as boiler fuel or for gas turbine generation of electricity. Since vertical boreholes to goaf areas are simple, effective and relatively inexpensive, they are being widely used in the coal mining industry.

The 200 mm diameter drill hole at Balgray Colliery (drilled after the three explosions) was drilled primarily to allow the goaf atmosphere to be analysed for methane and oxygen. The results of the analysis were to be used by management to decide whether it would be safe to send men into the roadways leading into the section to replace seals blown out by the third explosion. What was not realised at the time was that the hole could be used to drain methane from the goaf - either by natural ventilation or by forced exhaust means.

The holes are drilled prior to maximum extraction commencing and are drilled at intervals of 300 - 400 metres apart along the centre line of the panel. The depth of hole depends on caving characteristics. Where the superincumbent strata caves to the base of a dolerite sill holes should be drilled to a depth of ± 20 metres below the base of the sill. Where the superincumbent strata does not cave more than 30-40 metres above the extracted seam, holes should be drilled to an horizon ± 20 metres above the seam.

Figures 20 and 21 contrast the different drilling depths required for differing caving heights.

(a) Results obtained from recent studies at Springfield Colliery

Figures 22 and 23 show respectively the stratigraphic column in the B900 area and a plan of the sections where pillar extraction took place and indicating the methane drainage holes to surface.

The hole on surface is covered with a plastic U-tube to allow methane to escape to atmosphere through the water in the U-tube. A plastic U-tube is used so as to not attract lightning strikes. See Figure 24.

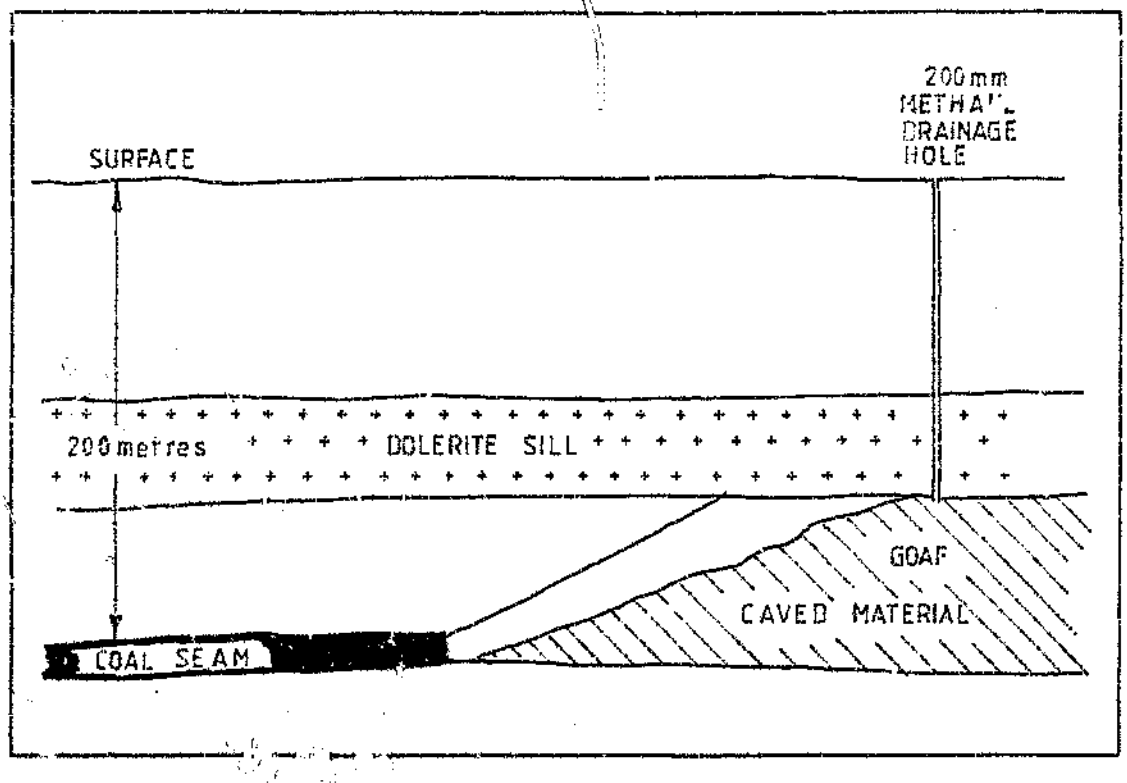


Figure 20

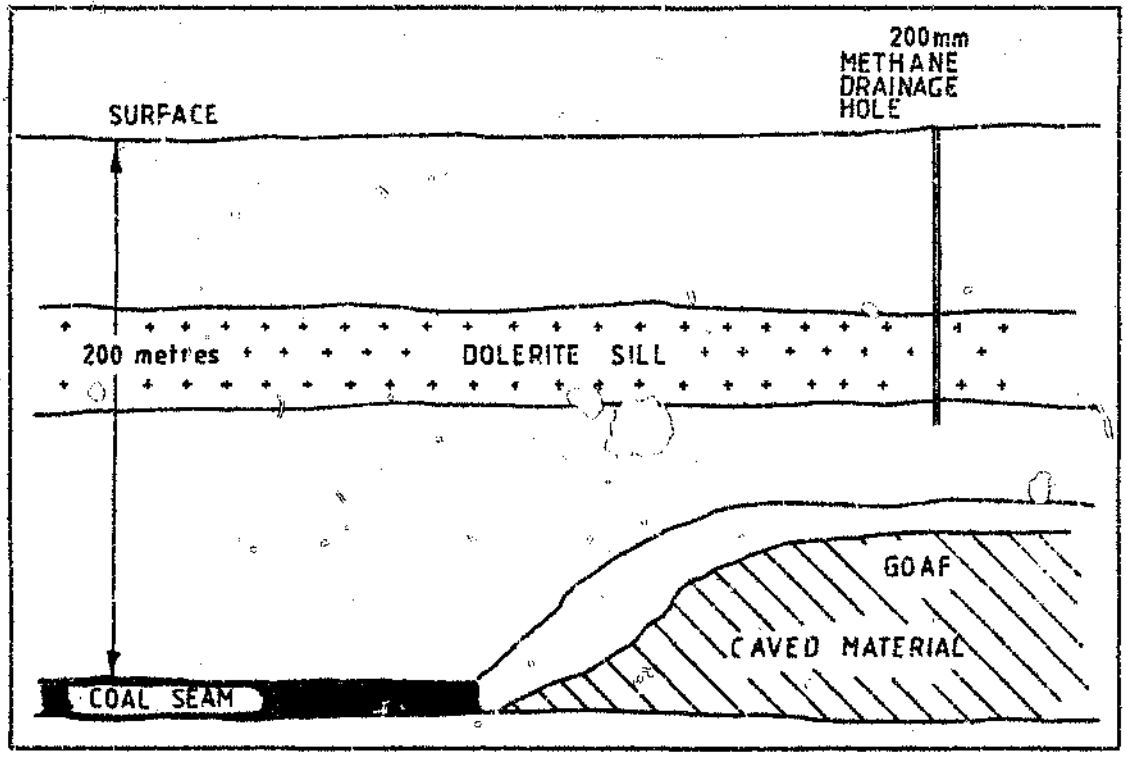


Figure 21

DIFFERENT DRILLING DEPTHS REQUIRED
FOR DIFFERENT CAVING HEIGHTS

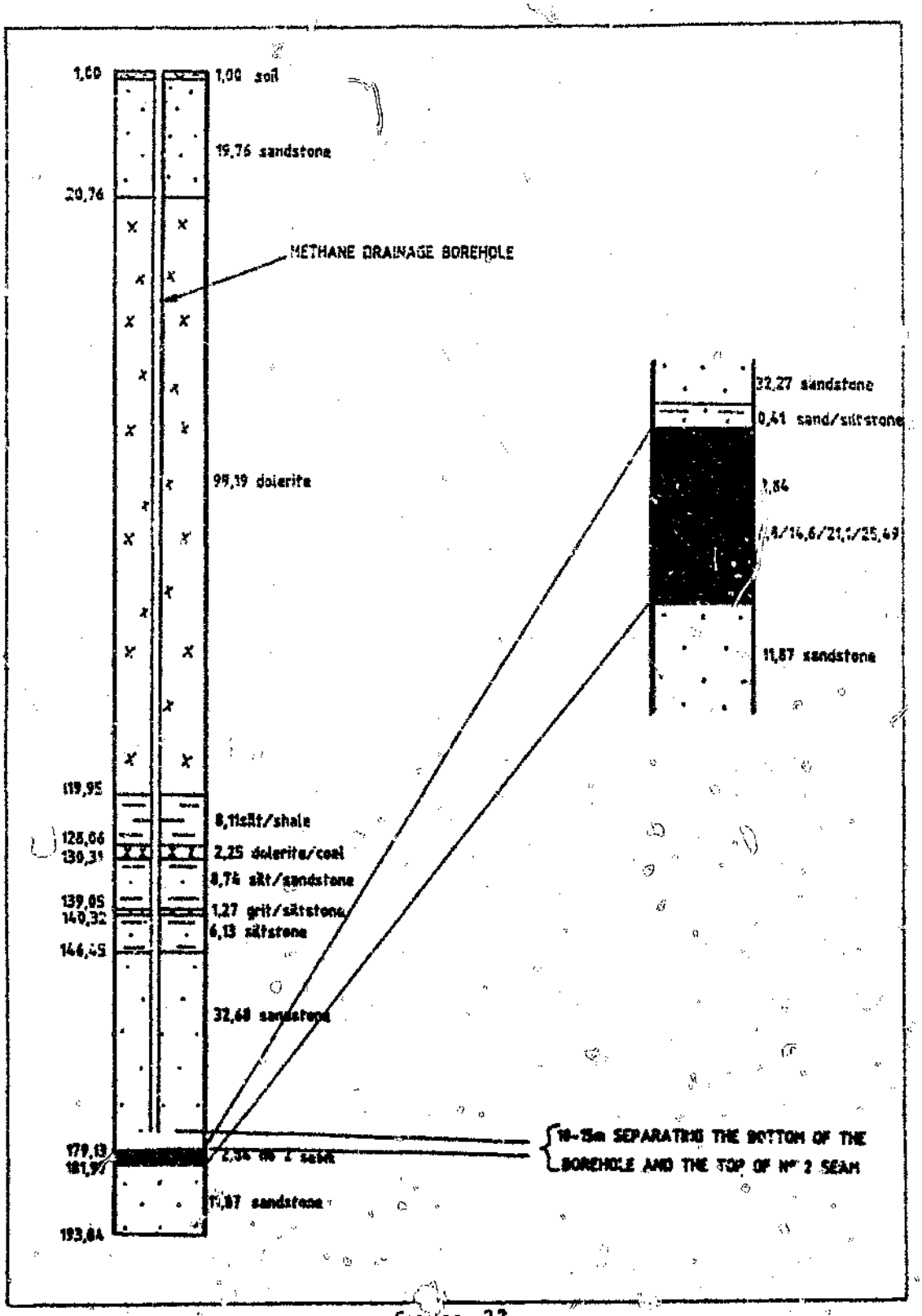


Figure 22

STRATIGRAPHIC COLUMN IN B900 AREA
SPRINGFIELD COLLIERY

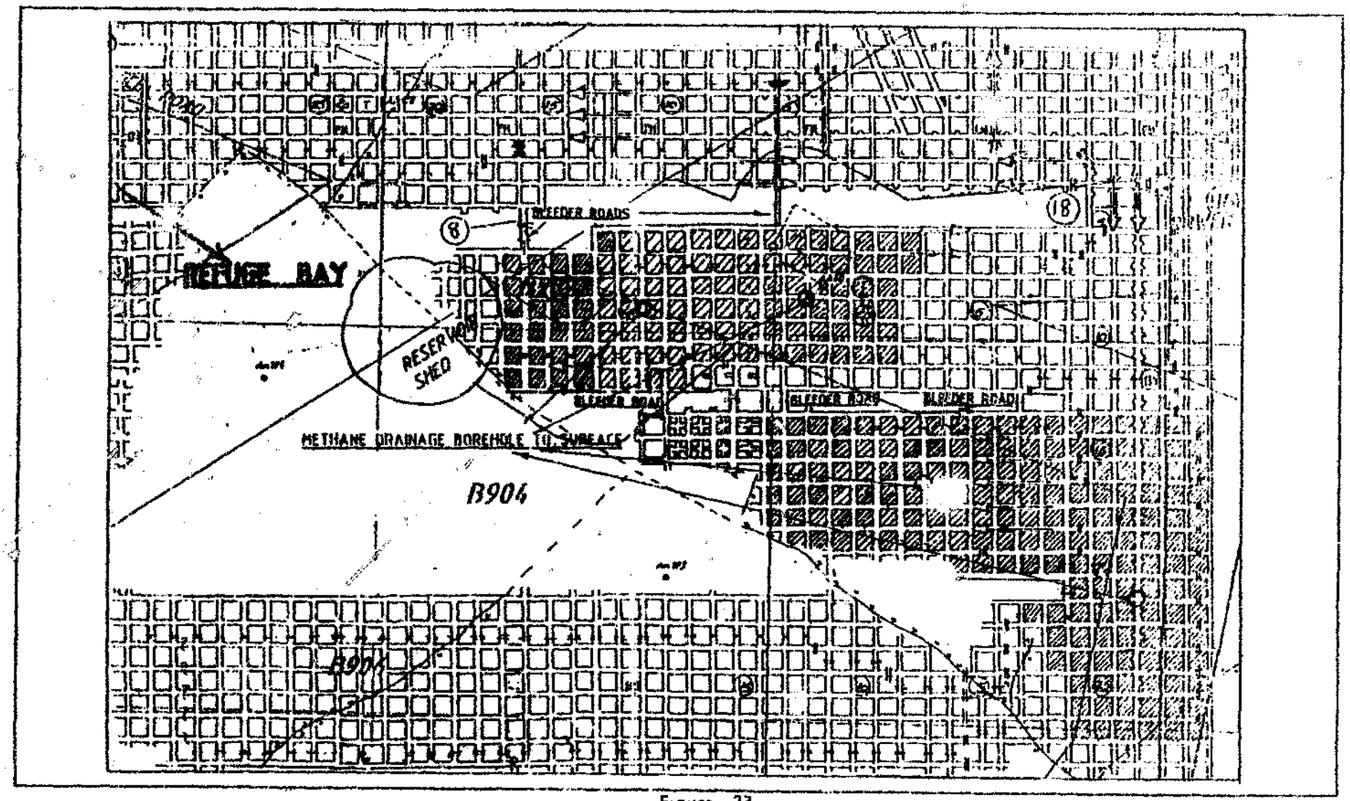


Figure 23

SECTIONS WHERE PILLAR EXTRACTION TOOK PLACE SHOWING METHANE DRAINAGE HOLES TO SURFACE SPRINGFIELD COLLIERY

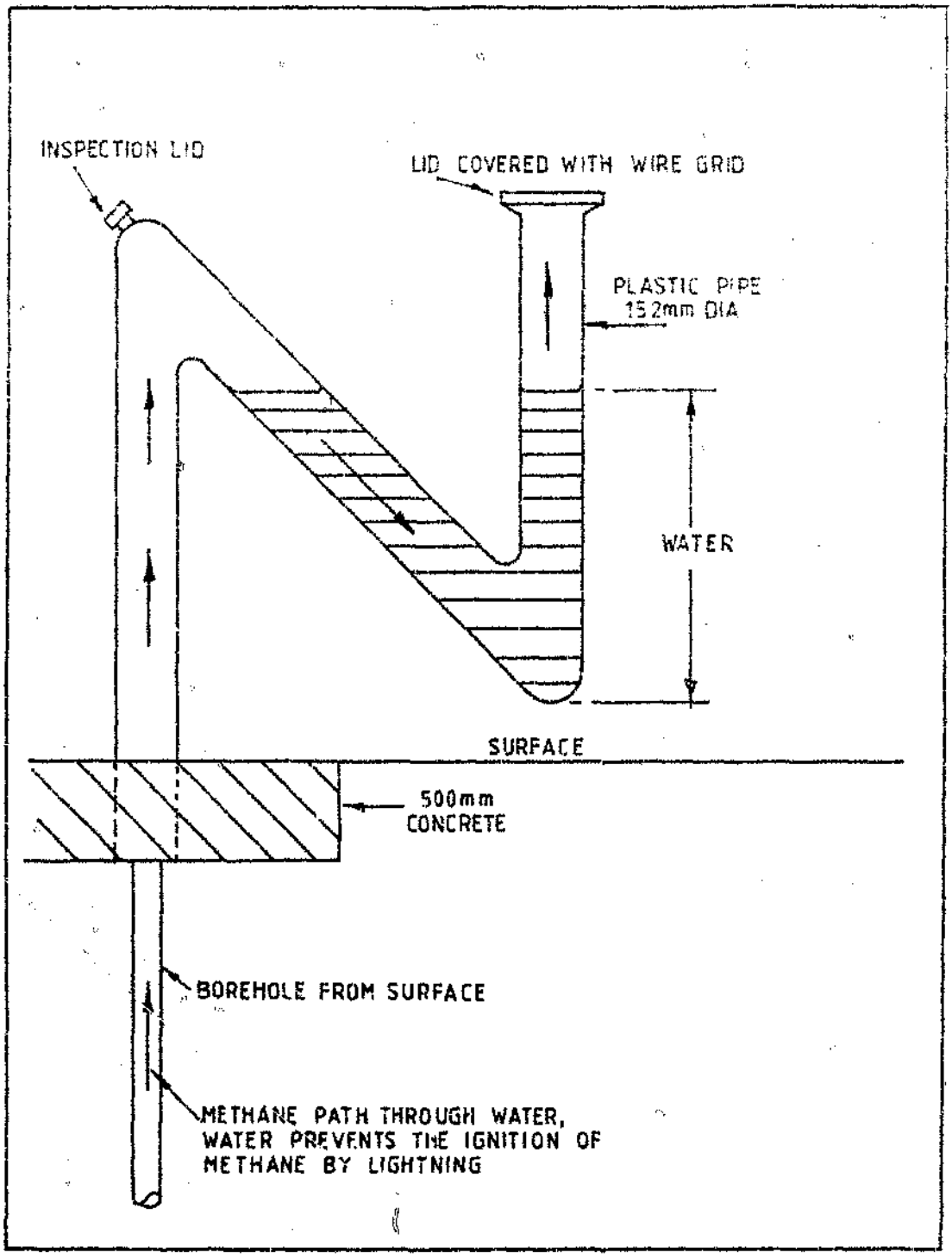


Figure 24

'U' TUBE ARRANGEMENT ON TOP OF METHANE DRAIN HOLE

Barometric pressure is recorded on a continuous basis indicating that a 'high' pressure is experienced during the mornings and a 'low' pressure during the afternoons.

As mentioned, this phenomena has a great influence on the readings obtained. Gas samples, by means of a Mogul gas pump, were taken on a weekly basis and sent for analysis to the Chamber of Mines. Testing for oxygen, carbon monoxide, methane and hydrogen was carried out twice daily (mornings and afternoons). The results were recorded in a book and any trend could be easily followed. Handheld instruments, locally obtainable, were used for daily testing.

Results

Methane gas escaped to surface via the surface borehole prior to goafing. This phenomena continued for three days before the methane gas tapped in the strata above the coal seam was cleared. After this initial period of three days the air in the hole became stagnant and the only change of air movement in the hole was caused by the influence of the barometric pressure changes. No further methane readings were obtained from the air emanating from the hole until the first goaf appeared.

For a period of 10 days after the first goaf had holed with the borehole, air was constantly upcasted through the borehole and sometimes contained methane gas of up to 63,5%. Thereafter the air started to downcast continuously for a period of 74 days. After this period of 74 days the air again started to downcast on the low barometric prescure period of the day (pm) and upcasting during the high pressure period of the day (am). Methane gas readings of up to 27,8% (28 November 1988) were obtained. Table 5.3 shows the details of the methane recorded in the drainage hole and these figures are graphically represented in Figures 25 and 26.

