Miner whether he had managed to get the place sealed or not. The scene was thus set for the accumulation of methane in the northwest corner of section 5, which was also the highest point in both sections, with the roof sloping upwards to this point. An additional factor which probably played a part in the release of methane from the coal seam is the presence of a very deep low pressure system, associated with a storm, which passed over the mine on Sunday 11th September 1983.

b. **The Early Examination**

On Monday morning 12th September 1983, the Miners were delayed in going underground and reached their section waiting places in section 5 and 10 at about 06h30. There are no witnesses left alive to describe the early examinations or lack thereof, carried out by the miners, but the inquest court was satisfied that with the amount of methane that must have been present in the north western corner of section 5 and with the stirring action caused by persons moving in a seam height of 1,2 m this gas would have been detected if a proper test with a flame safety lamp had been carried out. The conclusion of the court was that the Miner who was in charge of section 5, failed to carry out the statutory tests and that he was thus primarily responsible for the accident. No trace of the initials and date required by Regulation 8.9.4(b) could be found anywhere in section 5, while in section 10, it was obvious from the initials and dates found that this Miner had examined about 60% of his working places before the explosion.

c. **Igniting Source**

From an examination of all the available evidence, there is no doubt that the igniting source was the silicon controlled rectifier panel of battery driven scrooptram No. 56. This machine was found near the source of the explosion with its motor still running, after the accident. The cover of the flameproof enclosure of the control panel had at some time been replaced so that a conductor was caught between the flanges of the lid and the box, resulting in a gap of some 10 mm. This gap is more than adequate for the propagation of a flame. The panel had been worked on on 9th September 1983 and there is little doubt that in replacing the relatively heavy cover in an awkward place in the confined space of a 1,2 m seam height, the conductor was trapped and when one of the securing bolts failed to go into its hole, this was ignored. The machine was subsequently removed from the mine for welding repairs and only re-entered the section just before the accident, so that it was not conclusively proved that the repairs on 9th September 1982 resulted in the non-flameproof state of the machine on 12th September 1983.

Matters arising from the investigation

An intensive investigation was carried out and the following are some of the more important matters that arose from this investigation:

a. Lamproom

The records kept in the lamproom were found to be chaotic and of virtually no value in the identification of the bodies of the victims. Had the explosion been more violent this omission could have had very serious consequences.

b. Explosives Control

A large number of contraventions of the regulations governing the use of explosives were encountered and it is obvious that the control of explosives and the discipline exercised by supervisors was extremely weak. There was no indication that explosives or blasting had played any part in the explosion but this may be regarded as fortuitious as untamped holes, charging up in close proximity to electric drills or any of a number of other explosives malpractices could have caused an ignition of gas.

5. Only GME approved explosion protected equipment shall be used in hazardous areas. Permission must be given by the Consulting Engineer and exemption obtained from the Inspector of Mines for the use of non-explosion protected equipment in hazardous areas.

6. Maintenance areas should be well lit and have well drained smooth impervious floors.

Care must be taken that oil and grease washed off the floors does not accumulate in sumps or old workings.

Permanent maintenance areas and workshops must have concrete or impervious floors and walls painted.

Maintenance areas must be kept clean and well stonedusted and be provided with B or C class fire extinguishers.

7. Lights and cables must be kept well clear of storage drums and spillage must be avoided. The storage area must be kept clean, well stonedusted and fitted with B or C class fire extinguishers and fire hydrants and hoses on the intake side.

8. Audit or overview teams consisting of mining, mechanical and electric Engineers should, once a quarter, examine the electrical reticulation, substations and electrical mobile face equipment and report on the standards of maintenance and installation and highlight hazardous situations. At least 8 teams of 4 Engineers per team should be deployed in the overview to ensure that the whole of the underground operations are adequately covered. Check lists should be provided for each team. A total mark of below 90% is considered unsatisfactory. Follow-up visits on hazards which have been highlighted should be undertaken without delay. Seconding team members from other collieries should ensure a more objective view of the electrical operations.