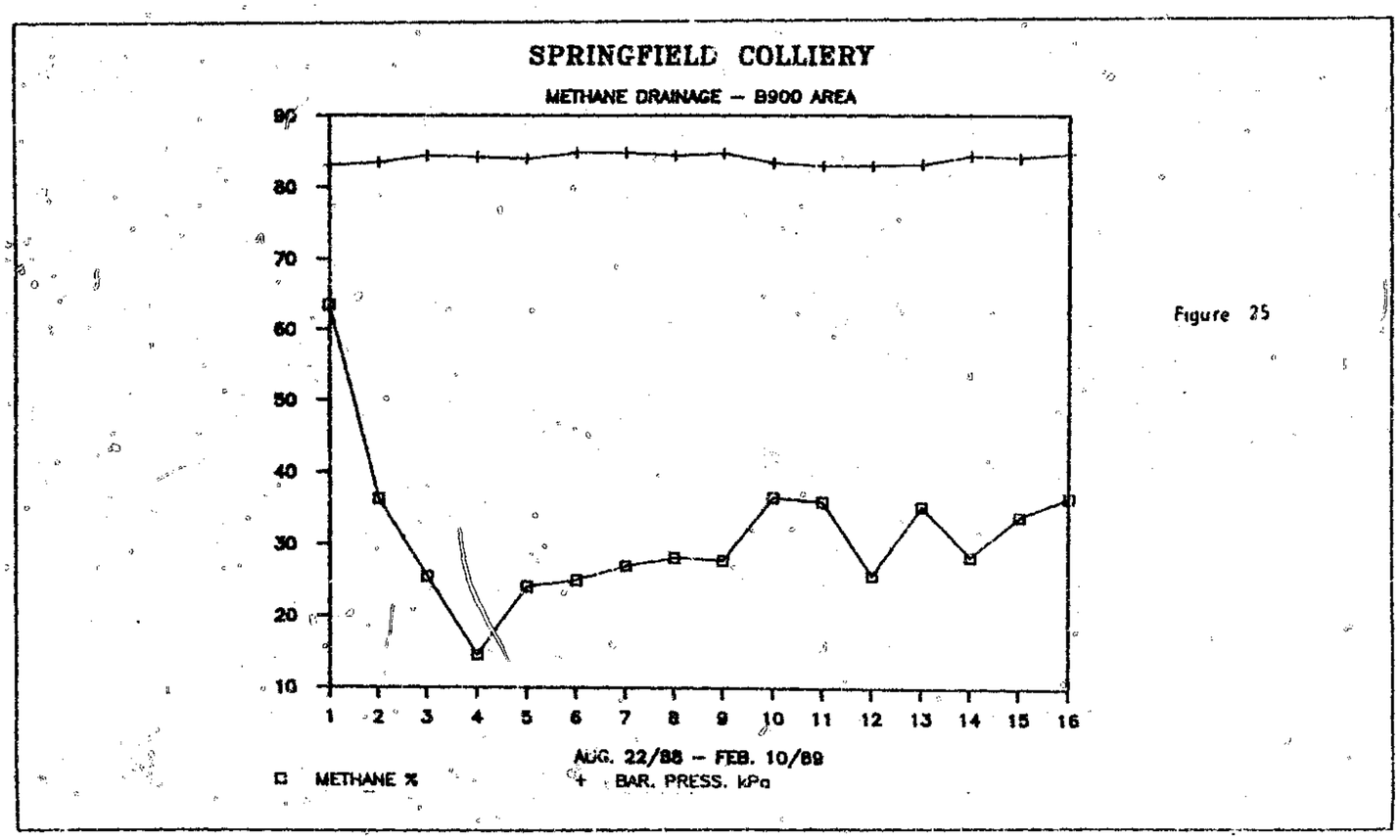

SPRINGFIELD COLLIERY
METHANE DRAINAGE – B900 AREA
90
80
70
60
50
40
30
20
10
1
2
3
4
5
6
7
8
9
10
11
12
13
14
15
16
AUG. 22/88 – FEB. 10/89
METHANE %
BAR. PRESS. kPa

Figure 25

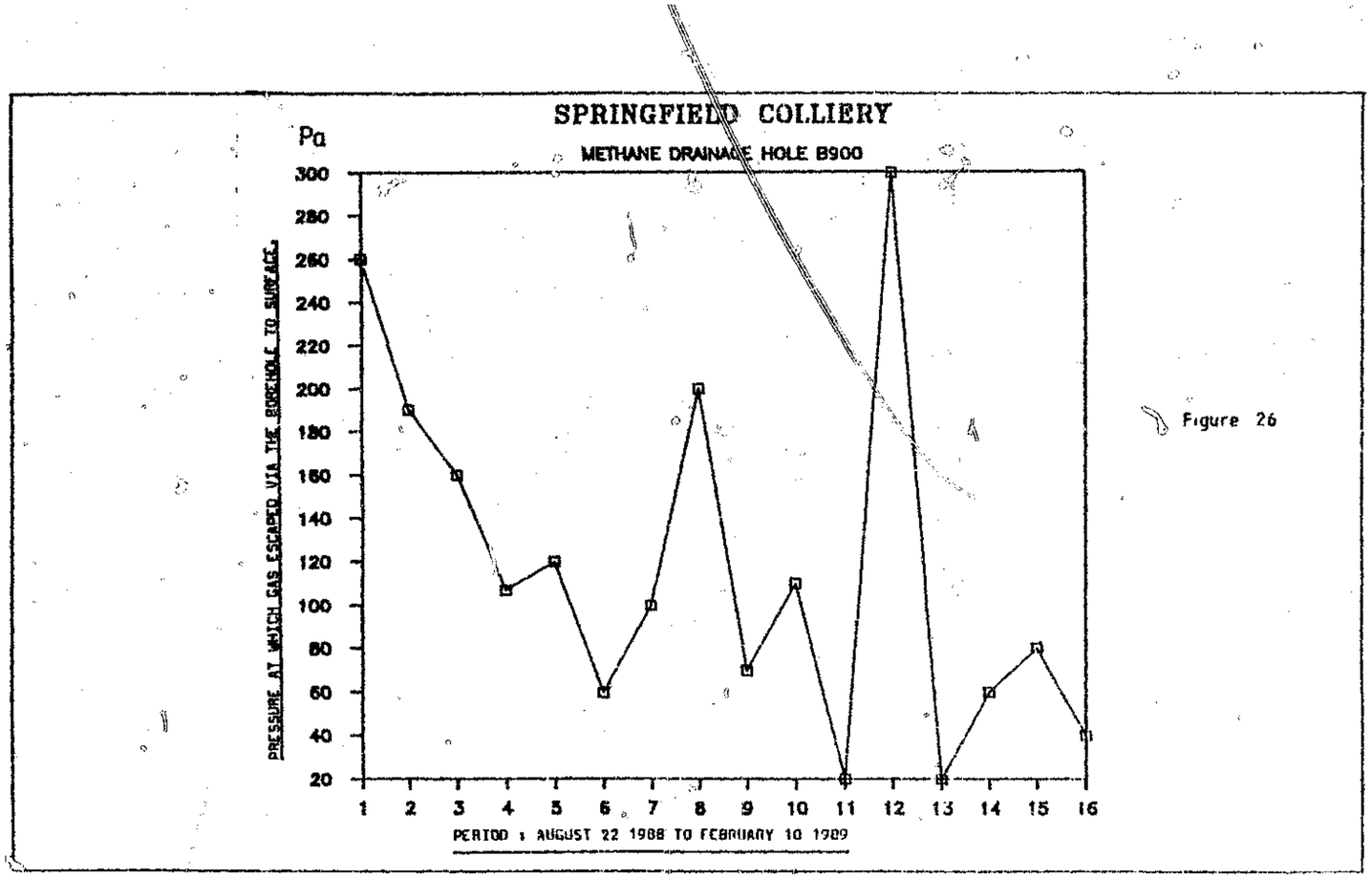

SPRINGFIELD COLLIERY
METHANE DRAINAGE HOLE B900
Pa
300
280
260
240
220
200
180
160
140
120
100
80
60
40
20
PRESSURE AT WHICH GAS ESCAPED VIA THE BOREHOLE TO SURFACE.
1 2 3 4 5 6 7 8 9 10 11 12 13 14 15 16
PERIOD : AUGUST 22 1988 TO FEBRUARY 10 1989

Figure 26

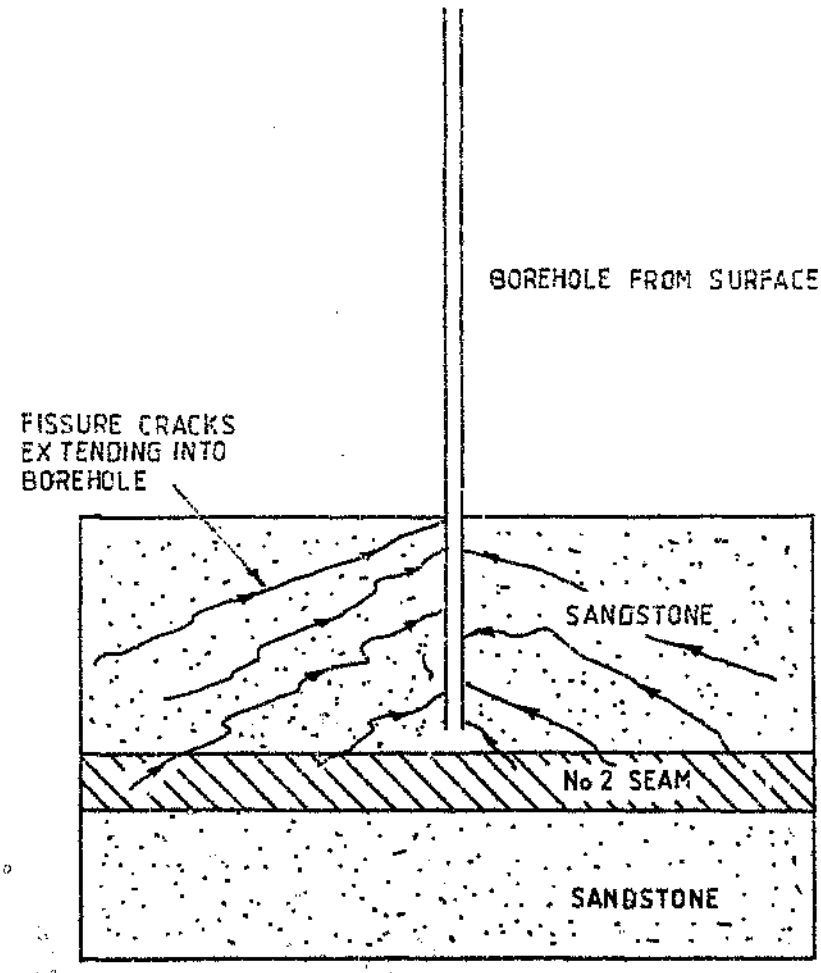
Table 5.3

Gas analysis of methane drainage borehole No. 1
Section B902 (stooping)

No.	Date	% Methane	% CO	% Oxygen	Barometric Pressure kPa	Pressure difference of gas ex hole Pa
1	22.8.88	63,5	5	2,8		
2	22.8.88	36,3	4	10,5		
3	24.8.88	25,5	4	15,5		
4	2.9.88	14,6	0	15		
5	2.12.88	24,1	0	5,6		
6	2.12.88	25	0	5,1	84,88	60
7	7.12.88	27	0	4,8	84,88	100
8	13.12.88	28,2	0	5,6	84,44	200
9	22.12.88	27,8	1	5,3	84,79	70
10	4.1.89	36,6	0	3,1	83,43	110
11	9.1.89	36	0	3,3	83	20
12	11.1.89	25,7	0	3,9	83,04	300
13	16.1.89	35,3	0	4,2	83,23	20
14	26.1.89	28,3	0	5,7	84,4	60
15	2.2.89	33,9	3	1,7	84,15	80
16	10.2.89	36,6	0	2,8	84,73	40

(b) Conclusion

The initial upcasting of the air in the borehole before stooping commences is caused by the release of gas which is trapped in the overlying strata and contained under pressure. The release of the trapped gas creates a lower pressure at the cavity from where the gas escapes. This in turn allows gas to escape from the coal seam via fissures leading into the exposed area and further into the borehole - Figure 27.



10 - 15m SEPARATES THE BOTTOM OF THE BOREHOLE FROM THE TOP OF THE NUMBER 2 SEAM

METHANE GAS ESCAPES VIA CRACKS AND FISSURES FROM THE OVERLYING STRATA AND THE NUMBER 2 SEAM INTO THE BOREHOLE AND OUT TO THE SURFACE

Figure 27

METHANE DRAINAGE BOREHOLE

Eventually the gas pressure in the borehole and the barometric pressure on surface becomes equal and the release of gas ceases. The influence of the change in barometric pressure now causes the air in the borehole to either upcast or downcast resulting in the movement of air into or out of the overlying strata and the coal seam.

When the goaf line reaches the borehole, a direct connection between the mine ventilation network and the borehole is established. The air immediately commences downcasting, as it is directly influenced by the negative pressure created by the surface fans. However, at borehole No. 1, at section B90?, ten days lapsed before this happened.

After ten days, a constant flow of air down the hole was monitored for a period of 74 days. Any make of gas in the vicinity of the borehole was removed by the natural flow of air in the section and over the goaf through the bleeder road.

Goafing in the worked out area continued and the borehole once more became separated from the ventilation network underground. This was noticed when the air in the hole again started to upcast during high pressure and downcast during the low barometric pressure period of the day. A "solid" goaf, which separates the borehole from the influence of upcast fan pressure, is created and gas is removed through the borehole from the sealed area at the back of the goafed area.

It is also calculated that, depending on the percentage of methane gas emanating from the surroundings, the methane gas may, in fact, overcome the pressure created by the surface fans and might, in some cases, lead to the upcasting of air during the original 'direct connection' period due to the low specific gravity of the methane (0,5) and the heat in the goafed area.

The process is illustrated in Figure 28 which depicts the five stages of methane drainage in the hole and the upcasting and downcasting cycles.

5.7.3 Prompt sealing of fires

The author in dealing with fires underground (and spontaneous combustion heatings) has been guided by the paper "Opening an old fire area at Northfield Colliery" by Watson (1964) who stated:

"From all accounts ten persons were gassed on one occasion (in a stooping section) and the following day twelve were again treated. We are told the section was double-shifted with a view to rapid extraction so as to smother and arrest the heating. This had no effect and the situation became worse. It was then decided to seal off the section at the first dyke-outbye at Pillar 20. Much time we assume was lost, due to the recovery of material from the section and also the preparation of sealing sites." Watson then concluded that the delay in sealing off the affected section finally resulted in the explosion which destroyed one of the seals.

At Bulgray Colliery in 1979 much time was lost in sealing off the affected area after the second explosion and at the Glencoe Colliery disaster in February 1908 it is recorded that:

"Your Commissioners are of opinion that had the Underground Manager of Glencoe Colliery carried out his intention - shown by ordering material to be sent into the mine - and promptly built off the fire, further explosions would have been prevented; and that he erred in assuming that the fire had been extinguished by the fall of roof, the conditions pointing to an opposite conclusion."

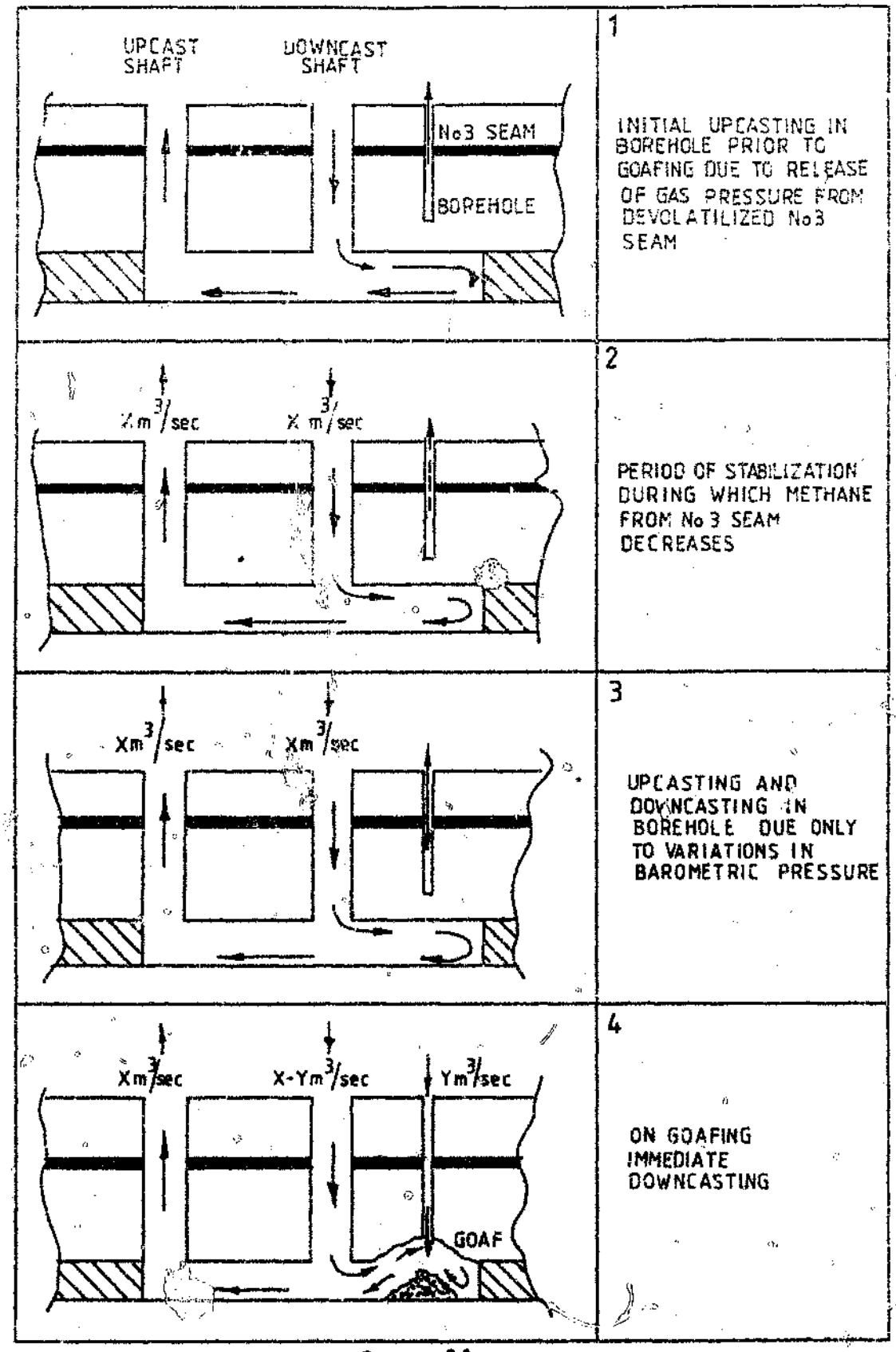
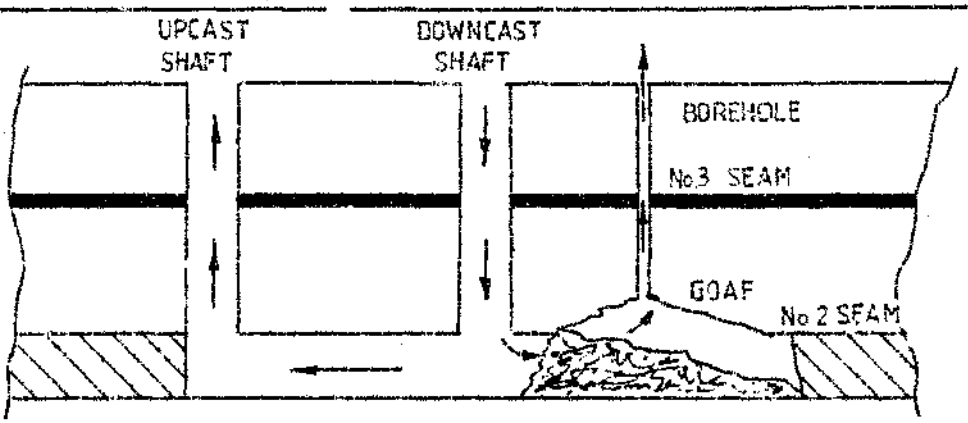


Figure 28

STAGES 1-4 OF METHANE DRAINAGE
FOR STAGE 5 AND DESCRIPTION SEE
FOLLOWING PAGE



5 STAGE 5 OF METHANE DRAINAGE

GOAF PARTIALLY SOLID
BOREHOLE UPCASTING 30% TO 40%
METHANE SOMETIMES OVERCOMING
FAN PRESSURE DUE TO HEAT IN THE
GOAF AND THE LOW S.G OF METHANE

Figure 28

THE 5 STAGES OF METHANE DRAINAGE

FOR STAGES 1-4 SEE PREVIOUS PAGE

Unless the fire can be speedily dealt with by either loading the coal out of the mine or dousing it with water it should be sealed off with the greatest speed.

The dangers which arise as a result of underground fires and spontaneous heatings can be classified as follows:

- The toxic effect of fires on the mine atmosphere.
- The asphyxiating effect of fires on the mine atmosphere.
- The danger of methane explosions.
- The danger of coal dust explosions.
- General dangers.

The toxic and asphyxiating effects of mine fires falls outside this investigation which investigation tends to highlight the explosions which have resulted from fires.

An active fire provides the ignition source to initiate a methane explosion and pioneer work by Coward (1929) who researched the relationship of various mixtures of methane, oxygen and nitrogen led to the well known "Cowards triangle" shown in Figure 29.

Coward (1929) stated that if the proportion of methane, oxygen and nitrogen in a mixture of these three gases are known, reference to Figure 29 will show at once whether the mixture is explosive or not, or whether it can be made explosive by admixture with air in suitable proportions. All that is necessary is to find in which area the point expressing the composition of the mixture lies. For example, if as shown, the mixture contains 9 percent of methane and no more than about 7,7 percent of oxygen, it will neither explode nor can it form an explosive mixture with air, whatever be the proportion in which they are mixed. If however, the oxygen content lies between

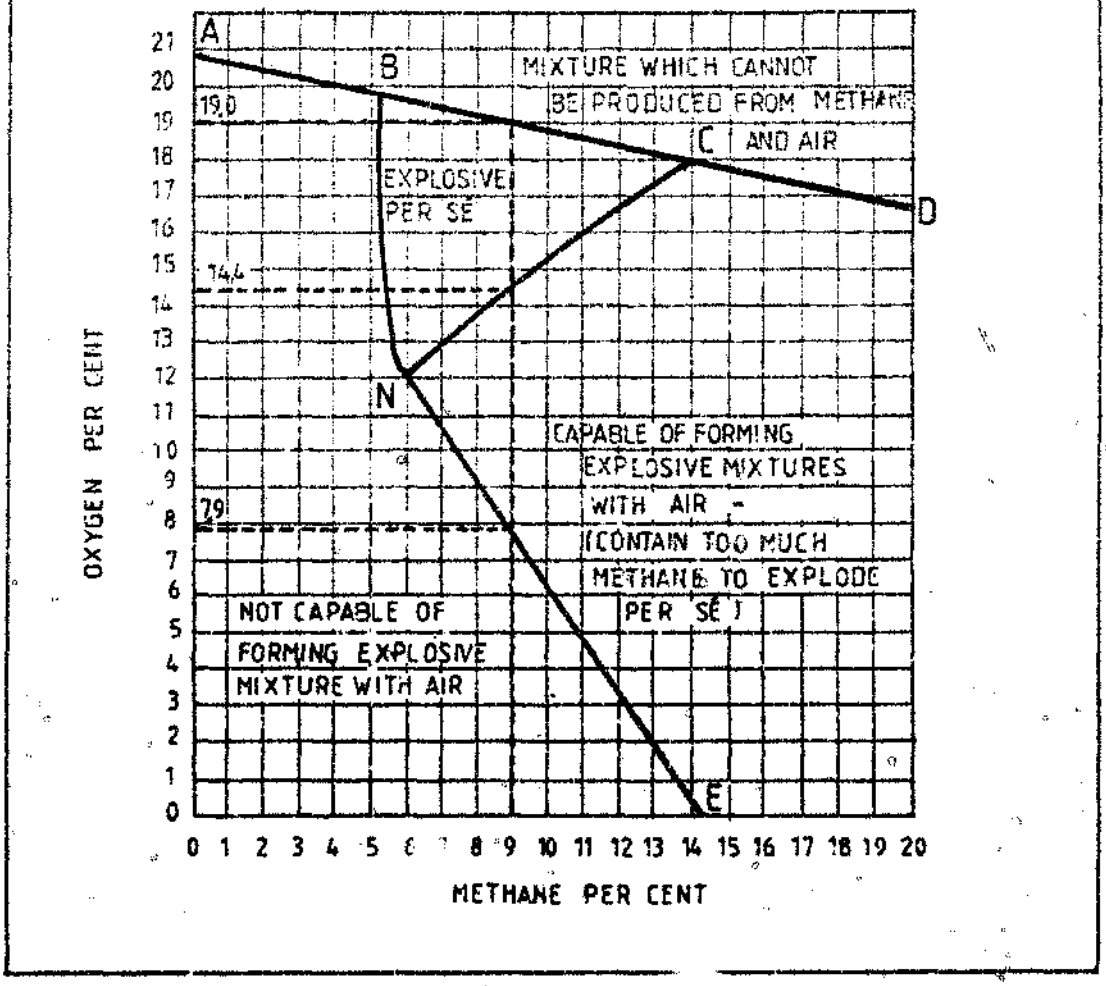


Figure 29

RELATIONSHIP BETWEEN THE QUANTITATIVE COMPOSITION AND THE EXPLOSIBILITY OF MIXTURES OF METHANE OXYGEN AND NITROGEN (AND CARBON DIOXIDE)

COWARD'S TRIANGLE

about 7,7 and 14,4 percent, the mixture, although not explosive itself can be mixed with suitable amounts of air to form an explosive mixture. When the oxygen content lies between about 14,4 and 19,0 percent, the methane remaining at 9 percent, the mixture per se is explosive.

A generalised form of Coward's (1929) diagram is illustrated overleaf in Figure 30.

In Figure 30, point A represents pure air and line AD mixtures of flammable gas with air. Point B represents the lower explosive limit, and point C the upper explosive limit of the combustible gas in air. Point N, commonly termed the "nose point", represents the oxygen and combustible gas concentrations below which no mixtures are explosive. In constructing this diagram, point B and C are initially plotted from the Lower Explosive Limit and Upper Explosive Limit of the mixture.