9. In the relatively low fault level systems traditionally used underground, well designed flameproof enclosures have proved very successful in preventing methane explosions.

10. A more serious problem has been cable faults. Because of the abuse to which cables, particularly trailing cables, are subjected, cable faults are fairly common. Even with the fastest protection the energy released in a short circuit is very considerable and is alm certain to burst the cable sheath and ignite any methane in the surrounding atmosphere.

In order to reduce the energy released during a fault it is necessary to ensure that the fault starts as an earth fault and restrict the earth fault current to the lowest value that can be safely detected. In trailing cables this is done by surrounding each phase core with an earth scree and by providing an impedance between the transformer neutral and earth. The flash due to a small earth fault current is likely to be contained within the cable sheath and even if the sheath is punctured the small spark is unlikely to extend any distance from the cable. This reduces the likelihood of the spark coming into contact with methane. There is the possibility of two phases going to earth simultaneously to produce a short circuit if the cable is crushed but the great majori of faults are single phase to earth faults.

An unwritten rule exists with regard to trailing cables that with the exception of shuttlecar cable damage, all other cable damage is almost entirely due to a lack of discipline. If cables are properly suspended in operating sections they will not become damaged with the consequential possibility of providing an ignition source. Shuttlecar cables are more likely to be damaged due to faulty reeling and spooling device, a situation outside the control of the Operator.

Equipment generally, and mobile equipment in particular, is subject to more abuse underground in coal mines than in normal surface applications.

Large heavy machines moving over uneven floors in confined spaces which are frequently poorly illuminated make accidental impacts fairly common. Machines are often severely overloaded.

Coal dust, water and oil do not make for an ideal environment for the operation of electrical equipment.

Roof heights of between one and two metres and drive widths of six metres place limitations on design and make handling of large units and long lengths of heavy inflexible cable difficult.

The regulations require armoured cables to be used underground in coal mines except for flexible cable for portable apparatus. They require that every cable unprotected by a metallic covering shall as far as practicable be so suspended as to protect it from damage. The operator of a mobile or portable electrical machine served by a flexible trailing cable must take all reasonable measures to safeguard the flexible cable against damage and cut off the supply to the trailing cable before leaving the section. Every flexible cable must be examined at the beginning and again at least once during the shift.

The inference is that flexible cable is not as safe as armoured cable. Flexible trailing cables are subject to much more abuse than armoured cables and experience has shown that well designed flexible trailing cables with individually screened phase cores and the correct protection are considerably safer than armoured cables.