Line EN is then drawn such, that, if continued, it would pass through Point A. The composition point, X, for the sample is then plotted from its oxygen and combustible gas content.

Three regions in the diagram are of interest. If a sample point lies in the area BCN it is explosive (capable of propagating flame after ignition). Samples lying in region DCNE are not explosive, but would become so if they were diluted with air (potentially explosive) Samples lying in the region ABNEO are not explosive and are not capable of forming explosive mixtures with air dilution.

The diagram also conveys information on the effect of composition changes on explosibility. If a mixture is diluted with air, such as in the opening of a sealed area, its point on the diagram will move along a straight line toward point A (direction a, Figure 30). This is especially important for samples initially in a potentially explosive condition, since air dilution means that they must pass through an explosive state.

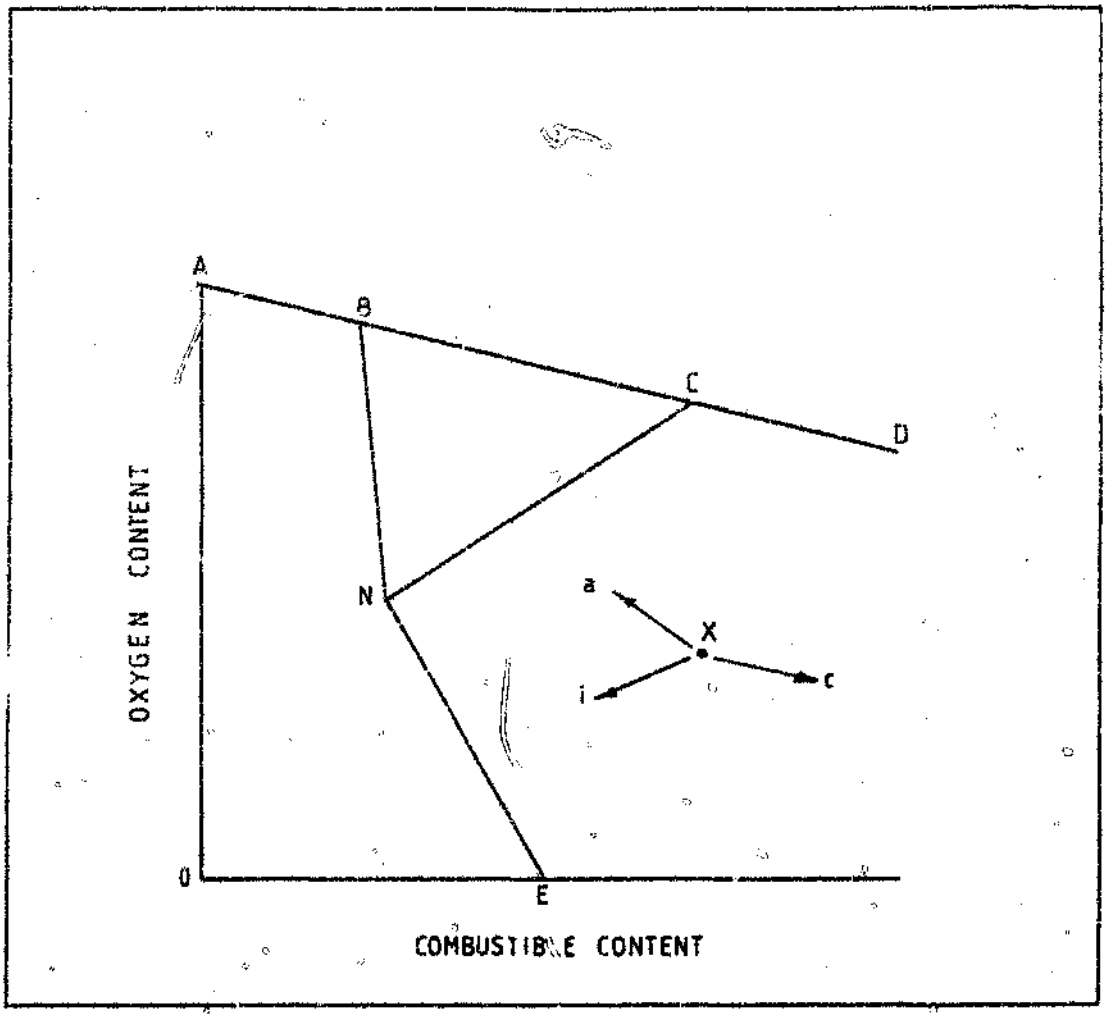


Figure 30

GENERALISED COWARD DIAGRAM

Dilution with an inert gas will cause a sample point to move towards the origin, point O, (direction i). Increasing combustible content of the mixture causes a shift in direction C. An important aspect of this is that increasing combustible content of a mine atmosphere may result from not only the presence of increasing volumes of combustible gas, but also from reducing the air supply to an area by sealing. This under some conditions, could take the atmosphere through an explosive state.

Coward (1929) continued that heated coal produces, by distillation and by partial combustion, methane, hydrogen, carbon monoxide and other inflammable gases and vapours. To represent by curves and tables the explosive properties of all the various possible mixtures of these with air and blackdamp as was done for methane alone would be an endless task.

United States Bureau of Mines (1978) explosibility triangle is based on the conversion of methane, hydrogen and carbon monoxide in the atmosphere into an effective combustible content and the nitrogen and carbon dioxide into an effective non-combustible (inactive) content (see Figure 31). A value, R, which represents the relation between methane and the total amount of combustible gases, is used to calculate the area of the explosibility triangle. The plotted position of a specified gas mixture on the diagram is calculated as follows:

$$Y = \text{Effective combustible gases} = \%\,CH_4 + 1.25\,(\%H_2) + 0.4\,(\%\,Co) \quad \text{(3-1)}$$

$$Z = \text{Effective incombustible gases} = \%\,\text{Excess}\,N_2 + 1.5\,(\%\,Co_2) \quad \text{(3-2)}$$

Where: $$\%\,\text{Excess}\,N_2 = \%\,N_2\ \text{in sample} - \%\,\text{normal}\,N_2 \quad \text{(3-3)}$$

And

$$\%\,\text{Normal}\,N_2 = (79.04/20.93)\,(\%\,O_2\ \text{in sample}) = 3.78\,(\%\,O_2\ \text{in sample}) \quad \text{(3-4)}$$

$$R = \%\,CH_4 / (\%\,CH_4 + \%\,H_2 + \%\,CO) \quad \text{(3-5)}$$

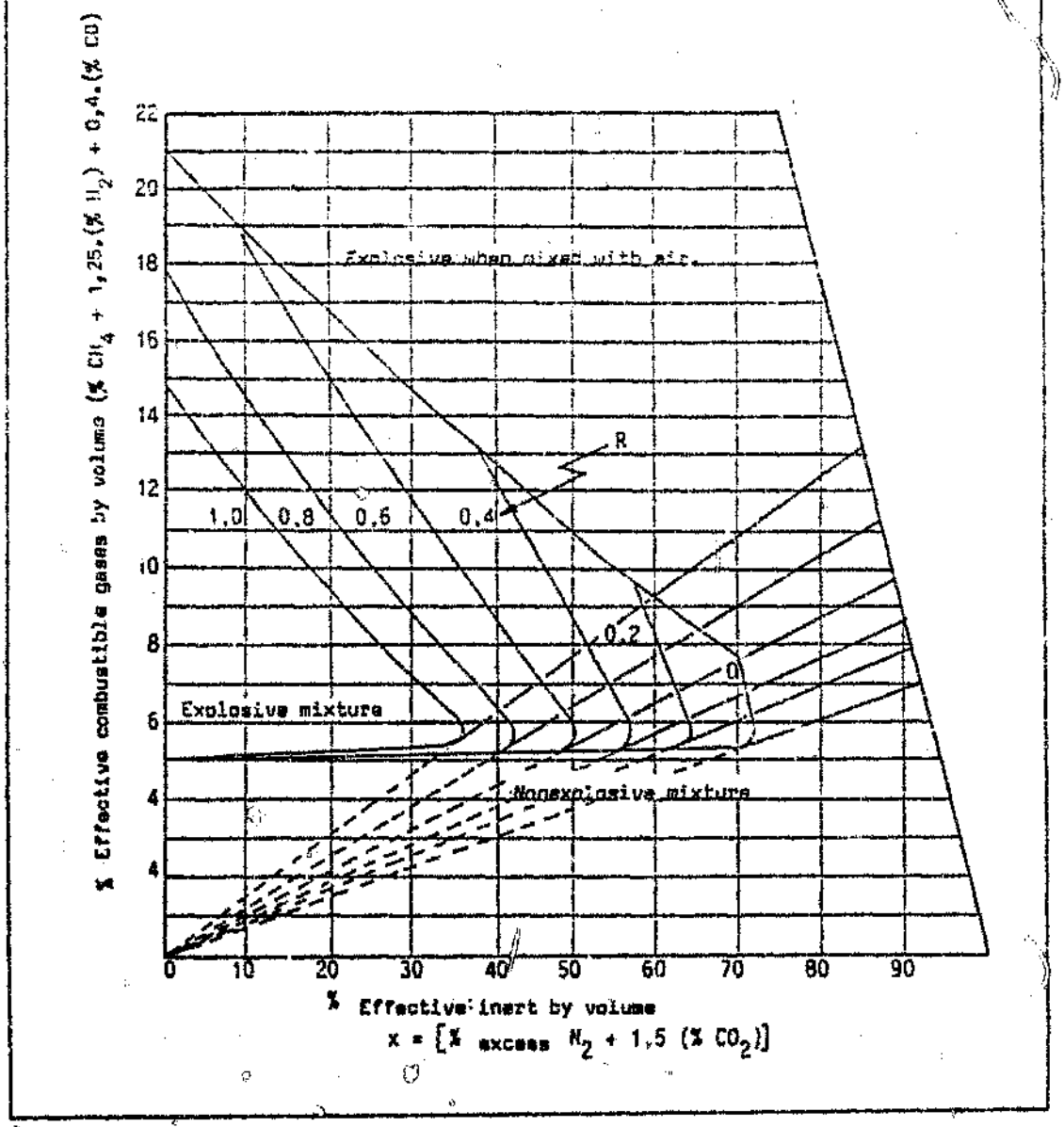


Figure 31

USBM EXPLOSIBILITY TRIANGLE

The Coward triangle (1929) was in general use as recently as 1977/78 in coal mines in South Africa and is still in use on some mines.

It has however been generally superceded by the United States Bureau of Mines (1978) triangle. The advantages claimed for the USBM explosion triangle is that it takes into account hydrogen and/or carbon monoxide which is present in the combustible gases in the y co-ordinate.

Where a heating, glowing or active fire is visible to mine staff (coal, timber, other inflammable material, electric fires) the seriousness can be guaged and the necessary action taken immediately especially when the presence or otherwise of methane gas can be determined using methanometers. The primary aim is to load out or douse the fire and if this fails to seal off the area with the utmost haste ensuring that the atmosphere contains a methane/air mixture which is not explosive.

An example of this situation was the fire which developed in the bottom seam workings at Springfield Colliery in 1952. The bord and pillar workings of the South Rand Colliery, which had ceased production in 1934 were flooded. These 2,5 metre high workings were on the floor of the 20 metre thick coal seam. When the New Springfield Colliery commenced production in 1950, one of the objectives was to dewater the old mine workings via the South Rand vertical shafts and drive declines into the dewatered workings and commence top coaling the old roadways. When the three declines reached the South Rand workings and these were ventilated, a visible spontaneous combustion fire broke out within four days. The area was quickly sealed and the fire doused by pumping water back into the old workings thus successfully avoiding further problems.

Four gas ratios are available to the Mining Engineer for use in determining whether or not a fire is developing in areas which are not easily accessible - for example, the goaf area, old

workings or return airways which have recently been rendered inaccessible - and to determine whether the fire has reached the stage of forming an ignition source.

These ratios are:

i. Graham's Ratio (1926)

This ratio attempts to relate the level of carbon monoxide production to the rate at which oxygen is being absorbed in the oxidation process. The oxygen deficiency value used is not the difference between the oxygen concentration in the sample and the normal oxygen content of fresh air (20,95%). In this context, an oxygen deficient atmosphere is one in which the proportion of oxygen to nitrogen is less than normal (normal being 20,93:79,04 = 0,26:1) and the oxygen deficiency is the oxygen percentage which would need to be added to restore the proportion to normal.

An example of the calculation of this ratio follows:

Sample results

CO - 0,0029 percent (29 ppm)

O_2 - 20,18 percent

N_2 - 78,92 percent

Normal O_2 in fresh air - 20,93 percent

Normal N_2 in fresh air - 79,04 percent

O_2 corresponding to 78,92 percent

$$N_2 = \frac{78,92}{79,04} \times 20,93 = 20,90 \text{ percent}$$

O_2 deficiency = 20,90 - 20,18 = 0,72 percent

$$CO/O_2 \text{ deficiency} = \frac{0,0029}{0,72} \times 100 = 0,4 \text{ percent}$$

The author made extensive use of this ratio when dealing with fires at Springfield Colliery from 1968 to 1973.

ii. Willett's Ratio (1943)

This ratio takes all the gases in the sample other than air (the air-free analysis) and expresses the carbon dioxide as a percentage of these gases.

The formula is:

$$\frac{CO_2}{\text{Excess } N_2 + \text{Total Combustibles} + CO_2} \times 100$$

iii. Young's Ratio

This ratio could be referred to as the carbon dioxide oxygen deficiency ratio and is given by the formula

$$\frac{CO_2}{O_2 \text{ deficiency}} \times 100$$

iv. Morris Ratio No. 7 (1988)

This ratio $\frac{N_2}{CO + CO_2}$

clearly indicates when a fire is active or passive. It is also very sensitive to a change in the state of gases within a sealed area.

The intention in this section of the conclusion is to indicate the use of these ratios in detecting a fire and establishing whether or not it is active. Graham's ratio has been successfully used to determine this trend.

Graham showed that as the temperature increased, so the rate of CO produced, to the percentage of oxygen absorbed, also increased. This is indicated in Table 5.4.

Table 5.4

Temperature	Carbon monoxide production as a percentage of oxygen absorbed
^{0}C	Percent
20	0,7
30	1,0
70	2,0
100	2,8
120	5,0
140	7,0

An interesting comment was made by Dr. Haldane:

"For tracking down incipient goaf fires I believe that Mr. Graham's method will be very valuable - if, indeed, in face of the extraordinary new Regulation which has just been forced on the coal industry, any Manager will care to look for goaf fires till smoke or flames appear. Mr. Graham's method is probably far inferior in delicacy to the sense of smell; but the sense of smell becomes very quickly dulled when one is present in an odoriferous atmosphere, and may be unreliable from other causes, as, for instance, the adsorption, during passage through finely divided coal, of the odoriferous substances."

Hence, as sample analysis indicates, an increasing ratio derived from Graham's formula so the progress towards an active fire (ignition source) is plotted.

Figure 32 shows the results obtained from a developing spontaneous combustion fire which demonstrates the need to erect seals promptly. These results were recorded at Springfield Colliery in 1971.

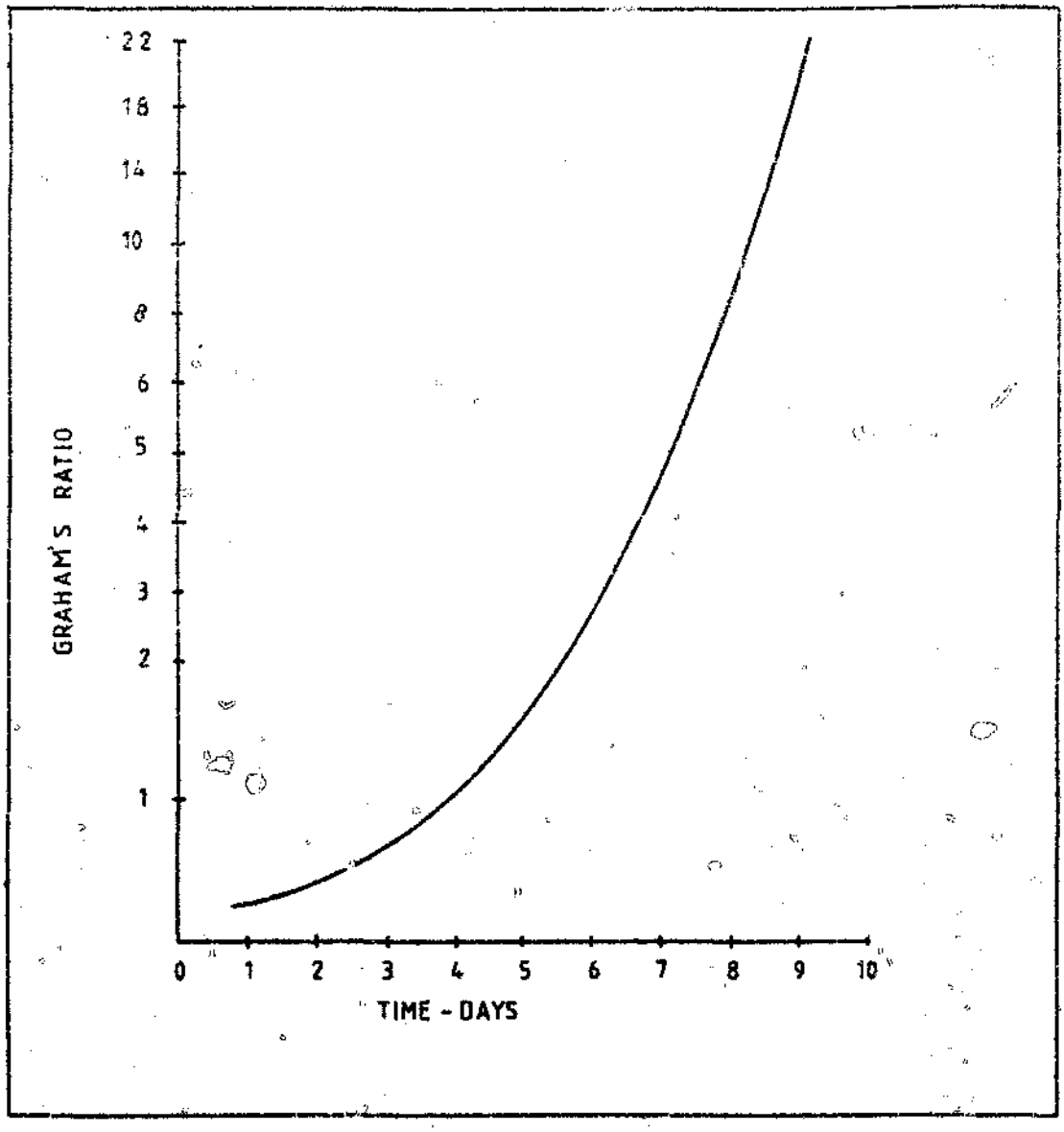


Figure 32

GRAHAM'S RATIO VERSUS TIME

dangers do exist that an explosion may occur behind the seals while the work of creating seals is still in progress. A comprehensive analysis of the three main fire ratios (Graham, Young and Willett) has been made by Morris (1988) and he has derived a new fire ratio which may be used to determine whether a fire behind the seals has become extinct and further when an explosion behind the seals is imminent. This ratio is N2/CO + CO2 and indicates when the fire is active or passive. Morris states that:

"Further, this ratio is very sensitive to a change in the state of the gases within a sealed fire zone and may be used to inform mine personnel that an explosion within the fire seals will occur in time to withdraw workmen from the mine."

5.7.4 Section ventilation appliances

In studying the 69 methane/coal dust explosions which have occurred in South Africa since the turn of the century in almost all cases an overriding feature is the lack of attention to the ventilation appliances in sections. This deficiency finally results in inadequate ventilation delivered to the coal face thereby allowing dangerous accumulations of methane to occur. The incidents dealt with in this chapter highlight:

- Section brattice cloth tied up to allow the easy path of equipment. This results in section ventilation being short-circuited and thus not reaching the working areas.

- Fans installed in positions which allow recirculation.

- Not installing check curtains at auxiliary fans to prevent recirculation.

- Broken ventilation ducting or ducting which has been throttled to divert more air into another face.

Supervision, discipline and proper training are the three ingredients necessary to ensure adequate ventilation to all faces.

Since brattice cloth tends to allow more leakage of air to the return airway than brick stoppings and the object is to prevent air leakage it follows that brick stoppings should be erected as soon as is practical in a section to replace brattice walls. The Northfield disaster in 1943 reference I.M.N./1943 is a good example of the result of a failure to erect permanent stoppings as soon as possible.

All auxiliary fan installations, and start up procedures, should be documented into a standard procedure file which should be used to train all those concerned with these fans and their operations (see Chapter 2).

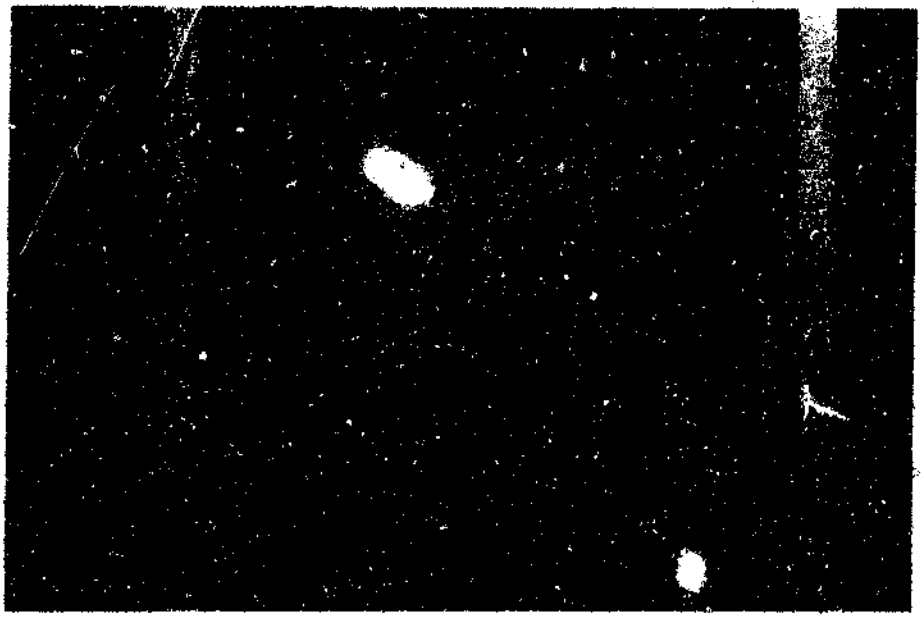
Rolling up brattice curtains to allow the passage of equipment defeats the objective in erecting it in the first instance.

When sealing off a panel in which a fire is known to be present, the two seals in the intake road and return airway road respectively should be equipped with steel doors. These two doors should remain open thereby allowing a passage of air through the section until all seals are complete. Both doors should then be closed simultaneously and workmen removed from the district.

Roadways which have developed through dolerite dykes require close attention to ventilation and gas testing - this aspect has been dealt with in Chapter 2.

5.7.5 Fires

Non-inflammable brattice only should be permitted underground. Photograph 1 shows the test carried out on brattice cloth currently in use underground on fiery coal mines. A new





non-inflammable brattice (Frame Manufacturing Group Q6133) has now been introduced. Photograph 2 indicates the danger of using piping underground which burns. This piping should be removed from the mine.