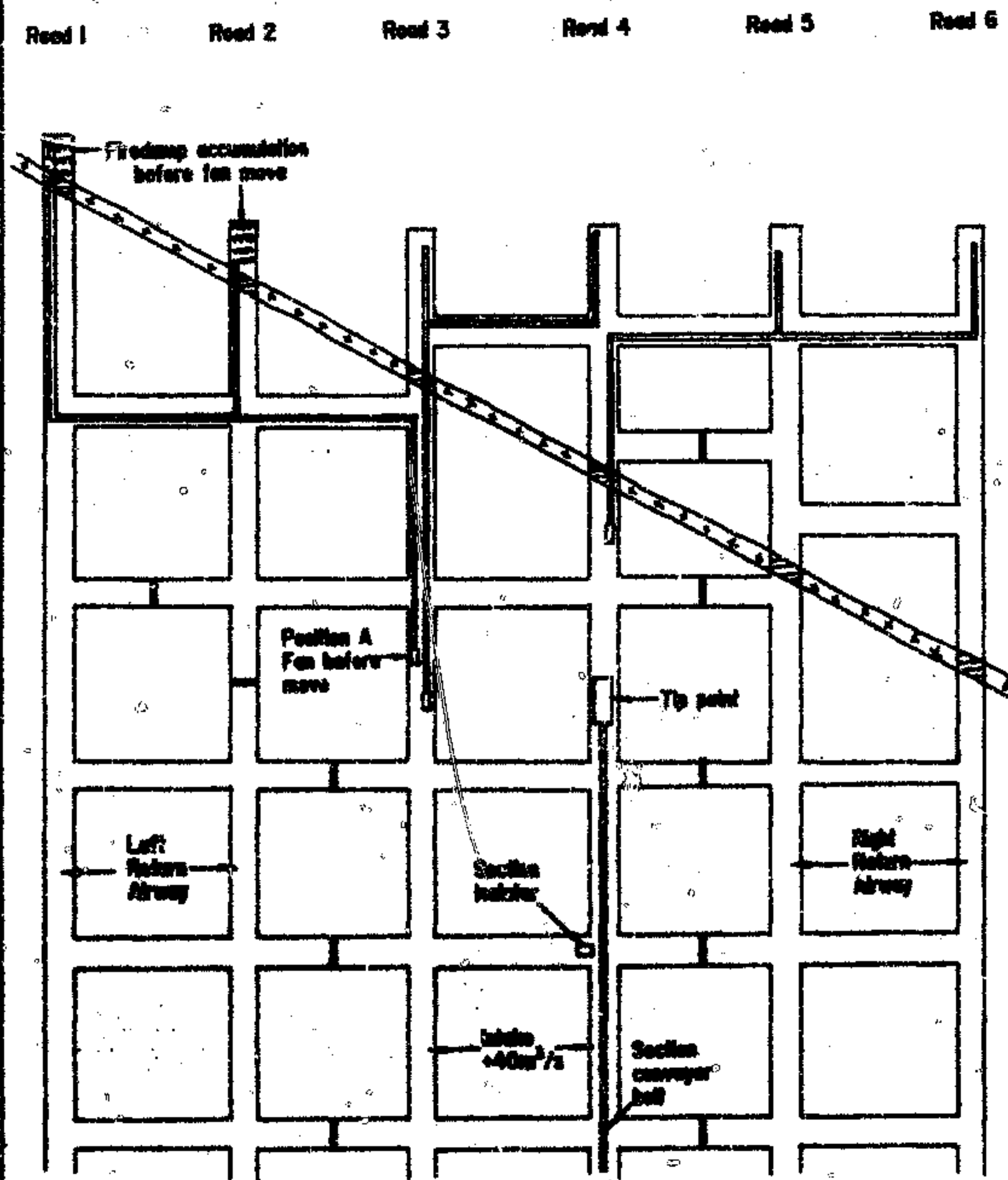


Figure 12

INDUMENI COLLIERY.
PLAN OF SECTION SHOWING
SCENE OF EXPLOSION
Not to scale

15. Electricians should be equipped with methanometers to test for methane in areas where they work. This provides a double safeguard for methane detection. Provisions fo. the issuing of automatic methane warning devices to users of electrical apparatus and other selected categories of workers should be made.

16. No flameproof equipment which has been repaired on surface, for instance an auxiliary fan, should be returned underground until it has been declared to be flameproof by a suitably qualified and experienced Engineer. Procedures geared to the specific inspection of flameproof equipment should be standard practice on every colliery. Such examination should be recorded in a log book provided for this purpose.

17. The path to earth when an electrical fault occurs should not be allowed to follow a route to the roof of the workings in a gassy seam. A methane explosion occurred in the face of a gassy pit when a flash or arc took place in an auxiliary fan starter. The electric leak travelled along a suspended wire and flashed across a roofbolt washer igniting the methane/air mixture.

The ignition occurred when a force fan which had been moved to a planned new position was started. No casualties were recorded.

Figure 12 shows the layout of the section of the mine workings where the incident occurred.

The section was a conventionally mechanised development section working on a double shift basis. A 2,5 m thick dolerite dyke had been encountered and all the roads were through the dyke and in clean coal. Nos. 1 and 2 roads had not yet advanced to a point where a holing could be effected between them in good ground conditions.