Adequate fire fighting equipment should be installed in all haulages and sections underground such that in the event of a fire occurring it may be doused immediately. The only effective method of ensuring that these standards apply is:

- to develop standard procedures to prevent and fight fires;
- audit or overview the installations at frequent intervals;
- develop mock fire drills to test the effectiveness of the systems. An example of suggested recommendations is set out below.

It is dangerous to assume that a fire is out. Mining Engineers should make use of all the fire detection ratios available together with sound experience to scientifically determine its state.

MINIMUM REQUIREMENTS FOR FIRE FIGHTING EQUIPMENT

These requirements are the minimum to be used where specified. Nothing shall prevent these requirements being exceeded should the conditions deem it necessay

PRODUCTION SECTIONS

Work Area	Fire Fighting Equipment Type	No Reqd	Size	Position	Other Requirements	Risk	General
1. Section Conveyor							
a. Drive Unit	Dry Powder Extinguisher	2	9 kg	Near switchgear on intake side		Friction Lubrication Electrical	
	Fire Hydrant Standpipe, Hose and fittings	2	75 mm	On intake air side	Hoses and fittings to be stored in locked container		
b. Conveyor	Fire Hydrant Standpipes spaced at 200 m intervals	As reqd		On water column along side of conveyor structure		Friction Rubber	Hoses to be used from drive unit or tail end positions
c. Tail End	Dry Powder Extinguisher	1	9 kg	On intake air side			
	Fire Hydrant Standpipe, Hoses and fittings	As reqd	75 mm	On intake air side near extinguisher	Hoses and fittings to be in locked container	Friction	Hoses to be of sufficient total length to reach extremities of production areas
2. Feeder Breaker	Dry Powder Extinguisher	1	9 kg	On intake air side		Friction Lubrication Electrical	
3. Switchgear	Dry Powder Extinguisher	2	9 kg	On intake air side mounted on trailer or on separate stand		Electrical	
4. Maintenance Bay	Dry Powder Extinguisher 2 Sand Buckets	2	9 kg			Electrical	One unit to be adjacent to oil storage rack
5. Section Store	Dry Powder Extinguisher	1	9 kg	Adjacent to inflammable materials e.g. resin, timber		Inflammable Materials	

All fire fighting equipment positions to be identified with reflective markings.

5.7.6 Supervision

Mining supervisors throughout their daily work are called upon to perform a multifarious set of duties. It is not the author's intention to attempt a set of guidelines for intensity and training of supervision since conditions and methods of working differ considerably. Rather by studying the nature of the various incidents will one be able to form an idea of what degree of supervision is required in a particular set of circumstances.

It is fact, proved by experience, that Mining Engineers, who absent themselves for lengthy periods from their operations, cannot and do not provide adequate technical guidance to their staff - officials are also quick to take advantage of this failing, as are they when supervision is weak and diluted.

Decision-making without technical support such as the theory of the acting Manager at Hlobane (I.M.N. 232/52) that the explosion was due to "atomised gelignite explosives" can be countered by requesting a technical report of a specialised nature as is shown below:

To the Inspector of Mines, Dundee.

"Your minute No. IMNA 245/52 of the 13th instant refers.

From your description of the accidents, I should say that there can be no doubt that in both instances they were due to coal dust explosions.

Mr. Nelson's submissions are rather ingenious, but in my opinion without substance.

High explosives, which category includes the various dynamites, gelignites and detonators, on detonation decompose practically instantaneously and this happens irrespective of whether the explosive is confined or is out in the open.

The explosion of these explosives is a phenomena which is dependent upon the transmission of shock. When, for instance, a quantity of 60% ammon gelignite is initiated by means of a detonator, fire from the safety fuse or the fusehead in the case of electric detonators, acting upon the top layer of powder in the detonator cause it to undergo a rapid chemical transformation which produces hot gas, and the transformation is so rapid that the advancing front of the mass of hot gas amounts to a wave of pressure capable of initiating, by its shock, the explosion for the next portion of powder. This explodes to furnish additional shock which explodes the next adjacent portion of powder and so on, the explosion advancing through the mass with incredible quickness, the velocity of the shock increasing until the maximum velocity for the particular detonator is reached, which for a standard No. 6 detonator, is of the order of 12 000 feet per second (4176 metres per second). The shock wave from the detonator then passes into the ammon gelignite which it induces to explode in the same way.

On occasions an explosive may become so insensitive, for instance through absorption of excessive moisture, etc. that it either does not detonate at all or detonates incompletely and only partway into its mass, leaving the rest largely intact.

Mr. Nelson's theory would seem to be that the first charges detonated only partially and, in the process, pulverised the unexploded portions to such an extent that they formed a cloud which was ignited by the flame from a subsequent charge. This is inconceivable. An explosive does not require oxygen from the air for combustion and if the explosion wave in a charge was so powerful as to pulverise the back cartridges, the gases accompanying the wave would have been still at such a heat that the pulverised material could not have failed to explode there and then or, at the very least, have caught fire immediately and burned out long before reaching the bend around which the boys sheltered.

It has been confirmed in practice that when a mass of explosive detonates incompletely the explosion wave starts at a low strength, increases up to a maximum and then decreases until it dies out, leaving the undetonated portion more or less intact.

Signed - Chief Inspector of Explosives.

5.7.7 Spontaneous combustion in the goaf

Spontaneous combustion in the goaf presents an ever present danger.

- In pillar extraction using cyclic mining machinery and the closed lift method fenders and snooks which cannot be entirely removed should be destroyed by blasting thereby preventing goaf hang-ups and subsequent crushing of snooks in the goaf.

- Pillar extraction should not be practiced adjacent to and into burnt coal zones which flank dolerite dykes. A sufficiently wide barrier pillar should be left on either side of the disturbance so as to prevent crush on the burnt coal zone and subsequent heatings occurring.

- All goaf bleeder roads should be monitored on a frequent basis for carbon monoxide build up.

- Stonedust should be blown into the goaf through the check curtains at the goaf line by mechanical means. This will render any coal dust in the goaf incombustible.

5.7.8 Systematic Rehabilitation of Airways

With the passage of time intake and return airways deteriorate (especially in thick coal seams). Roof falls, floor heave and pillarside scaling result in the:

- lowering of the safety factor below the planned safety factor levels,

- increase in the mine resistance leading to lower ventilation quantities circulating throughout the mine,

- possibility of spontaneous combustion occurring in main haulages and returns,

- neglect of the mandatory examination of return airways since these eventually become impassable, due to accumulations of broken coal.

As a result of these circumstances the dangers of spontaneous combustion and then mine fires leading to explosions become a real danger.

5.8 CONCLUSION

This chapter has been devoted to methane and coal dust explosions which have their origins in and at the edge of the goaf in pillar extraction sections, and also in bord and pillar sections where fires have resulted in an ignition source.

Spontaneous combustion in the goaf and fires on the goaf edge as a result of blasting operations, faulty electrical equipment and contraband coupled with poor supervision, inadequate ventilation and the circumventing of standard procedures and the Mines and Works Regulations have been analysed. Fires in development sections have led eventually to major coal dust explosions which have ravished the whole mine and partially destroyed the shaft systems. In other instances, the prompt and efficient attention to fires has resulted in the minimum disruption to operations and the restoration of the environment to a safe condition.

Methods to be adopted to prevent in the first instance and secondly deal with these incidents when they occur are discussed with particular reference to the draining of methane from the goaf cavity, and preventing recirculation of the ventilation at auxiliary fans and adopting training and operational procedures to improve the safety of mines. It is incumbent on Mining Engineers to increase supervision when major work is necessary to load out coal from old workings (falls of roof and spalling of ribsides) and repair the roof and pillars. Operational staff tend to ignore this work in favour of production. Coal dust which has gathered in roadways throughout the mine due to one cause or another should be systematically swept up and sent out of the mine.

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6.1 INTRODUCTION

The Hlobane Colliery explosion on 12th September 1983, which was initiated by an electric spark in a silicon controlled rectifier panel of a battery operated scooptram, highlighted the danger which exists in the use of electricity underground. The trend in coal mining, particularly over the last 4 or 5 decades, has seen the replacement, on an increasing scale, of compressed air and steam as a motive power by the almost exclusive use of electricity. Supply voltages used have also increased from 500 volts to 6,6 kV.

The flexibility provided by the use of electricity has led to the continued increase in both physical size and power of coal winning equipment, together with the introduction of portable auxiliary ventilation fans of 37 kV each. Larger substations and distribution equipment of greater capacity have been a logical step in the increased usage of electricity underground.

An examination of the causes of explosions initiated by an electrical source reveals the use of unapproved (known to be unsafe) apparatus, poorly installed equipment, in some instances mechanically damaged equipment (falls of roof) and generally poorly maintained equipment especially the mobile flameproof equipment. In one instance, the author is aware of an explosion resulting in a multiple fatality as a result of the use of "homemade" flameproof control boxes on mobile section equipment.

Higher currents through conductors and connections which result in overloading, short circuits and earth faults cause overheating which results in insulation failures and failures in connections.

Overheating can also occur due to inadequate ventilation and dirt

any joint, varies according to the volume of the empty casing and the nature of the apparatus but is usually 25 mm. However, widths of 18 mm and even 12 mm are sufficient for certain small enclosures such as bells, telephones and lighting fittings.

The methane explosion which occurred in Section 16 at Springfield Colliery in May 1957 and which resulted in 8 fatalities and 11 injuries was caused by the use of a coalcutter control panel fabricated in the mine workshops which did not conform to flameproof standards. The switch was turned off while the operator reorganised his cables. The control box cooled down and methane was drawn into the controller through the gap in the flameproof faces. While starting up the coalcutter a flash or arc occurred in the controller igniting the methane mixture. The faulty flameproof design allowed the flame to pass through the joint faces and diametrical clearances, thereby igniting methane in the casing heading.

i) Fixed Equipment

The most likely causes for incendive sparking from the various type of equipment are listed below:

a. Switchgear and Control Gear

This switchgear is used to isolate various circuits on the underground circuit and is non-flameproof. Initially air-break switches which permitted long and persistent arcs were tolerated, but when it was established that oil immersion of the breaking contacts reduced the arc oil circuit breakers became normal practice for a number of years. Oil is a potential fire hazard and current practice is to use oil-free switchgear.

Physical damage to this type of equipment could cause a possible short circuit across the live conductors. With the units only being semi-dust proof there is a possibility of coal dust entering the arc suppression unit again causing excessive sparking, possibly an ignition.

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CHAPTER 6

IGNITIONS OF FIREDAMP BY ELECTRIC SPARKING FROM FAULTY ELECTRICAL EQUIPMENT

6.1 INTRODUCTION

6.2 STATISTICS

6.3 THE CAUSES OF ELECTRIC FAULTS LEADING TO OVERHEATING, FLASHES, INCENDIVE SPARKS

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(c) Motors
(d) Lights
(e) Batteries
(f) Signalling Wires
(g) Control cables
(h) Lighting cables
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(j) Instrumentation and intrinsically safe equipment

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6.4.2 Duration

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6.5 INCIDENTS AND PHENOMENON

6.6 DETAILS OF INCIDENTS

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6.7 SUMMARY OF THE PRACTICES REVIEWED ABOVE

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6.9 CONCLUSION

6.10 REFERENCES

on electrical equipment preventing adequate heat dissipation from electrical equipment. This leads to insulation failure.

High surface temperatures on motors and transformers due to overloads and faults and on light fittings due to excessively large globes can ignite methane and lead to methane explosions.

Flashes and sparks occur when conductors mechanically fail while they are carrying current and when insulation failure results in short circuits between live conductors and between live conductors and earth.

Electrostatic discharges are not produced by faulty electrical equipment and so they have beeen grouped with lightning, stray currents, light alloys and friction sparks.

The requirements for the prevention of methane and coal dust explosions as a result of electric sparking from faulty electrical equipment is discussed in this Chapter.

6.2 STATISTICS

An analysis of the sources which contributed to explosions between 1970 and 1986 is given in Chapter 2 (Table 2.5). Electrical faults caused 9% of the explosions and resulted in 68 fatalities and 16 persons injured.

Of the 123 explosions which have been recorded in the Natal Coalfields between 1891 and 1984, 8 have been as a result of electrical sparks from faulty electrical equipment. The first recorded incident of this nature was at Utrecht Colliery in 1941. In the Transvaal and Orange Free State, 4 electrically related explosions have been recorded up to 1988.

Figure 1 shows the increase in kW of electric motors installed at British Coal Mines from 1912 to 1954. By 1958 the installed power had risen to nearly 3500 MW.

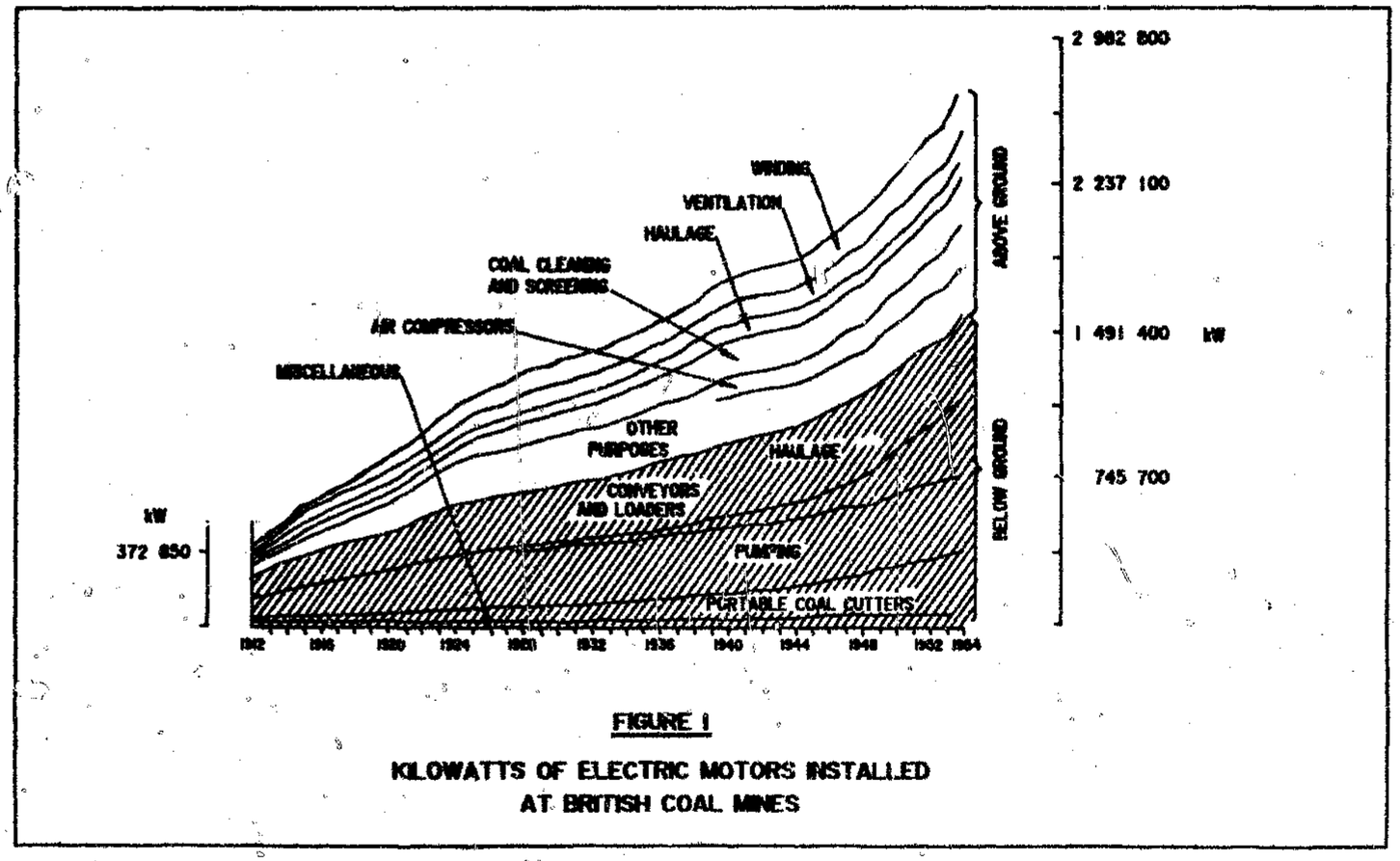


FIGURE 1

KILOWATTS OF ELECTRIC MOTORS INSTALLED
AT BRITISH COAL MINES

6.3 THE CAUSES OF ELECTRIC FAULTS LEADING TO OVERHEATING, FLASHES AND INCENDIVE SPARKS

6.3.1 Outside Hazardous Areas

A hazardous area is defined as an area where there may be risk of igniting gas, [illegible] dust or other explosive material. It includes any area within 180 metres of a working face, any return airway or any other area declared hazardous by the Manager. Only flameproof equipment is used in these areas.

The electrical equipment used in non-hazardous areas is generally non-flameproof i.e. normal quality industrial equipment. A flameproof enclosure is defined as one that will withstand, without injury, any explosion of the prescribed inflammable gas that may occur within it under practical conditions of operation within the rating of the apparatus (and recognised overloads, if any, associated therewith), and will prevent the transmission of flame such as will ignite the prescribed inflammable gas which may be present in the surrounding atmosphere.

The Regulations (1956) state: "in any part of a mine below ground in which inflammable gas, although not normally present, is likely to occur in a quantity sufficient to indicate danger, no electrical apparatus shall be used if there is in the normal working thereof a risk of incendive sparking therefrom."

Flameproof apparatus is categorised into Groups. Group 1, in the United Kingdom, which includes equipment for coal mines where Regulation 22 (1) applies the gaps between joint surfaces and diametrical clearances for operating rods and spindles are restricted to a maximum of 0,15 mm. The minimum breadth of all joints, and the length of the flameproof path through or across

b. Transformers

Usually these are oil filled and if any fault develops within, this could ignite the oil under the correct conditions. Loose connections which could be caused due to vibration would overheat. If tap changing facilities were operated whilst the transformer was on load this would cause arcing. Arcing in oil creates hydrogen and acetylene - both of which are more explosive than methane.

c. Motors

Badly lubricated bearings would cause heating and if left undetected could lead to the rotor coming into contact with the stator and the stator windings, causing a short circuit and hence sparking. Physical damage to the connection box could lead to a short circuit across the terminals.

d. Lights

The use of unprotected lamps would result at some time in a globe being broken and exposing live connections. If a metallic object then came into contact with these the result would be sparking. A fall of roof onto the supply cables would bring a whole length of lights down with the same effect.

e. Batteries

Where traction batteries are used with open terminals any metallic object coming into contact with a positive and negative terminal would cause a spark. Great care must also be taken in charging stations to protect the charging cables from physical damage.

f. Signalling Wires

Where any signalling wires are used that are not supplied from an intrinsically safe source, even though a low voltage supply may be in use, the chances of incendive sparks are highly probable.

g. Control Cables

The most likely occurrence in this case would be from physical damage. Outside the hazardous area, these would normally be collectively screened cables and the chance of a short circuit would be high.

h. Lighting Cables

As with control cables, the most likely occurrence in this case would be from physical damage. Outside the hazardous area, these would normally be collectively screened cables and the chance of a short circuit would be high.

i. Power Cables

As above, but with the added problem that overloading is always a problem. When overloading occurs any termination which has not been done correctly is a very high stress point and will overheat and possibly burn off.

j. Instrumentation and Intrinsically Safe Equipment

When referring to instrumentation, the most likely place to have a problem would be on a current or voltage transformer. When the secondary side of a current transformer becomes open circuit this causes it to burn out and therefore high temperatures are present.

The certification of intrinsically safe apparatus means that whilst maintained in its original condition, this equipment should not be able to cause an ignition. The only problem is that the components rendering the equipment intrinsically safe are not easily tested or seen, and failure of these components does not cause any malfunction of the equipment, therefore there is always a slight possibility of an ignition due to component failure.

enclosure is not required. The decision as to whether a particular device is intrinsically safe rests mainly on practical tests which are described below, but research has been undertaken to try to discover what characteristics of a spark determine whether or not it will ignite gas.

The definition of Intrinsic Safety and a description of the relevant test are given in B.S. 1259 (8) and in the Ministry of Power Testing memorandum No. 10 (9). The tests make use of various forms of "Breakflash Apparatus" in which a switch can be mechanically operated to give repeated and controlled opening and closing, the contacts of the switch being enclosed in a chamber filled with the most easily ignitable gas mixture. Many forms of breakflash apparatus have been devised, but they all have these basic features.

If any part of the equipment under test is liable to produce a spark, either in the course of its normal operation or in the event of any kind of fault or accident, then temporary connections are made from the point of the equipment where a spark might appear to the contacts of the breakflash apparatus. This arrangement simulates the condition in which the equipment might produce sparks in an explosive atmosphere and allows an assessment to be made of the margin of safety against ignition.

Some devices which are not normally intrinsically safe can be made so by the addition of an electrical "safety device". A common procedure is to add one or more metal rectifiers at appropriate points in the circuit. The use of such rectifiers in suitable cases can often make an otherwise dangerous device intrinsically safe by providing a safe discharge path for energy, especially energy stored in inductance, which would otherwise be dissipated in an arc or spark. Some circuits using transistors also lend themselves to this type of safety arrangement.

Another common safety device is a copper sleeve or a short-circuited winding which is used in addition to the normal

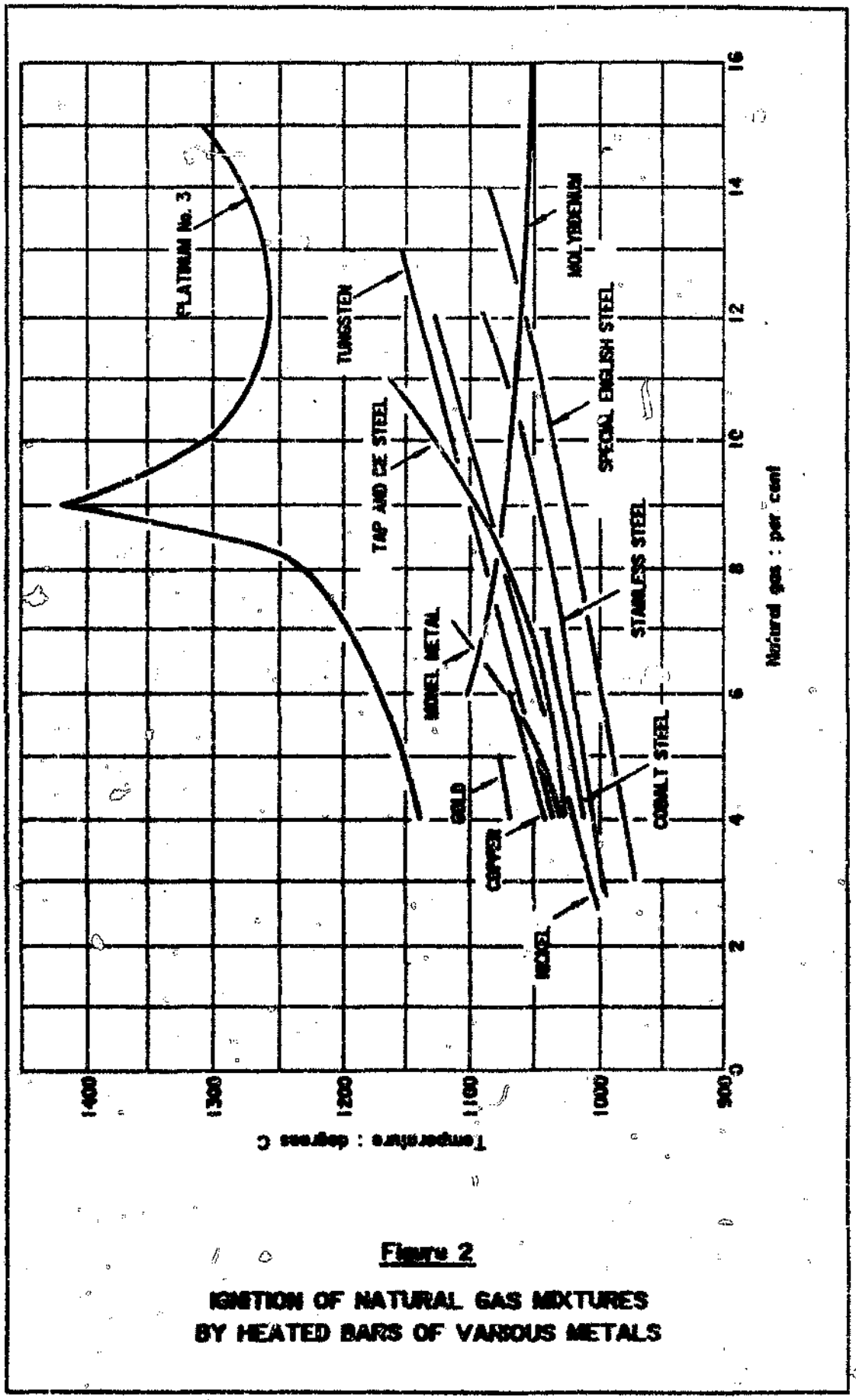


Figure 2

IGNITION OF NATURAL GAS MIXTURES BY HEATED BARS OF VARIOUS METALS

winding of an inductive device and has the effect of reducing the rate at which stored energy is dissipated. It is essential that safety devices should be effective and should remain effective during the life of the equipment, and the normal procedure for intrinsic safety certification appraisal includes an examination to ensure that the safety devices, in common with all other parts of the apparatus, are soundly constructed and properly fitted.

6.5 INCIDENTS AND PHENOMENON

As far as can be determined, the following 13 incidents represent a complete list of methane ignitions in South African coal mines which were caused by sparking of electrical apparatus:

a. Utrecht Colliery. 1941. Ignition of methane due to a damaged 500 volt section power cable on the haulage system. 15 killed.

b. Cambrian Colliery. 1948. A coalcutter severed the trailing cable while flitting out of a stall. Nil injuries.

c. Utrecht Colliery. 1949. A coalcutter damaged the cable of an electric drill. Nil injuries.

d. Cambrian Colliery. 1956. Firedamp was ignited shortly after a booster fan was switched on in a stooping section. 2 killed.

e. Schoongezicht Colliery. 1956. Methane ignition caused by faulty electric auxiliary fan. 8 killed.

f. Springfield Colliery. May 1957. An electric spark in a non-flameproof coalcutter control panel ignited methane causing an explosion. 8 killed, 11 injured.

g. Springfield Colliery. 1962. A faulty booster fan in a section caused a methane explosion. 1 killed.

h. Indumeni Colliery. 1967. The main fan had been stopped for repair over the greater part of a weekend and firedamp accumulated in the workings as a result. The electrical

the cover he restored power and manipulated the switch which ignited the firedamp. The Electrician and his two Black helpers were killed. Coal dust did not contribute to the explosion. 3 killed.

k. Newcastle-Platberg Colliery. 1979. Scooptram cable pulled out. 1 injury.

l. Hlobane No. 1 Colliery. 12th September 1983. A methane and coal dust explosion was initiated by a non-flameproof scooptram. 68 killed.

m. Ermelo Mines Services (Pty) Limited. 25th February 1984. A 6,6 Kv cable exploded and ignited methane which resulted in a methane explosion. Nil injuries.

ii) Mobile Equipment

To generalise, it should be stated that all the possibilities for the existence of an ignition source mentioned for fixed apparatus apply to mobile equipment however, due to the mobile nature of the equipment, the vibration inherent in the machines and the danger posed by rotating parts, the abovementioned dangers become more prevalent.

iii) Portable Equipment

a. Hand Tools

Most hand tools such as drills are not flameproof and the danger, as with all portable equipment, is that the tools are easily taken into a section where methane is most likely to be encountered. The methane explosion in the incline shaft at the Coalbrook Collieries in 1961 (see Chapter 7) which killed 7 men was the result of the use of portable equipment in an explosive atmosphere. Arcing may result due to dirty brushgear or faulty contacts.

b. Pumps, Fans and Instruments

The greatest danger is that non flameproof pump motors and switchgear is inadvertently used in hazardous areas. Portable fans have been the cause of many explosions; flameproof fan motors have been repaired on surface and returned underground and declared flameproof when in fact they are not flameproof (see Cambrian Colliery explosion in 1956 - Chapter 5).

6.4 THE NATURE OF ELECTRIC SPARKS

6.4.1 Intensity

Intensity is defined as a quality or amount of some quality such as force, brightness or magnetic field.

There are two main ways in which electrical energy can be released in such a manner that it might ignite inflammable gas. A wire or other conductor carrying an electric current may become heated, and if the current is strong enough the wire may become so hot that it can ignite gas directly. Secondly, an electric spark capable of igniting gas may pass between the ends of conductors which are close together but not actually touching; for example, if a wire carrying a current is broken, then immediately after the break occurs the broken ends are close together and a spark may pass between these ends. In addition, incendive sparks can be caused by electrostatic discharges.

A heated conductor is used as the filament of the electric cap lamp bulb, and sparks are produced during the normal working of many kinds of electrical equipment, including electric bells, relays, telephone generators, electric motors and switches which vary from a simple push button to a main circuit breaker.

Sparks can also be produced as a result of accidental short circuit or open circuit and electrical faults can cause overheating. It might therefore be thought that there is considerable risk in using any of these devices in parts of a coal mine where the presence of inflammable gas is a possibility, but fortunately, methods have been developed by which equipment

of these types can be designed, manufactured and used so that the risk of ignition is infinitesimal.

The minimum quantity of electricity which can cause an ignition is, by the standards of the power Engineer, very small; it is in fact less than 0,5 millijoule (1 joule = 1 watt-second). This means, in principle at least, that a comparatively slight fault in electrical equipment could lead to the release of energy in sufficient quantity to ignite methane. The continued growth in the amount of electrical equipment and in the size of individual units means that there is a tendency for the possibility of ignition of gas to increase and the growing complexity of many electrical installations means that, in addition, there are more and more ways in which a fault could occur.

Kinghorn and Evans (1989) state that "very low energy sparks are capable of igniting methane. Except for sparks from low energy intrinsically safe circuits electrical sparks are incendive". The intensity of electric sparks which do not ignite inflammable mixtures of methane and air are dealt with in paragraph 6.4.4 (intrinsically safe equipment).

6.4.2 Duration

A spark is not by any means a simple phenomenon; it may be any one of three quite distinct kinds of electrical discharge, or it may comprise combinations of two or all three of these kinds of discharges. A common form of discharge is the electric arc, which is characterised by a fairly steady flow of current, perhaps a few amperes in the case of mining equipment, with a voltage drop of perhaps 20 or 25 volts between the ends of the wires forming the electrodes between which the arc appears. Another kind of discharge is the flow discharge in which the voltage between the electrodes is often in the region of 300 or 400 volts and the third kind of discharge is the oscillatory discharge in which the discharge is repeatedly established and

extinguished. All three types of discharge are capable of igniting gas. We have records of sparks which, although they each lasted for only one ten thousandth part of a second, exhibited all three kinds of discharge and changed character four or five times during this period.

It is not yet known precisely what characteristics of a spark determine whether or not it will ignite methane/air mixtures. It is known that ignition is very unlikely if the electrical energy dissipated in the discharge is less than a third of a millijoule, for example, less than a third of an ampere at twenty volts for a twenty-thousandth part of a second. It is also known that in some circuits a discharge which dissipates considerably greater energy than this may be quite safe. On the basis of our present knowledge, it is possible to examine the circuit diagram of a piece of electrical equipment and often to be able to make a forecast of the character of the sparks which might occur, for example, when switch contacts open or close. In some cases it is possible to estimate whether these sparks will be capable of igniting gas, but this cannot be done with confidence in all cases and every piece of equipment which is received for examination by the recognised testing authority is therefore subjected to practical tests.

Kinghorn and Evans (1989) further state, as a general rule, that cable faults are cleared quickly and the flame retardant sheaths ensure that burning sheaths extinguish the moment the fault is cleared. A cause for concern is short circuits in long cables with small conductors in low fault level systems. Under these circumstances short circuit currents are of the same order of magnitude as full load currents and short circuits may not be cleared by the protection. A sustained short circuit will cause the cable to burn for as long as the short circuit is allowed to continue.

6.4.3 Temperature

Ramsey and Hartwell (1965) in dealing with the temperature at which various mixtures of methane/air ignite and the lag of ignition, state that the temperatures of an electric arc can be 2500°C or higher.

Various people have attempted to find the minimum temperature at which methane may be ignited. This temperature varies considerably with the conditions in which the determination is made, but iguition at hot surfaces can be fairly readily obtained at as low as 600°C to 700°C and ignition by adiabatic compression may take place at compressions producing temperatures no higher than 450°C. However, when a methane/air mixture is to be ignited by a hot surface freely exposed to convection ventilation, the minimum ignition temperature is in excess of 1 000°C. Figure 2 shows some typical results.

From the above we may conclude that in order to ignite methane/air mixtures within the inflammable ranges (5,3% - 14,8% methane) temperatures, from hot surfaces such as copper, as low as 1050°C are necessary.

6.4.4 Intrinsically Safe Equipment

Almost any piece of electrical equipment is able to produce sparks, either in the course of normal operation or in abnormal conditions resulting from a fault, misuse or damage. The danger that such a spark might ignite inflammable gas can be removed by placing the equipment in flameproof enclosures, but these are heavy, bulky and expensive and it would be very inconvenient to provide a flameproof box for every electrical device on a modern face. It has been found, however, that the sparks produced by some small devices are incapable of igniting gas, and this has led to the establishment of a class of electrical equipment which has the property of "intrinsic safety" for which a flameproof

power cables serving that part of the mine had not been isolated. It would appear that either when the power was switched on when the fan repairs had been completed, or as the result of a lightning flash during a severe thunderstorm that took place while the fan repairs were in progress, open sparking occurred in the gas filled area. There was no one in the workings at the time and there were no casualties. Three sections were extensively damaged. Local coking of dust suggested that coal dust contributed to the explosion to some extent but the mine was well stonedusted and this undoubtedly limited flame propagation. Nil injuries.

i. Indumeni Colliery. 1967. The explosion occurred in the breakaway workings off the bottom of a newly sunk circular shaft some 210 metres deep. The workings were force ventilated by two fans on the surface supplying air through ventilation ducting. One of the fans was stopped for repairs. Shortly after it had been restarted a violent explosion occurred which toppled the headgear. The electric cable supplying power to switchgear at the shaft bottom had not been isolated. It would appear that either a "plug" of firedamp that had built up during the fan stoppage was moved when the fan was restarted and encountered an open spark or an electrostatic spark of sufficient intensity to ignite the gas was generated by the flapping of a plastic tube extension, to the ventilation ducting. There was no one in the workings and there were no casualties. It is unlikely that coal dust played any part. Nil injuries.

j. Indumeni Colliery. 1967. An electric pump was positioned in a main intake airway just outbye of a dip created in negotiating a fault which was giving off water and methane. The switch at the pump broke down and put the pump out of commission. The water rose in the dip and sealed off the flow of ventilating air over the pump gear. Firedamp accumulated. An Electrician removed the cover of the flameproof switch and repaired the switch. Before replacing

6.6 DETAILS OF INCIDENTS

In order to illustrate the ease with which electrical sparking can lead to an explosion, three particular incidents will be reviewed in detail:

6.6.1 Ermelo Mines Services (Pty) Limited

This mine, situated in the Ermelo district, is a particularly gassy pit, as is shown in the evidence of this explosion.

Details of the Occurrence

Methane explosions occurred in two sections simultaneously when the current was switched on from approximately 2 km away and a 6,6 kV joint box blew up.

Nobody was killed or injured by the explosion while damage to equipment was minimal.

Circumstances leading to the Explosion

Refer Figure 3 and 4.

The main surface fan had been switched off for approximately 6 hours for repairs.

One hour after the fan had been switched on again, the electricity supply at the distribution substation was switched on near the shaft. Due to a defect in the switchgear in the intermediate substation, the current flowed through to the section. En route a 6,6 kV joint box exploded and ignited a tongue of methane which resulted in the explosions.

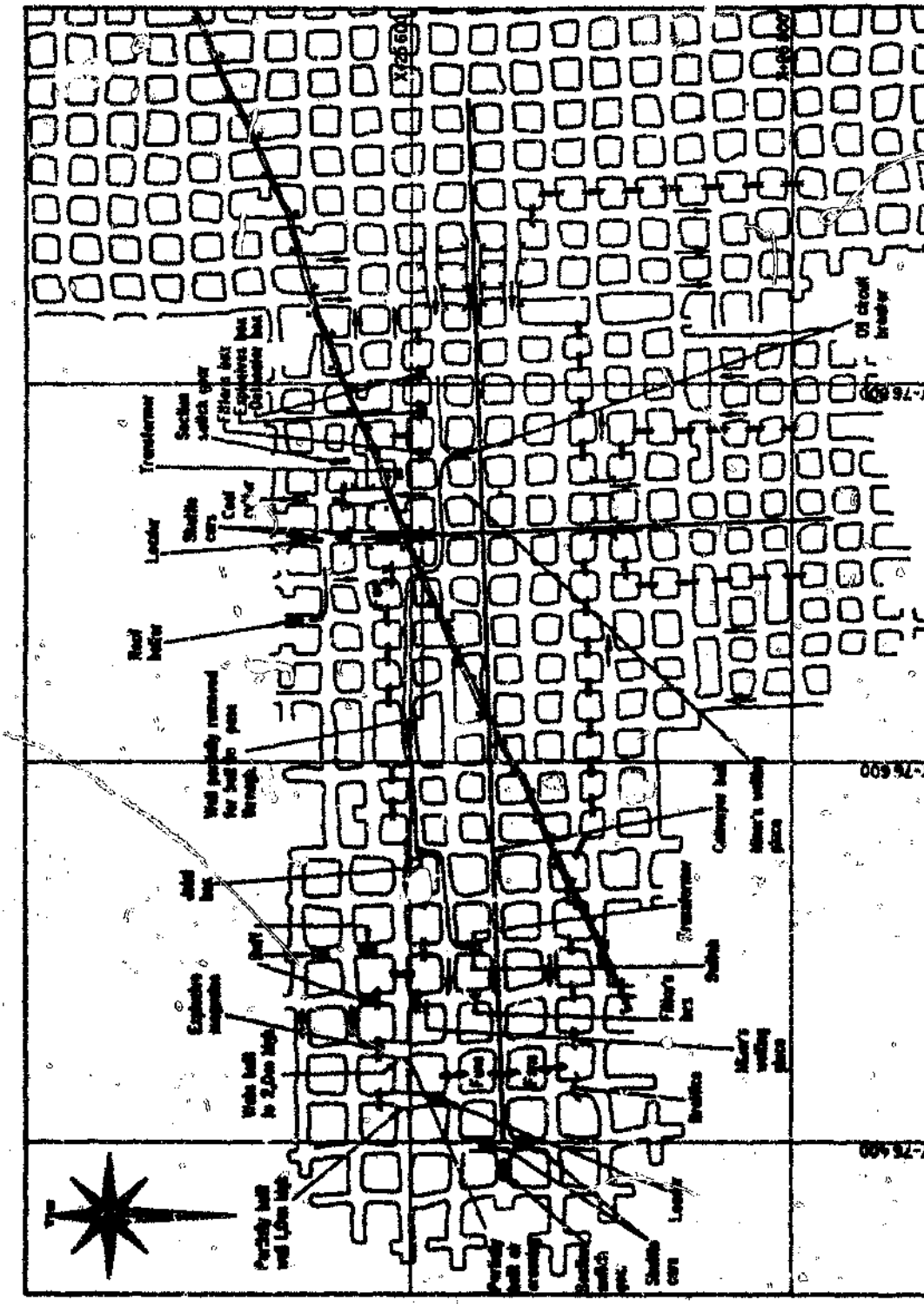


Figure 3

ERMELO MINES SERVICES (PTY) LTD.
PLAN SHOWING SECTION 7 WEST MAIN
BEFORE METHANE IGNITION
ON 25/02/84 AT 19:00
Not to scale

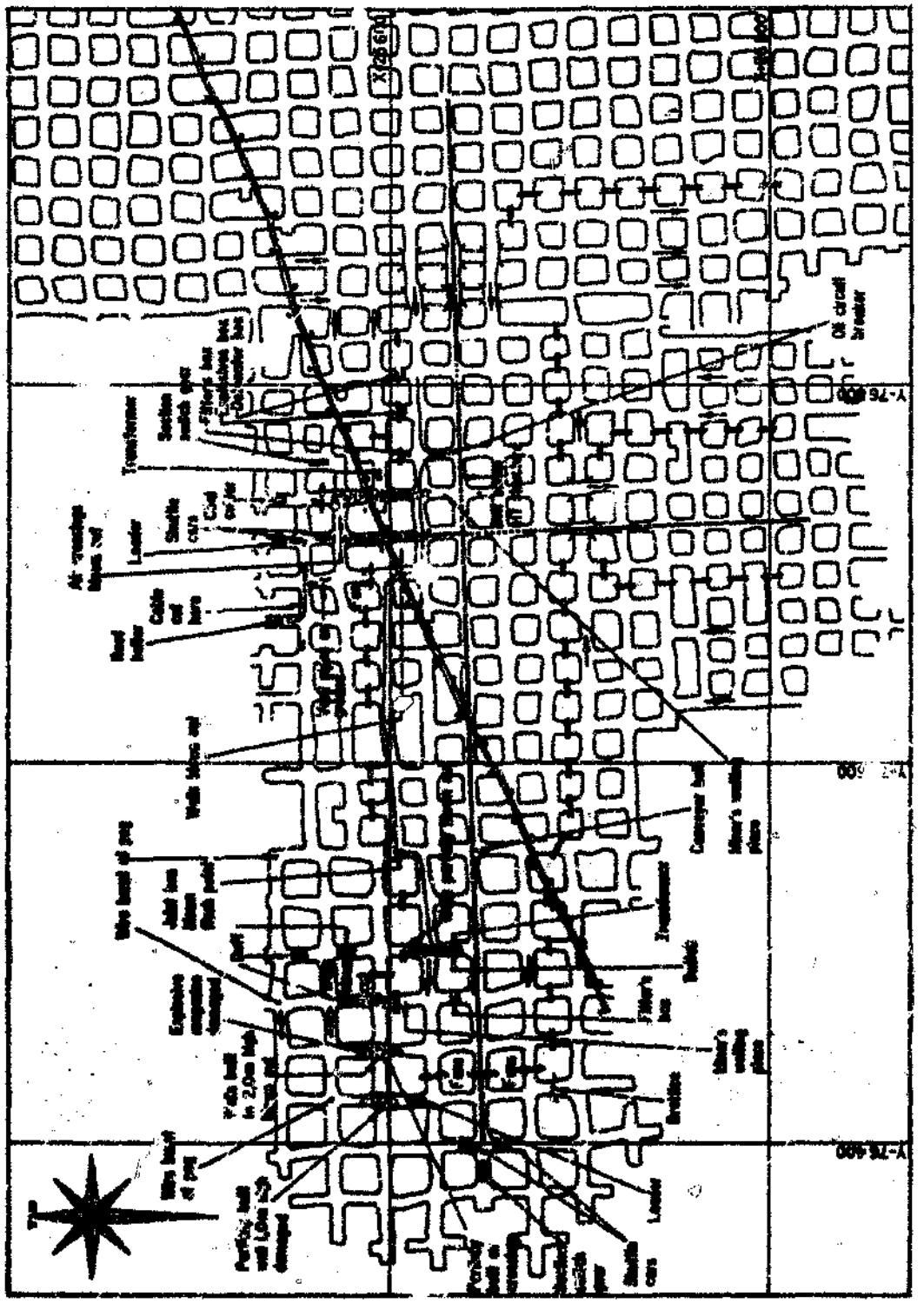


Figure 4

ERMELO MINES SERVICES (PTY) LTD.
PLAN SHOWING SECTION 7 WEST MAIN
AFTER METHANE IGNITION
ON 25/02/84 AT 19:00
Not to scale

Inspection In Loco

The in loco inspection was carried out within hours of the explosion.

Walls, cross-overs and other minor structures were damaged in two separate areas, viz. Section 10 and the right hand side of Section 3, while the conveyor belt was thrown off the structure in and opposite Section 10. This indicated that two separate explosions occurred.

The joint box is indicated on Figure 3 and 4 and comprised two halves, one being a "Proof" and the other a "B I C C" type. The "B I C C" section exploded. This joint box was situated against the roof on the fresh air side of a brick stopping. The stopping had openings of approximately 2 cm between it and the roof.

All the electrical units in the sections were inspected by the Inspector of Machinery and, except for a missing bolt on one of the shuttlecars, they were all found in a flameproof condition. There were no indications that the explosion resulted from the non-flameproof condition of the shuttlecar. During the subsequent tests described below, no gas was found at any of the electrical switchboards, transformers and oil circuit breakers near the sections.

Subsequent Tests

During a test carried out subsequently, the main fan was switched off for 3 hours and tests for gas were carried out. The results are indicated on Figure 5. Considerable methane build-ups were encountered in Section 10 and the right hand side of Section 3 in spite of the fact that there was still 7 m^3/s air flowing through these sections.

The fan was then switched on again and within one hour these accumulations were dispersed and the highest concentration of

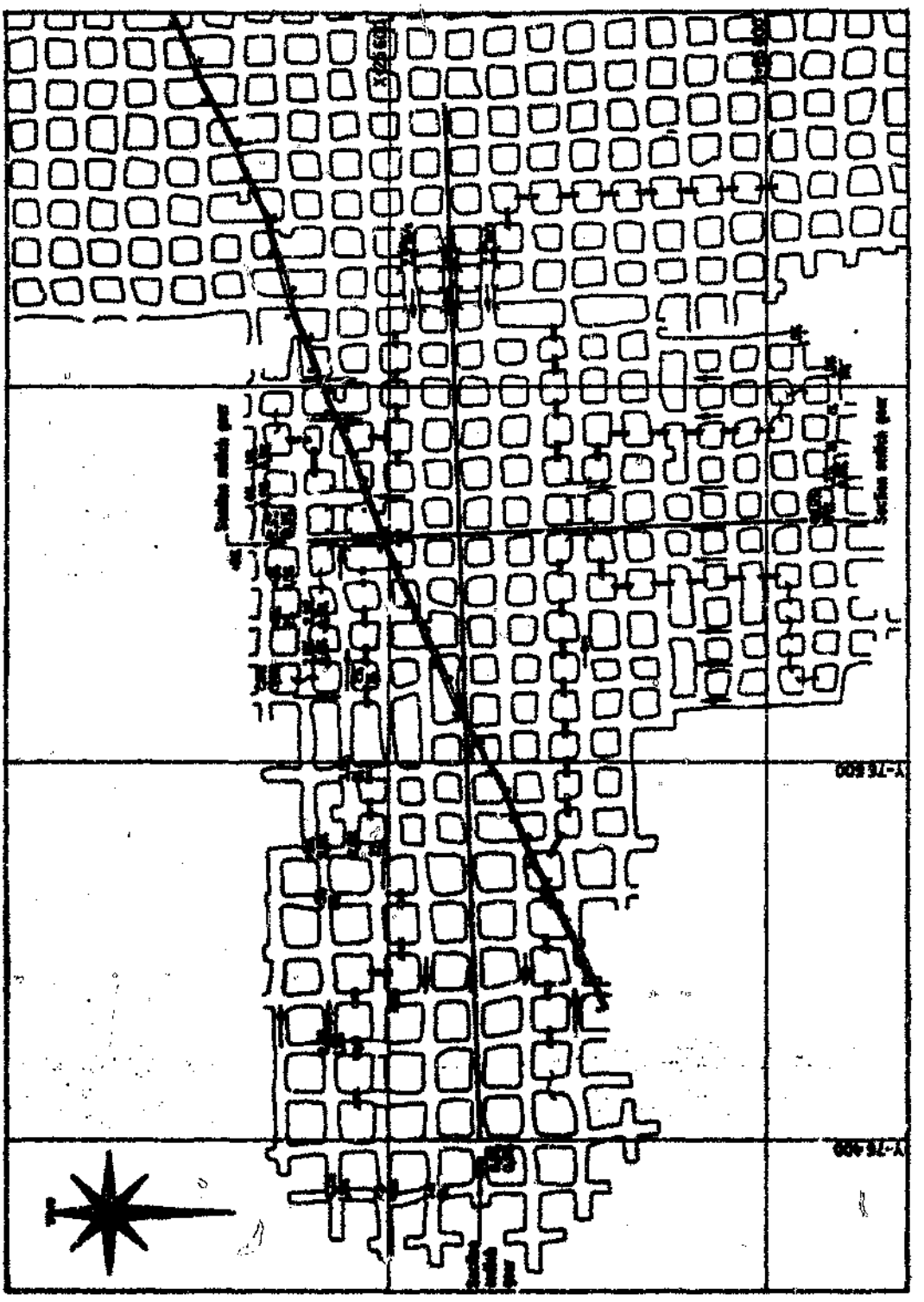


Figure 5

ERMELO MINES SERVICES (PTY) LTD.
PLAN SHOWING CH_4 BUILD UP
3 HOURS AFTER SWITCHING OFF
POWER AND FANS - TRIAL
Not to scale

methane observed was 1,9%.

Tests conducted at the original site of the joint box prior to the restarting of the fan, yielded a methane concentration of 2,8% on the return air side of the stopping and although the stopping had been sealed since the explosion, a concentration of almost 2% was found close to the stopping on the fresh air side. No source of methane other than seepage through the wall, could be found anywhere in the vicinity of the site on the fresh air side.

Discussion of Evidence

From the evidence it was clear that the Shiftboss, who was responsible for compliance with Regulation 10.13.3(c), conducted no methane gas tests other than in the substation where the switching on took place. The Electrician had assured him that the current would not flow past the next substation which was still approximately 1 000 m from the sections. Although the standard procedure governing this inspection prior to switching on was vague, the Shiftboss admitted that from previous experience he knew exactly what to do.

The reason why the intermediate substation did not trip out on "no volt" was because the coil was burnt.

There was no way in which either the Shiftboss or the Electrician might have known that the current would not be interrupted at the intermediate substation.

The evidence of the Manager provides a general overview of the state of the affected area.

"I am the certificated appointed General Manager in charge of Kmelo Mines Services (Pty) Limited, a controlled fiery coal mine in the magisterial district of Davel, Transvaal.

On 25th February 1984, the 7 West main section comprising working

sections, 3, 10 and 11, formed part of the Ermelo mine, being underground sections in the Tafelkop area.

On that day I was informed of an accident which occurred in the 7 West main area and being out of town, I arrived at the mine at 21h50.

I was then informed that there had been an ignition of methane in this area. Nobody was killed or injured during this occurrence.

I proceeded underground with proto teams and in the presence of the Inspectors of Mines and Machinery and other persons. A fresh air base was established and the proto teams inspected the affected areas before we entered to ascertain the cause of the incident.

It appeared that the main surface fan at Tafelkop had been switched off at 12h00 on 25th February 1984 for necessary repairs and maintenance. Once repairs had been affected, the fan was restarted at 17h45. At 18h45 the Shiftboss proceeded underground to clear the area, that is, to examine for methane and if found "clear" to declare it safe for resumption of work and the restoration of the electrical supply to the section.

At 19h00 the power was restored in this area by the Electrician and at this time the explosion took place in this area.

During the inspection in loco in Section 10, it was found that a considerable amount of damage was caused to air crossings as shown on Figure 4. The original condition and layout of the area is depicted on Figure 3.

Ventilation ducting had been torn down during the explosion. I found two broken cables, one serving an electric fan and the other served the roofbolter. There was no serious structural damage to any equipment in this section.

I was informed by the Manager that he thought that he had found the cause of the explosion. We then visited Section 3 where it was noticed that a cable joint box, indicated on Figures 3 and 4, had blown up. This was a "B I C C" 6,6 kV joint box. It was positioned against the roof and it was obvious that it had given off a considerable amount of heat or flames.

The rest of Section 3 was similarly damaged when compared to Section 10. Several brickstoppings and two partially built air-crossings had been blown out. There were also three pieces of burnt wires at different pegs, namely, numbers 7866, 7674 and 7648. All the ventilation ducting had also been torn down. Very little damage was observed in the left hand side of the section except for a few brattices which had been blown down.

The only damage to equipment was to the explosives magazine which contained 7 cases of explosives. This magazine (a portable unit) had been blown about 20 m from its original position next to the brickwall - see Figure 4.

Opposite Section 10 the force had removed the conveyor belting from the structure as indicated on Figure 4.

I cannot explain why the force of the explosion, which in my opinion occurred in Section 3, did not blow the conveyor belt off the structure opposite Section 3.

It was clear that a methane explosion had occurred and that this was triggered by the blow-up of the connector box or joint box, referred to above.

At the time of the incident, the following persons were in charge of working 7 West main area:

Underground Manager
Mine Overseer
Shiftboss

Underground Engineer

Senior Foreman

The only persons underground at the time of the incident were the Shiftboss and the Electrician who were at the station, a distance of approximately 2 km from the explosion.

By Court

The 7 West main area has a history of methane occurrence. If unventilated, methane can be detected in the working faces within 30 minutes. Bleeding holes are drilled at all intersections to relieve the pressure on the roof and methane issues from these holes. Methane is also given off from the freshly exposed coal where mining is done.

The mine operates under a standard procedure in the case where a main fan is switched off for some reason. This procedure dictates the following:

Power supply to the underground workings will be automatically switched off.

Upon restarting of the fan, the underground electrical supply will only be switched on after verbal instruction to do so has been received from the Shiftboss in charge of each ventilation district. Before giving such an instruction, the Shiftboss is to make sure that there is no danger of any inflammable gas being ignited by such action. Although not detailed in the standard procedure, I see this instruction as making it incumbent on the Shiftboss to inspect his entire area, that is, into his working places, for methane gas before giving the instruction for the power supply to be switched on.

There is no written standard regarding the time lapse from switching the main fan on until the current is restored to the workings.

The Shiftboss doing such an inspection is also required to enter the results in a book. I was informed that the Shiftboss had not carried out this instruction.

A simulation of the circumstances was carried out in the same sections on 3rd March 1984 under the supervision of the Manager and he was accompanied by two Inspectors of Mines.

From the results of these tests, it was obvious to me that, after the main fan had been switched off for 3 hours, it was found that approximately 7 m^3/s of air was still circulating through this section. This was caused by the two main fans operating from another area. After three hours it was found that the gas build-up was up to + 5 percent in Section 10 while in Section 3 from 2 to +5 percent had accumulated during this period.

The coupling which exploded was an approved coupling.

Sections 3, 10 and 11 form one ventilation district. The total production from these sections is approximately 1 200 tons per shift. The air flowing into these sections is about 60 m^3/s."

The series of events which led to the explosion are described by the Shiftboss and the Electrician who were sent into the mine to test for methane, inspect the working places and restore the electricity supply.

"On 25 February 1984 I was the Shiftboss in charge of work in the Tafelkop area when the explosion took place at approximately 19h00 in 7 West main.

I arrived at Tafelkop shaft at about 17h45 on the day of 25 February 1984 to accompany the Electrician to test for gas and to

inspect the work places before the electricity supply was restored.

On the way back from Twee ..ein shaft I called at the main fan and ascertained from the Engineer that he was busy switching on the fan. It was about 17h55.

At about 18h45 I went underground. I found that the ventilation flow was strong. I tested at the station for methane gas and as far as the E substation near the shaft. I used a methanometer. I also examined the E substation for methane - it was negative.

Figure 6 shows the electrical distribution into the mine.

The Electrician was with me. We arranged that he would switch on the power in substation E. He assured me that the power would only be switched on to substation F from E. Substation F is about 1 km from substation E. I stood at the entrance when he switched on the power. However, immediately after the power had come on, it went off again. He said it seemed as if something was wrong as the power did not want to come on. I do not know if he tried to switch on for the second time. I heard or felt nothing during the switching on.

We then left for substation F. I did not test for methane at substation F before we switched on the power at substation E.

About 30 m before we reached substation F, there was dust in the air. I then tested for the first time at substation F. There was no gas. The Electrician went into the station and said that the switches had not cut out. He then switched them off and we went to station E and en route examined the cable for defects. There were none.

We then went to the OCB switch at the Section 3 conveyor belt head next to the road. There were no defects on the cable. We then went into Sections 3, 10 and 11.

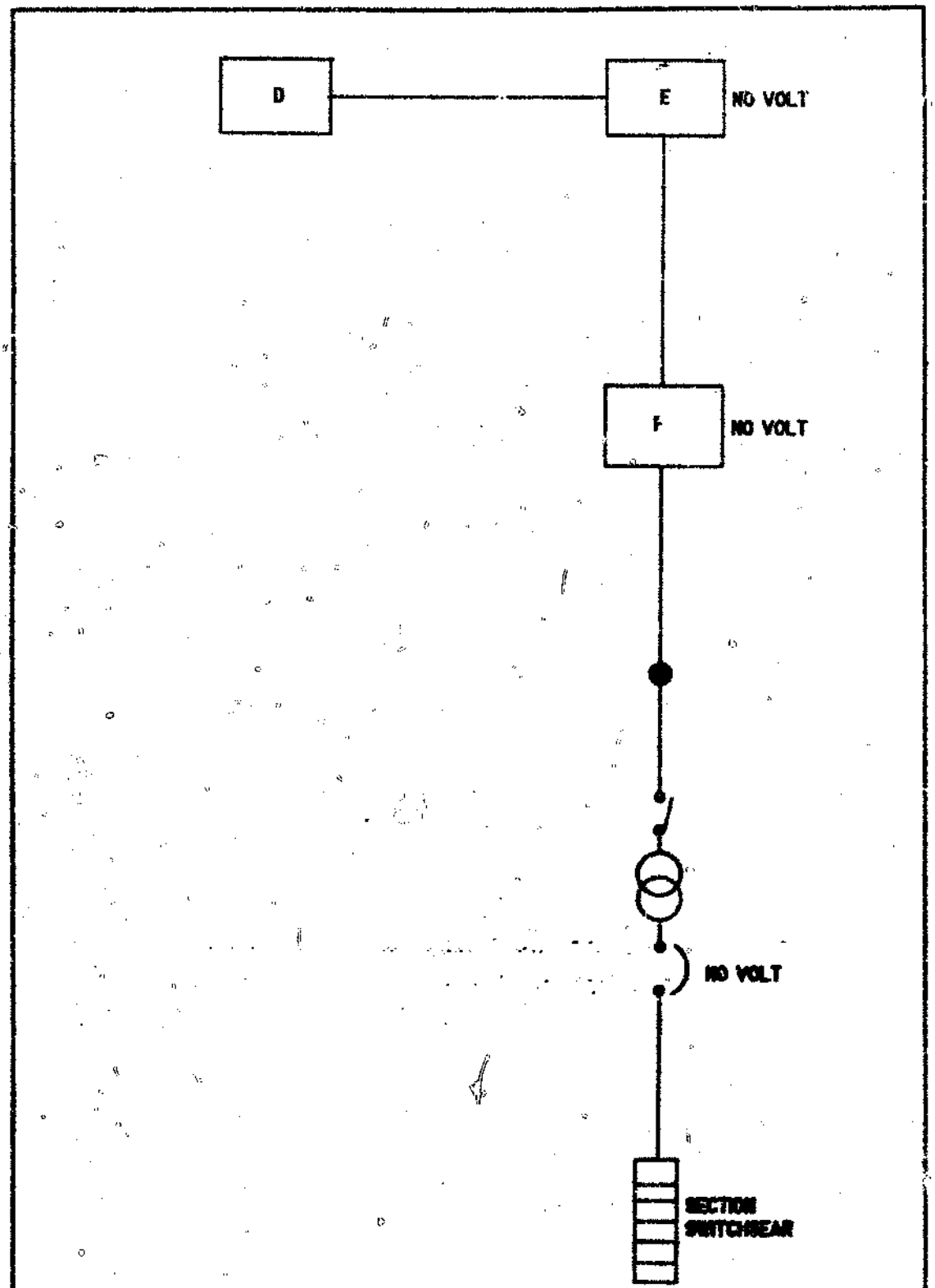


Figure 6

ERMELO MINES SERVICES (PTY) LTD.
DIAGRAM OF ELECTRICAL RETICULATION
FROM SURFACE TO SECTION 7 WEST

At the air crossings between Section 2 and 10 dust was noticed. I tested for gas and it was negative.

At the belt head in Section 10 there was a great deal of dust and we noticed that something abnormal had occurred.

We went to this side of the Section 10 air crossings and moved to the transformer. There were no cable defects and neither was there any gas.

When I reached the air crossings of Section 10, I saw that something abnormal had occurred. The section was full of dust. We then reported the accident.

By Court

When the main fan has stopped and before the power supply can be restored, one has to test at all the switchgear for methane gas, and also at the section fans before they are switched on.

I realise that before I gave permission to switch on the E substation I should have tested in both E and F substations for gas because the power would then have only been restored up to F station. If the power were to be switched on from F to the sections, I should have first carried out tests at all the switchgear in all the sections.

I know what the procedure is through previous experience.

I did not test in the F substation because I assumed that there would not have been any gas.

I did not make the necessary notes in my log book on that day. Because of everything that had happened that day, it slipped my mind.

No further questions.

On 25 February 1984 I was the Electrician accompanying the Shiftboss to switch on the power supply to the sections. When I switched on the power in substations E the OCB fell out. However, it did not trip because of overload or earth leakage.

I then tried to switch on again. It then showed that it had tripped on earth leakage. I locked the switches in the down position.

In the area of substation F there was dust in the air. The OCB had not fallen out. The power would therefore have flowed through the substation to the transformers outside the sections. It was strange that it had not tripped, because when no power flows through the unit, it automatically trips. I then switched off the switchgear.

By Court

When I switched on at substation E, I assumed that the switchgear at F station had tripped on no-load."

Conclusion

The main surface fan had been switched off for approximately 6 hours for repairs.

One hour after the fan had been switched on again, the electrical supply at the distribution substation was switched on near the shaft. Due to a defect in the switchgear in the intermediate substation, the current flowed through to the section. En route a 6,6 kV joint box exploded and ignited a tongue of methane which resulted in the explosions.

From the evidence it was clear the the Shiftboss, who was responsible for compliance with Regulation 10.13.3(c), conducted no methane gas tests other than in the substation where the switching on took place. The Electrician had assured him that

the current would not flow past the next substation which was still approximately 1 000 m from the sections. Although the standard procedure governing this inspection prior to switching on was vague, the Shiftboss admitted that from previous experience he knew exactly what to do.

The reason why the intermediate substation did not trip out on "no volt" was because the coil was burnt.

There was no way in which either the Shiftboss or the Electrician might have known that the current would not be interrupted at the intermediate substation.

Even if the Shiftboss had checked all electrical installations in the section, the observations made during the subsequent tests indicate that he would not have found gas at such units. It is unlikely that he would have tested at the joint box, the only electrical unit where gas was present.

Had the current been interrupted at the intermediate substation, this would not have prevented the explosion.

The "B.I.C.C." half of the joint box was submitted to the SABS who found that the blow-out was a result of a phase to phase fault on the box.

This type of joint box has previously caused similar blow-out problems.

Although approved, this half of the joint box did not display the CMR stamp of approval. This is the result of a factory error and the mine erred in using equipment which did not fully comply with the conditions of approval.

6.6.2 Indumeni Colliery

Within a period of one year this colliery was to experience four serious methane explosions, the third explosion causing the deaths of three employees. The dates of these explosions are given in Table 6.1 below:

Table 6.1 Summary of Explosions at Indumeni Colliery

Date	Details of Incident	Casualties
10th April 1967	Main fan was stopped for blade and motor changeout. An explosion devastated 2 sections after the ESC power was restored	Nil
1st October 1967	Explosion destroyed new vertical shaft headgear and shaft bottom infrastructure	Nil
15th December 1967	Explosion occurred when power was restored to a flameproof switch which was being worked on by an Electrician	3 killed
26th March 1968	Blown out shot ignited methane in a development section	Nil

In the first explosion on 10th April 1967 it was fortunate that no persons were underground since the force of the explosion devastated two sections.

On 15th December 1967 a methane explosion in Section 300 killed an Electrician and his two Helpers. The accident occurred at 09h30 and the proto team who entered the section reported that:

- There had been an explosion in the Main road A of Section 300 as shown on Figure 7.

- That 3 bodies, one White and two Black, had been located.

- That the electrical switch near the pump motor had been left open with the cover off and the cover bolts neatly packed on one flange of the switch box. The position of the pump is reflected on Figure 7.

- That the water level in Road A was some 1,2 metres below or inbye of the pump position.

- That at a position approximately 5,5 metres inbye from where the water started in A, the level of the water was approximately 1 metre below the roof.

The sequence of events prior to the explosion were as follows:

On instructions by the Mine Overseer who said he had examined Section 300 on 12th December 1967, the Electrician and his two helpers went through the waiting place on 13th December 1967 and replaced a push button station on the pump starter. Prior to switching on power after the repair he replaced the flameproof cover. The pump was started.

However, on Thursday morning, 14th December 1967, the pump Gang Leader reported that the pump had stopped. The Gang Leader accompanied by the Electrician and his two Helpers again entered the section inbye of the waiting place (this time on the instructions of the Shiftboss) and repeated the repair work of 13th December 1967. They started the pump.

Again the pump stopped some time thereafter and on 15th December 1967, the Electrician and his helpers went into the section, this time to replace a contactor coil in the starter.

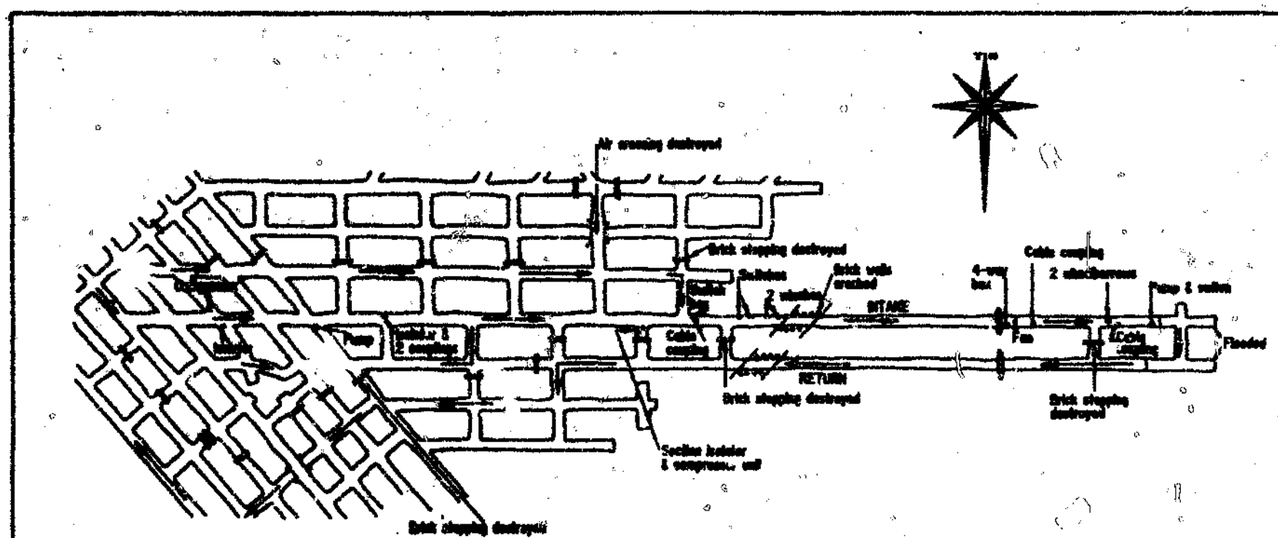


Figure 7

INKUMENI COAL MINES LTD.
PLAN SHOWING 300 SECTION
WHERE EXPLOSION OCCURED

On this occasion, after replacing the contactor coil, he had the power restored to the switch at the No. 2 section isolator before replacing the flameproof cover. He wished to test the switch by starting the pump. When the contacts opened, an arc was drawn between the contactors of sufficient intensity to ignite the methane/air mixture in the roadway. The switch was 500 volts and the spark or flash could have been as much as 25 mm in length. The switch was 30 cm above the floor of the roadway.

An air quantity of 2,03 m^3/second was circulating through the section prior to the explosion and before the water had risen to the point where it closed off the split C thereby stopping all air flow to the section. Tests taken after the explosion when the ventilation was returned to normal, showed methane concentrations of 0,2% to 0,4% in the general body of the air. This methane was emitted from a small blower in the floor near the pump site.

Electricians were not equipped with methanometers and it follows that the Electrician would not have been aware of the presence of methane at the pump site unless he had been accompanied by a person qualified and equipped to test for methane (Miner, Shiftboss or some other senior Official).

Evidence was that with 2,25 m^3/second circulating through roadway C the methane content of the ventilation remained at 0,4%. Methane was therefore being emitted from the fissure at the pump at a rate of 0,008 m^3/second.

The volume of air in the roadway A from the split D to the face when the water level sealed the split C is calculated to be approximately 240 m^3.

With a methane emission of 0,008 m^3/second and the split C sealed by water, a concentration of 9.5% methane would be recorded 47 minutes after the split was sealed by the rising water level.

This confirms previous tests at the Durban Navigation Colliery and elsewhere that dangerous methane concentrations are liable to be recorded in roadways in gassy seams when ventilation is disrupted.

6.6.3 Utrecht Colliery

Introduction

Fifteen workers were killed in a section at the commencement of shift on 20th November 1941 when a fall of roof in the main haulage of Section No. 1 resulted in an electric cable being pulled out of a junction box. The resulting electric arc ignited methane which was present in the haulage and the explosion extended into the return airways on the right hand side of the section, where dangerous quantities of methane were present. The force of the explosion escalated and destroyed all the section ventilation appliances.

The ventilation to the section was sluggish (1,25 m^3/second) and had to be boosted (to 3,75 m^3/second) by the use of an auxiliary fan. Three dykes had been penetrated and several days prior to the explosion a fissure in the floor of the left hand companion commenced giving off methane at high pressure.

The Incident

Figure 8 is a plan of the section at the time of the explosion. The explosion occurred at 07h00. Figure 8a is a plan of faces of Section 1.

At 12h15 in company with a rescue team, the Manager and a Shiftboss, reached a point in No. 3 North Haulage some 300 metres outbye from No. 1 Section waiting station. There were signs of carbon monoxide in the air 76 metres inbye of this point.

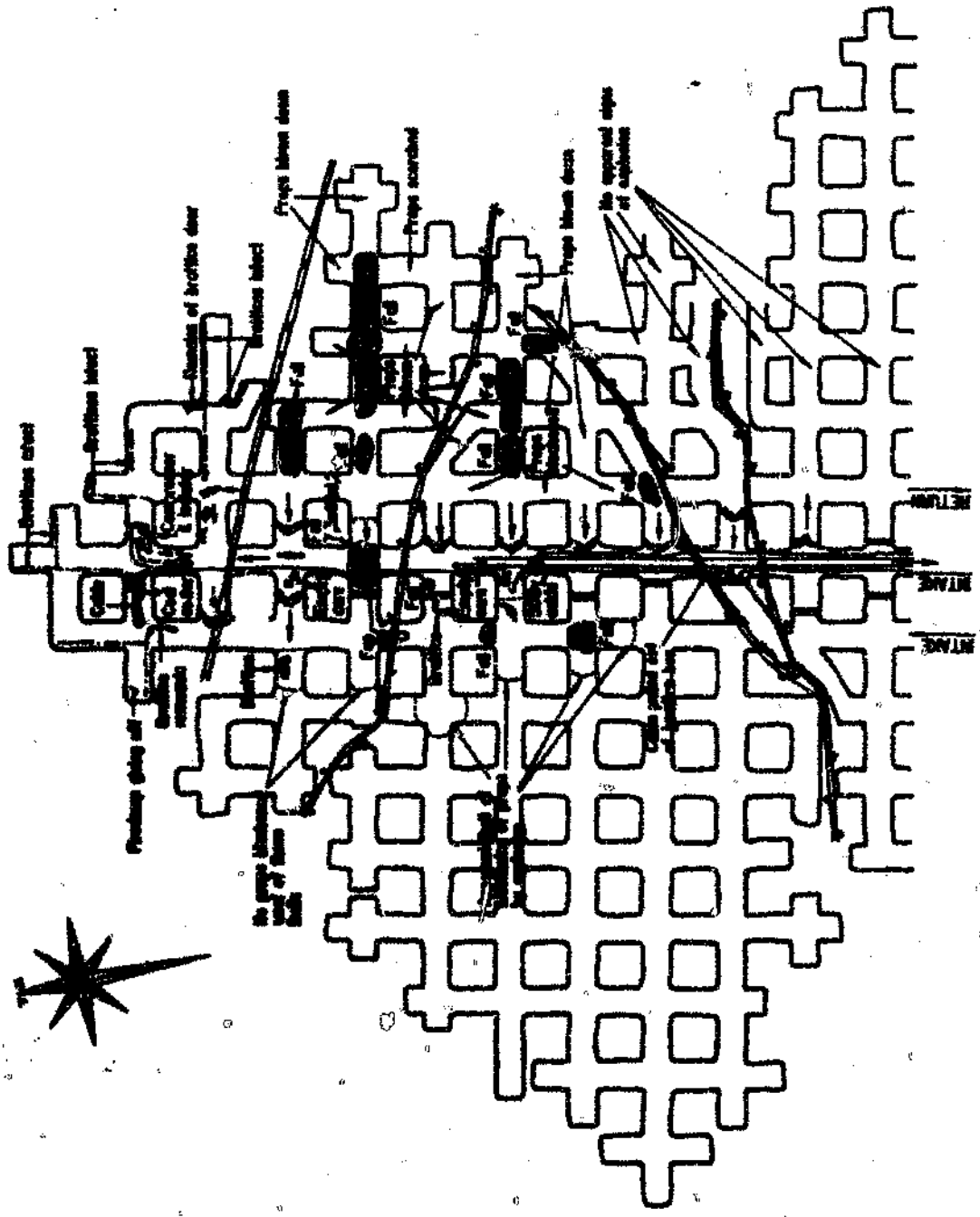


Figure 8

UTRECHT COLIERY
PLAN SHOWING SCENE OF EXPLOSION
IN SECTION I ON 20/II/41
Not to scale

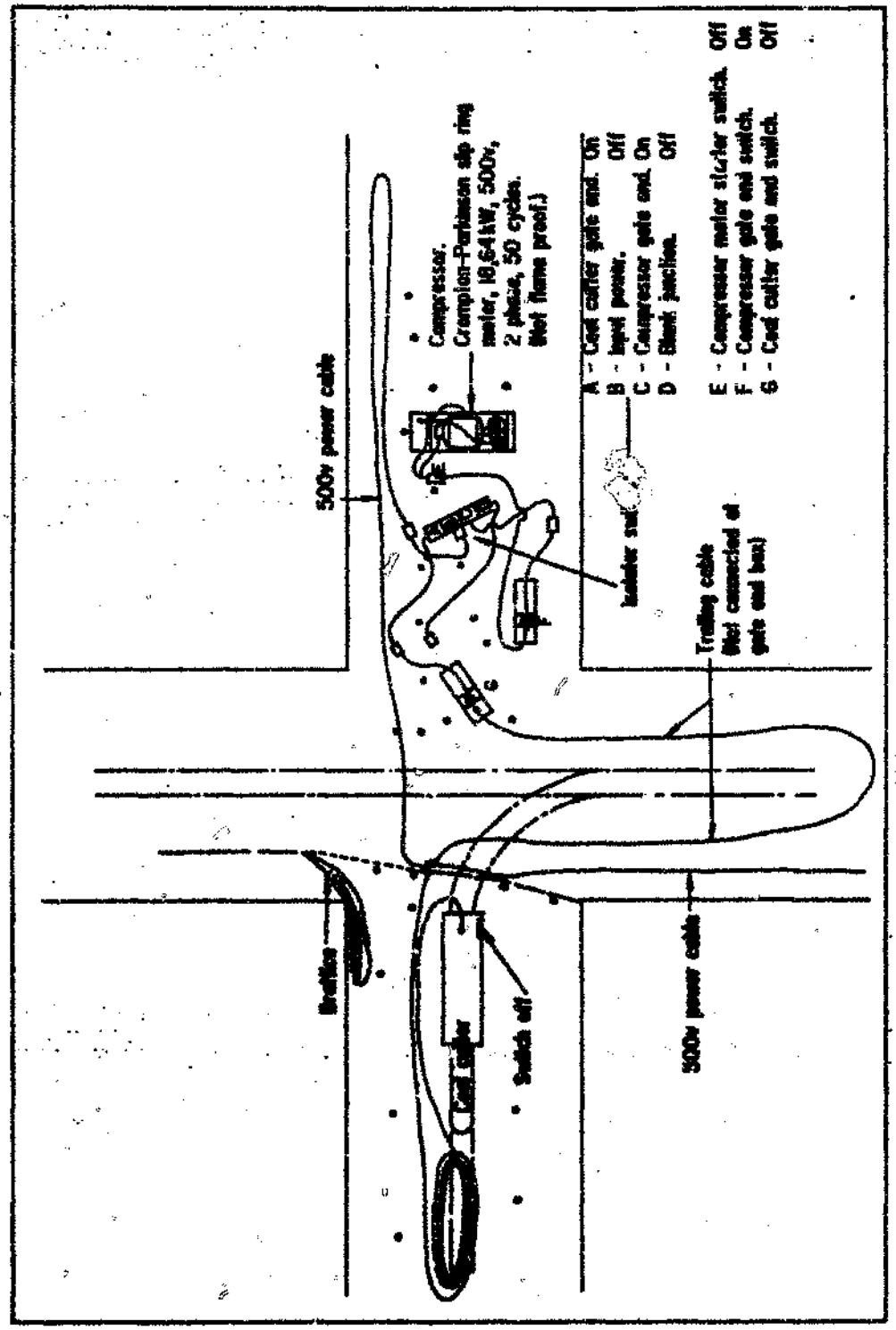


Figure 6a

UTRECHT COLLIERY.
DETAIL FACE PLAN
Not to scale

On the way in along No. 3 North Haulage the bodies of three Blacks were seen in the haulage engine chamber of the No. 2 section haulage station. The body of a fourth Black was lying on a trolley 370 metres outbye of the No. 1 section waiting station. These bodies had been carried out from where they had been found near the No. 1 section waiting station.

A preliminary exploration of the explosion area was made by rescue men wearing Proto apparatus who reached the face in the main road between the furthest inbye dykes. They returned to report the presence of over 5% of firedamp and dangerous quantities of carbon monoxide in the vicinity of the waiting station, sufficient to kill the canary they carried with them.

Ten bodies, all Blacks, were reported to be lying in the immediate neighbourhood of the waiting station.

A short while later, rescue men explored the No. 1 Section working places to the left of the main and inbye of the furthest dyke. Much firedamp was reported to be present in these workings. No more bodies were reported found. The working places explored were on the intake side of No. 1 Section.

It was clear that no living person remained in the affected area and from the reports of the rescue men, who had carried out a partial examination, it seemed most improbable that a fire was burning inside.

The flow of fresh air into the section was extremely sluggish and it was evident that until the noxious gases had been cleared no useful purpose could be served by further exploration or other work.

The Manager went into the return airway from No. 1 Section by going through the single door separating it from No. 3 North Haulage and found a strong after-explosion smell and about 3%

firedamp. This door is situated 210 metres inbye from the No. 2 Haulage station.

At 20h10 on 20th November 1941 it was found that although the ventilating current had not improved appreciably, despite the bratticing off of other splits, it was possible to reach the holing in the main through the first dyke inbye from the waiting station without Proto apparatus. Beyond this point firedamp was present to over 5%. A start was made at the temporary repair of blown out stoppings with brattice cloth.

Ten bodies were found at the waiting station or its immediate vicinity and were sent out of the mine. The other four bodies had been sent out previously. Points where the bodies were found were marked and the only means of identification then was by taking the number of the lamp, if any, found at the body. It was subsequently found that the lamproom records were so incomplete and badly kept that no reliance could be placed on them. It is therefore not possible to indicate where each particular person was found.

The large fall in the main which had taken place at a cross-roads between the two furthest inbye dykes and on top of a three-way junction box on the 500 volt power cable, was inspected. The cavity in the roof was clean and there was no traces of firedamp in the cavity.

On Monday 24th November when detailed measurements were being taken, it was observed that at a junction box, on the 500 volt power cable, situated 292 metres from the face of the main and in between the third and fourth dykes from the face of the main, the inbye end of the cable had been pulled right out of the box (see Figure 8).

The freed ends of the three conductors bore no signs of sparking, nor was there anything else in the vicinity to suggest that there had been a strong electrical flash. It appeared that the cable

had been under no electrical load when it was pulled out of the box.

The large fall in the main on top of the 500 volt power cable was cleared sufficiently to expose the threeway junction box on Wednesday 26th November. It was found that this junction box had been protected by the stacking of timber round it and that it had not been damaged by the fall. There was no internal damage to the box and no signs of any sparking.

On Wednesday 26th November the broken junction where the cable had pulled out was repaired and the Electrician tested the cover and reported it to be safe and undamaged.

By then a week had elapsed since the date of the explosion and although stoppings had been rebuilt and plastered with cement instead of clay and brattices had been restored, it was found that the ventilating current into No. 1 Section was inadequate to clear the right hand side workings of firedamp in order that an inspection could be carried out with safety. For this reason it was considered necessary to start the auxiliary fan.

The links were replaced in the three-way box at the No. 2 Haulage station and on Friday 28th November the auxiliary fan was started without untoward incident, the fan and switchgear having suffered no damage from the explosion. Improvement in the conditions regarding firedamp resulted and the workings on the right hand side were inspected. Several extensive falls were encountered. Their positions are shown on the plan (Figure 8).

Table 6.2 below sets out the ventilation readings and gas concentrations taken in the section on the 3rd and 4th December 1941 after the stoppings and brattices had been restored. The main fan was operating at the time.

Table 6.2

1. An auxiliary fan drift in the left hand companion

a. Auxiliary fan running

Quantity flowing inbye 5,45 m^3/second.
M.S.A. methane detector reading 0.25% to 0.3%.

b. Auxiliary fan stopped (17 hours)

Quantity flowing inbye 5,22 m^3/second.
M.S.A. methane detector reading 0.25% to 0.3%.

2. In main road at the miner's box

a. Auxiliary fan running

The air moved into the main in an outbye direction but so slowly that the anemometer would not register the flow. M.S.A. methane detector reading 0.5%.

b. Auxiliary fan stopped (18 hours)

The air moved into the main in an inbye direction but so slowly that the anemometer would not register. M.S.A. methane detector reading nil percent.

3. In right hand companion at holing through furthest inbye dyke

a. Auxiliary fan running

Quantity flowing outbye 3,02 m^3/second.
M.S.A. methane detector reading 1.0% to 1.1%

b. Auxiliary fan stopped (20 hours)

Anemometer failed to register the slight current moving outbye.
M.S.A. methane detector reading 3.2%.

4. In right hand companion at holing through third dyke outbye from the face

a. Auxiliary fan running

Quantity flowing outbye 2,839 m^3/second.
M.S.A. methane detector reading 1.0% to 1.1%.

b. Auxiliary fan stopped (20 hours)

Quantity flowing outbye 0,99 m^3/second.
M.S.A. methane detector reading 2.2%.

5. M.S.A. methane detector readings

On 4th December 1941 the auxiliary fan having been stopped for 18 hours.

At the junction box where the cabled had pulled out nil %.
At the waiting station - nil %.
At the miners' box - nil %.
In the return at the door 213 metres inbye of No. 2 haulage station - 9%.
At the face in heading to the left off the left companion inbye of the last dyke - 0.3%.
In the left hand companion at the turn off to the above heading - 0.3%.
At face of left hand companion - 0.75%.
At face of main - over 5%.
At crossroads 18 metres outbye from face of main - over 5%.
At compressor - 3.0%.

It appears that prior to the explosion an explosive mixture of air and firedamp had accumulated in the workings on the return side of No. 1 Section. If the auxiliary fan was started on the 20th November a portion of the explosive mixture may have been drawn into the main road from the right hand companion as a result of ineffective brattice doors or leaking stopping between the main road and the right companion.

Due to the fact that the affected section was damp and well stonedusted, the explosion did not develop into a coal dust explosion.

After the explosion and when circumstances permitted inspection it was found that firedamp was being given off freely from a crack in the floor of the heading to the left of the left companion and inbye of the dyke. This crack started some 6 metres from the face of the heading and when first observed on 21st November, was not extensive, but during the next week it developed and reached back to the crossroad at the left companion.

Although firedamp was being given off freely and bubbling vigorously there were no signs that a blower of any great violence had existed there previously since most of the crack was covered over with small coal.

At the face of the left hand companion a moderate amount of firedamp was issuing from the floor. Firedamp was being given off freely at the face of the main where 5% and over was detected a fortnight after the explosion. There were no signs in any of the working places that an outburst of methane had occurred.

The view was expressed that during the night of 19th November, while the auxiliary fan was stopped, firedamp was being given off freely from the exposed coal faces and the floor of the working places inbye of the last dyke and was carried slowly around

towards the right hand side workings by the feeble ventilating current of approximately 1 m^3/second.

The starting of the auxiliary fan on 20th November, shortly before the explosion, would have had the effect of driving the firedamp into the return. It is clear that a considerable body of explosive firedamp-air mixture must have been lying to the right of the main road between the working faces and the fourth dyke outbye from the faces.

At the time of the explosion the electrical reticulation of the section was as follows (see Figure 8):

A 2000 volt cable was carried along the left side of No. 3 North Haulage to a transformer located at the No. 2 Section Haulage turn off. From this transformer a 500 volt cable, carried along the left side of No.3 North Haulage, took power into No. 1 Section, as far as the isolating switch at the compressor.

A 230 volt lighting and signal cable was installed along the right hand side of No. 3 North Haulage. Light boxes were situated 45 metres apart. Each light box was fitted with a two pin socket.

The 500 volt power cable from the transformer extended to a 3 way box at No. 2 haulage Station. One lead ran from here into No. 2 Section to supply the coalcutter and the other lead extended to No. 1 Section for supplying power for the coalcutter, the auxiliary fan and the compressor.

An overload trip switch is in place between the 3 way box and the transformer.

Inside No. 1 Section a lead from the power cable was taken from a 3 way Ellison junction box to the fan motor.

The lights in No. 3 North Haulage and along No. 2 North Haulage were on the same circuit and should any short circuit develop

this cable was protected by a fuse located at a lighting transformer situated to the left off No. 3 Haulage some 400 metres from No. 5 West Haulage.

On the morning of the 20th November after the explosion it was found that the switch at the transformer at No. 2 Haulage Station had not tripped, that is, power was still on in the 500 volt cable into No. 1 Section. On instructions from the Mine Overseer the switch was closed. The fuse for the lighting cable had blown.

On Monday 24th November it was brought to the Manager's notice that sparks were coming from the No. 3 North Haulage rope at No. 5 West Haulage, near the No. 3 North Haulage engine. On investigation, it was found that a leakage of current was taking place to the frame of the haulage engine through the outer armouring wires of the lighting cable. This was as a result of an electric light lead with defective insulation suspended from the lighting cable by a piece of wire. The outer armouring wires of the lighting cable were thus being earthed to the frame of the haulage engine.

In addition, a bridge earth wire between the armouring wires of the two light cables at this spot were broken.

At the source of the fault the voltage between the haulage rope and any earth was low and the sparks were not intense. Persons were unable to feel any shock on touching the rope. The fault was causing the fuse at the lighting transformer to blow.

The inquiry established that this fault did not exist on the morning of the explosion, as the haulage lights had been on day and night. The fault would have been apparent on the day after the explosion when the fuse at the lighting transformer was replaced. After this fuse was replaced the fault did not develop until three days later.

At the auxiliary fan motor in No. 1 Section were two switches. One was a gate end switch the other a star mesh starting switch. Both these switches are trip switches, that is, if any electrical fault developed on the outgoing connections from these switches they would trip out. The switches were not fitted with "no volt" release devices. If an overload were imposed on the fan the switches would trip. It is doubtful if any vibration in the air would cause these switches to trip.

It would have required a considerable force to pull the cable out of the junction box where this happened on the haulage into No. 1 Section. The cable could not have pulled out had the vibration of the explosion shaken the cable loose from wires suspending it from tapes causing the junction box to drop a metre to the ground.

Checks on the earthing systems in the section after the explosion indicated that they were in order with the exception of the earth wire which was broken between two lighting cables.

Eight metres below the seam being worked was another coal seam 1,5 metres thick. Evidence given by the mine Officials was that it was common for cracks to appear in the floor of the upper seam. Evidence is that the cause of the cracks could have been due to either pressure from the coal pillars or fissures which existed in the parting. It was possible to see into a crack to a depth of 4 metre of gore and water issued from these fault planes. Two weeks after the explosion (with the auxiliary fan in operation), methane tests indicated 0,7% methane in the left hand companion, 5% in the face of the main and 2% in the general body of the air in the return airway.

Does this evidence not tend to show that, though on the night before the explosion there may have been a considerable quantity of firedamp coming off from the crack, nevertheless sufficient

firedamp was coming off from the faces to form an explosive mixture with only 1 m^3/second of air flowing into this section when the auxiliary fan was stopped.

The auxiliary fan in the section was always stopped at the end of dayshift by the miner (16h30) and started up at 06h15 on the next shift.

Details of the main and auxiliary fan (in Section 1) are given in Table 6.3.

Table 6.3 Main and Auxiliary Fan

Main fan - (at adit)

- Sirocco - centrifugal
- Diameter - 3,2 metres
- Quantity - 20 m^3/second
- Pressure - 249 Pa
- Drive - squirrel cage motor 46 Kw

Auxiliary fan (Section 1)

Quantity - 5 m^3/second
Drive - 7 Kw; 500 volt squirrel cage electric motor

Air was coursed through two other sections before entering Section 1. The reason given for stopping the auxiliary fan at night was that no adverse reports were received on methane in Section 1 during normal operations.

No rescue sets were available on the colliery.

Place of origin of the explosion

It is possible from information now available, to indicate the locality of origin of this explosion.

Particular importance was attached to the directions in which stoppings on the main road (between this road and the return airway) were displaced by the force of the explosion.

The stopping furthest from the face of the main road to be displaced was at a distance of 320 metres from the face. This stopping which was designated as No. 6 had collapsed and the way was thus open to the return airway on the east side.

The five succeeding stoppings on the same side of the road, at distances of 30 metres apart, had been blown in the opposite direction, from east to west. The four stoppings opposite (on the left side of the main road) were not disturbed. Further in towards the face, on the left hand side, the two brattice cloth stoppings (which acted as entrance doors to the fan) had been blown in towards the fan, also from east to west. The stopping at a distance of 30 metres further in and on the left side of the main road was undisturbed. The remaining two stoppings situated at 76 and 106 metres from the face in the left hand companion were again blown out from the east side towards the west.

It will be observed that ten stoppings (including the two brattice doors) were blown out. It will also be noted that nine of these were displaced from east to west. One only did not appear to have been blown out, but had collapsed.

It is certain that at the time of the disaster an explosive atmosphere existed on the right hand or east side of the main road, including the return airway, because the force caused by the explosion was particularly developed on that side.

If the explosion had originated anywhere on the east side of the main road it is certain that every stopping between the main road and the return airway would have been displaced in the same direction. There could have been no exception as was the case with No. 6 stopping.

It is clear that the force which collapsed No. 6 stopping and opened the way to the return airway came from the direction of the main road itself. This view is further supported by examination of dust deposits on the timbers between No. 5 and No. 6 stoppings. These indications point to force advancing from north to south before reaching No. 6 stopping.

Considering all the evidence of the directions of forces set up by the explosion, it is the view that, at the time of the accident, an explosive mixture existed in the main road in the vicinity of No. 6 stopping. This explosive atmosphere had been drawn through No. 6 stopping by the action of the fan, which is by no means an impossibility in the ventilation conditions existing at that time.

The accumulation of inflammable gas became ignited. The force developed by this ignition was sufficient to collapse No. 6 stopping and it passed out towards the return airway whereupon the flame at once extended into the more highly explosive atmosphere in the return airway. Here the explosion became more violent, travelled down the return airway blowing out the five other stoppings (from east to west) and extending also with great violence throughout much of the older workings on the east side. Thus the greater force did come from the east side of the main road, though the origin was not on that side.

Source of ignition of explosion

It is significant that between No. 5 and No. 6 stopping is the locality where the electric cable had been pulled from its junction box.

The junction box had, before the explosion, been suspended from timbers. It can be said that, if this cable had been pulled out whilst there was an electrical load on it, such as would have been the case if the auxiliary fan had been working, a considerable flash of flame (or electric arc) would result. The

temperature of such flash would be sufficiently intense to ignite an explosive mixture of firedamp and air.

To understand how the cable became disrupted from its box, it may be pointed out that it had extended from this junction box towards the face a further distance of 152 metres where it entered another junction box, a lead being taken off this box to the left to convey power to the fan.

This junction box was situated on the floor, on the left side at the crossroads, and it was here that it had been decided to take down a portion of the roof. The roof was, in fact, supported by props and it was only necessary to remove these to cause the roof to come down by its own weight.

In anticipation of this decision, the junction box on the floor had been well protected by means of a strong framework of timber which had been placed around and covering it.

Evidence is that on November 20th this portion of roof did in fact break away by its own weight. The fall was a heavy one, the dimensions being approximately 7,6 metres by 4,5 metres by 1,3 metres. The western end of the fall struck and covered the framework protecting the electrical junction box and the explosion followed immediately.

Although this heavy fall did not damage the junction box because it was so well protected, it did nevertheless push the entire framework of this timber guard (which surrounded the junction box) forward for some distance. This in turn set up a sudden pull on the cable attached to the box which pulled out the other end from the junction box near No. 6 stopping. One intense flash was produced and became the ignition point of the explosion which immediately followed.

That a flame actually did appear on the main road near this locality is confirmed by the evidence of Forager Banda. At the

time of the explosion he was walking to work along the main road and when at a distance of what is indicated as being some 120 metres outside the first dyke, he saw a blue flame which seemed to him to entirely fill the haulageway. He turned to run away but was knocked over by the blast from the explosion which followed the appearance of this blue flame.

The blue flame described by Forager Banda may have been the flame produced by the electric flash which would occur with the disruption of the cable from its junction box (which has a characteristic bluish colour) followed almost instantaneously by the firedamp flame which also is of bluish colour.

Considering the evidence, it is concluded that on the morning of November 20th, an explosive atmosphere existed in the vicinity of No. 6 stopping, which mixture became ignited when the 500 volt electric cable was pulled out of the junction box in the same locality. The fan was at work, the cable was under electric load, therefore a considerable flash was bound to result. No. 6 stopping collapsed when the explosive mixture present became ignited. The flame of the explosion passed through into the return airway where the explosion became greatly intensified because of the richer explosive mixture there and which blew out all the succeeding stoppings from east to west.

A demonstration on surface several weeks after the explosion in which a 0,04 cable carrying a 15 amp load was pulled apart, showed a distinct flash but no burning of the terminal wires.

6.7 SUMMARY OF THE PRACTICES REVIEWED ABOVE

- Although methane had been reported in a working section the previous week, it was not attended to and was discounted as being false and to be the excuse for low production.

- A holing into a return airway on a Saturday, which should have been sealed off immediately, went unattended; this caused a major short circuit in the section ventilation. The substantial drop in the ventilating quantity over the weekend resulted in a dangerous methane build-up in the section. The coal seam was emitting some methane which had not been detected.

- The machines in the section were not all properly flameproof.

- Water was not available at the start of the shift on Monday resulting in dry and dusty conditions. This dust exacerbated the force of the resulting methane explosion.

- Miners often do not test for methane at the commencement of the shift and thus do not detect the presence of methane before allowing men and machines into the section.

- A non-flameproof scrooptram in the one area of the section initiated an explosion as a result of a faulty silicon controlled rectifier panel, containing contacts.

- Examination of lamproom records show a serious state of neglect with regard to the correct recording of lamps issued.

- Polypropylene brattice cloth was inflammable in terms of an S.A.B.S. or British National Coal Board specification. However, as a result of the explosion the brattice cloth was extensively vapourised producing large quantities of carbon monoxide and carbon dioxide.

- Air is coursed from one section to another as a result of insufficient ventilation in the mine as a whole.

- Another company took over the financial and managerial control of a mine in June 1983. Only top management were changed, so theoretically a minimum of disruption should have taken place. However, whatever the cause, it was clear from previously mentioned incidents combined with other irregularities observed, that discipline was of a poor standard. Continuous rumours prevalent over many months about a possible closure of the mine would undoubtedly have had an unsettling effect on Personnel.

- Considering the low seam heights and physical discomfort, the area to be covered by each miner with all the required statutory duties to be carried out during his shift, was, if anything, too spread out and led to serious physical discomfort.

- The stopping of main fans for maintenance and repair work allows for the build-up of methane in sections. Start up procedures, once the main fan has been started, as far as restoring power to sections is concerned is often haphazard. The use of methane bleeder holes from over or underlying coal seams increases the rate of methane build-up. A lack of communication between mining and engineering Personnel, when restoring power, adds to the possibility of ignitions occurring.

- Using non-approved electrical 6,6 kV joint boxes in the electrical reticulation. These joint boxes had previously

given problems in the form of phase to phase faults. Suspending these boxes from the roof only increased the possibility of a methane ignition when a fault occurred in the box. B.I.C.C. joint boxes are of the pin type.

- After the main fan had been started, the electricity supply at the distribution substation at the shaft bottom was switched on. Due to a defect in the switchgear in an inbye intermediate substation the power supply flowed through to the section where a 6,6 kV joint box exploded and ignited a tongue of methane. The miner had not tested for methane in the section since the Electrician assured him that no power would reach the affected section.

- Assuming that unworked headings are adequately ventilated.

- Allowing Electricians, who have had little mining experience, to work on electrical switchgear prior to examining the faces for the presence of methane.

- Electricians render flameproof switchgear non-flameproof in working faces or hazardous areas; and simultaneously restore power to the opened unit to test for faults.

- Allowing water to build up in sections which then seals off roadways required for the ventilation of the section.

- Inadequate training of Electricians with regard to the dangers of methane and its detection.

- Switching off booster fans in a section at the end of day shift and restarting the fan the next morning. The siting of such fans can lead to recirculation of section ventilation which may contain dangerous quantities of methane.

- Electric cables carrying in excess of 500 volts can be pulled out of joint or distribution boxes as a result of some excessive force such as a fall of roof onto the cable.

- The installation of non-flameproof electrical equipment (control boxes and motors) in close proximity to working faces which are giving off methane freely could lead to a methane explosion as a result of arcing between contacts and brushgear.

- Mobile auxiliary fans which recirculate ventilation and as a result may draw methane from the face over a faulty fan motor or switch.

- Flameproof boxes which do not conform to C.M.E. standards.

- Poor discipline and supervision with respect to electrical distribution installations and maintenance of electrical machinery.

- The use of open wire signalling systems which are not intrinsically safe. On 14th October 1913 at the Universal Colliery in Senghenydd, Britain's worst mining disaster occurred when 439 men lost their lives in a coal dust explosion which devastated the colliery. John Brown (1981) summarises the Inquiry:

"So the two inquiries had come up with different answers to the same questions. The inquest jury, voicing the opinions of the Mining Engineers and Evan Williams, saw the probable point of origin at the lamp station and the flash point the flame of an open lamp. Redmayne, Smillie and many Mines Inspectors plumped for the Mafeking Hard Heading with ignition from 'sparks from the electric signalling apparatus or from rocks brought down from the fall'.

c. Flameproof Machinery

A very careful inspection of all the flameproof machinery in the two sections was carried out (28 items). Fourteen items were found to be non-flameproof for a variety of reasons varying from damage probably caused by the explosion to lack of packing in cable glands. By a careful analysis of the available evidence both with regard to the origin of the explosion and the extent of the variation from required specifications it became obvious that scooptram No. 56 was the igniting source of the explosion. An alarming discovery was the poor state of flameproof equipment that had allegedly been overhauled and restored to standard by firms contractors.

d. Machine Design

The fact that the fuses on the scooptram are contained in a large box with a heavy lid secured by 24 bolts in a relatively inaccessible position is regarded as extremely poor design. Items that require relatively frequent attention should be placed in conveniently located positions in boxes with round screw-in type lids. Mines, when purchasing underground machinery, should pay more attention to design features for easy maintenance and repair and also for operator comfort and safe operation.

4. Cable couplers/joint boxes which are to be installed should be positioned not more than 1 metre from the floor. The three pronged B.I.C.C. type coupler is particularly prone to phase to phase faults. Situated close to the floor it is less likely to ignite explosive methane/air mixtures which will be layered against the roof.

When equipment faults continue to manifest themselves the system should be immediately isolated and the cause of the fault sought and then rectified.

It would have been all so very neat and tidy if the findings of both inquiry and inquest had been identical but here again the happenings of 1913 were reflecting those of 1901 by not providing firm answers to the questions of source and cause. The inquiry was undoubtedly more extensive and technically thorough than the inquest for that reason. Redmayne's conclusions can be argued as carrying the greater weight."

- An electrical couplers insulator cracks are difficult to discover. Three pin male/female type couplers such as the B.I.C.C. type are difficult to line up and connections are not always of a tight fit - this could lead to sparking across loose pin connections. This in turn leads to heat and causes brittle insulation.

- Lack of proper earthing across couplers. When a high resistance fault occurs more heat can be generated.

- Surge current could blow the coupler as a result of:

  - cracked insulators
  - high resistance on pins or poor contact.

- It is considered bad practice to mix couplers; in other words, having halves which are manufactured by different suppliers. This may result in incorrect tolerances on male and female pins and sockets and lead to arcing.

- Couplers are made up underground where the expertise and environment is not conducive to good workmanship.

- Bridging out, by Electricians, of switches thereby resulting in trips not operating when a fault occurs.

6.8 PRECAUTIONS TO BE ADOPTED TO PREVENT IGNITIONS BY ELECTRIC SPARKING FROM FAULTY ELECTRICAL EQUIPMENT

The preceeding sections have provided details of several documented and researched incidents. Based on these, the following conclusions and observations can be listed:

1. The dilution and removal of methane is a prerequisite and this subject has been dealt with in other chapters. The provision of adequate ventilation to mining districts is not the sole factor in diluting methane emissions. Ventilation appliances have to be properly installed and maintained. The photograph below shows a well installed brattice curtain in a section.



2. High standards of installation and maintenance of all underground electrical distribution systems and equipment is necessary. The following photograph highlights the above statement.



3. The events which led to a methane/coal dust explosion in a colliery are described below and provide the point of departure for further discussions on precautions to be adopted.

a. Ventilation

On 10th September 1983, a cut was made which holed through the remaining coal in roadway No. 17 between intersections C and D, Figure 9. The miner in charge of section 10 reported this fact to his shiftboss who gave him no instructions. Later during the shift, the coal was blasted down. Because of the booster fan on the Gus seam, Figure 10, the air in the return airway was under higher pressure than in section 10, and air was thus blown into the section through the holing, reversing the flow of air on the left hand side of section 10. This disrupted the ventilation, and the supply of air to section 5 was reduced by some 60%. A diagrammatic representation of the ventilation after the holing is shown on Figure 11. The figure of 60% reduction in the ventilation of section 5 was arrived at by an experimental reconstruction of the ventilation arrangements after the accident. Under these circumstances, all the brattices, stoppings and so on, were in perfect condition. It is reasonable to assume that these ideal conditions would not have been the case on 10 September 1983 and that the reduction in the supply of air to section 5 was thus in excess of 60%.

The Miner reported the holing to the Shiftboss who told him to close it off. The Miner said that this was not possible as it was near the end of the shift, he was short of labour and his only roofbolting machine was at the opposite end of the section. The Shiftboss told him to make a plan and left the section. He reported to the Mine Overseer that the Miner had been told to close off the holing. As was to be expected, the Miner did not seal off the holing and in spite of the fact that both were well aware of the importance of doing so, no further action was taken that weekend; not even the elementary precaution of asking the

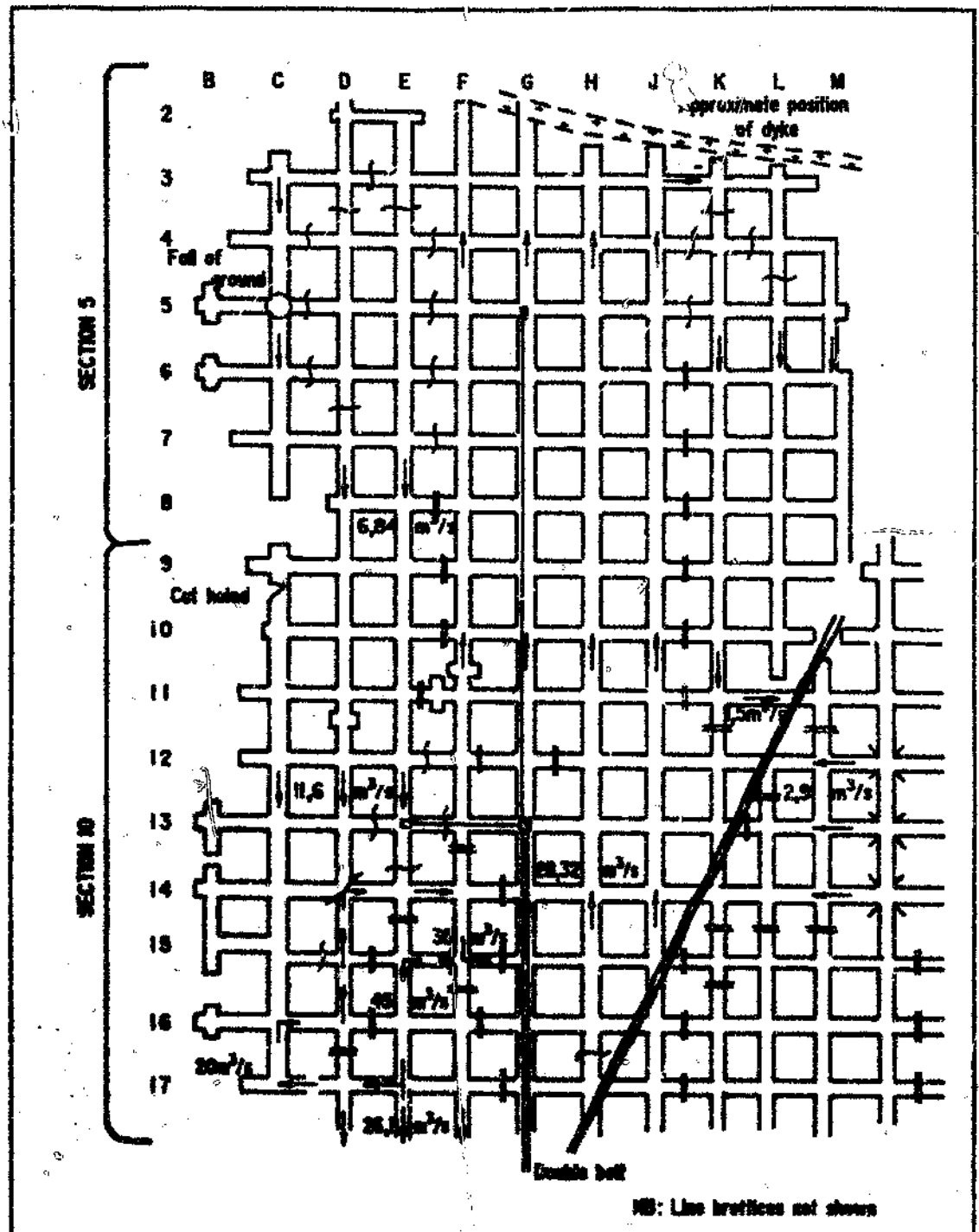


Figure 9

HLOBANE COLLIERY.
BOOMLAGER SECTIONS 5 AND 10
AFTER HOLING 17 C TO D
Not to scale

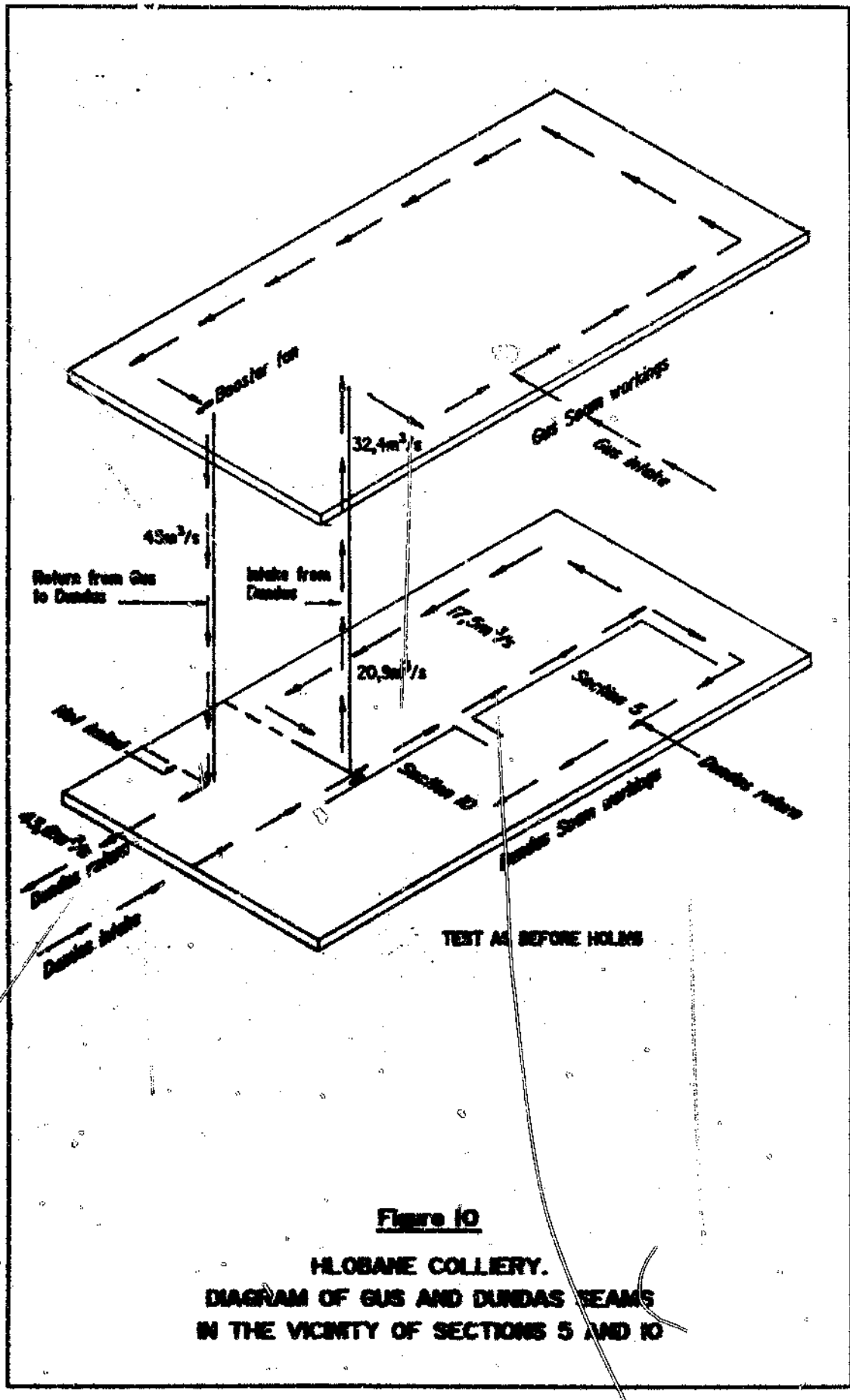


Figure 10

HLOBANE COLLIERY.
DIAGRAM OF GUS AND DUNDAS SEAMS
IN THE VICINITY OF SECTIONS 5 AND 10

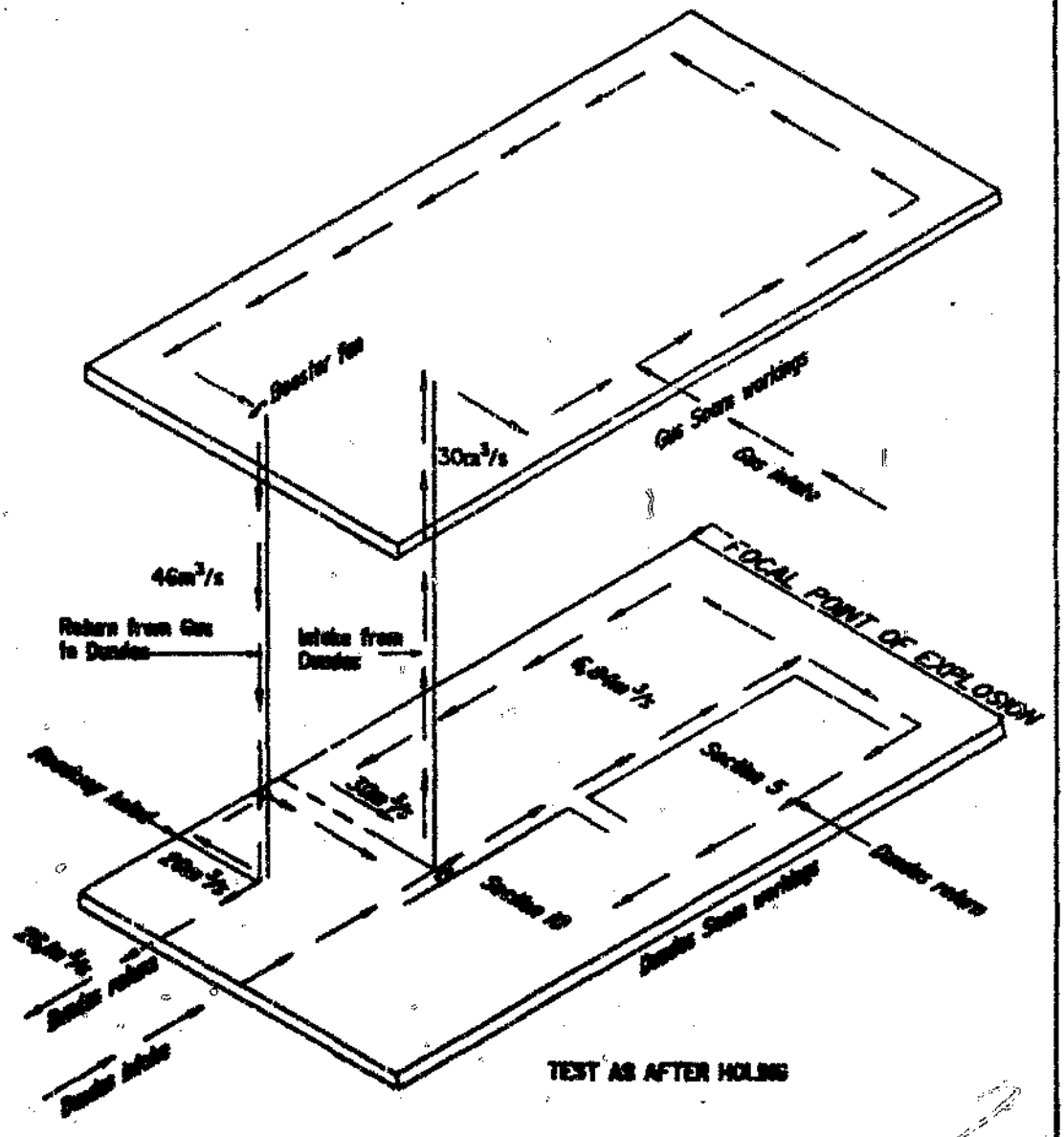


Figure 11

HLOBANE COLLIERY
DIAGRAM OF GUS AND DUNDAS SEAMS
IN THE VICINITY OF SECTIONS 5 AND 10