Heavier than usual firedamp emission was normal when the first cut went into clean coal on the inbye side of a dyke and usually persisted until a holing across between companion roads was effected enabling the through ventilation to be brought forward. The layout of ventilation arrangements in the section was as shown on Figure 12 with the intake splitting left and right to the return airway on either side. Total intake quantity was + 40,0 m^3/second. Faces were ventilated using 37 kW electrically powered force fans and 760 mm diameter flexible, flame retardant, antistatic ventilation tubing. This tubing was suspended along a length of No. 8 galvanised wire secured to the support roofbolt washers at 1,2 m intervals. The ends of the ventilation pipes were kept to within 5 metres of every face at all times. Fans were interlocked with the section isolator so that the section isolator could not be switched on unless all connected fans were running.

Figure 13 shows the left hand side of the section drawn to a larger scale. Roadways 1 and 2 are relevant. The fan ventilating Nos. 1 and 2 roads was standing at Point A with the ventilation columns in the positions as shown. Brick ventilation stoppings were up to date with the last one being completed on Wednesday end of dayshift. Because of the dyke, the line of stoppings had been offset to keep the cross ventilation as close to the face as possible. With completion of the last stopping, arrangements were made for the fan serving Roads 1 and 2 to be moved forward to Point B, and so at the end of the Wednesday nightshift the Electrician stayed on after shift to put in a length of cable in the feed to the fan so that the fan could be moved forward on the following morning shift.

Putting in the length of cable meant stopping the fan, disengaging the interlocks with the section isolator and the figure of eight coiling of the cable so that the move could be done without interfering with the rest of the section. The No.8 gauge suspension wire which was still there from previous ventilation pipe installations was attached to the usual hitch point on the top of the fan housing. This wire extends angled down from the roof to the fan to carry the ventilation column over the span. See Figure 14.

The ventilation piping along the original route to the intersection at Point C was taken down and the section from Point C to the fan in its new position was installed. See Figure 13. The Shiftboss checked and found that there was a good flow of air passing the new fan site and that the atmosphere around the fan was clear of methane. He instructed the Electrician to put on the power and then depressed the start button at the same time checking the gland. The fan started but immediately tripped all the power in the section because the Electrician had re-engaged the interlock at the section isolator. He turned to the crew supervisor to send for the Electrician to show him the fault and saw the crew supervisor pointing excitedly towards the face of Road No. 2. The whole face was burning right back along the roof to the intersection.

It was established that the cause of the crackling sound and the smoke from the gland was what was termed a "hot" connection of one of the phases in the starter, and that a flash had occurred to the casing. The only other fault that could be found was that there was a rubber seal missing in the gland where the feed cable entered the starter. An independent specialist in earth faults and stray currents was employed to investigate the set up underground. Figure 15 indicates the set up.

All that could be established was that no electrical path to earth could be found at any of the roofbolts to which the wire was suspended except the last one in the face. All the other points were found to be rusty and non-conductive.

11. With electrically powered rubber tyred equipment the risk of electrocution is high if a fault occurs to the frame of the machine and the frame is inadequately earthed.

It is generally accepted that touch potentials of less than 50 volts are unlikely to be lethal particularly if the earth fault is cleared rapidly.

The use of high impedance neutral earth restrictions, sensitive fast pilot circuits for monitoring th earth bond and sensitive fast earth leakage protection make it possible to limit touch potentials to safe values on 950 volt systems. At higher voltages it becomes more difficult.

12. Since underground work is less popular amongst Electricians than surface work, it follows that when shortages of Electricians occur on a mine, underground will suffer most; it is thus a fact that as inexperienced Electricians are employed there is a temptation to short circuit the training and induction procedures and deploy these Electricians underground as soon as possible. See Chapter 2 for a suggested induction programme for Electricians and procedures for re-establishing power to underground sections after a power trip or shut down.

13. All power feed cables should be suspended from the roof in a neat manner where they can be periodically examined. Dangerous roof and sidewalls should be made safe immediately to prevent falls from damaging such cables.

14. Specific procedures should be laid down for the isolation of power to areas which have been temporarily stopped; if for any reason the power has to remain on in this area, special rules/procedures should be enforced governing the use of switchgear and machines in the area. Initial inspections for methane build-ups and ventilation arrangements should be made.

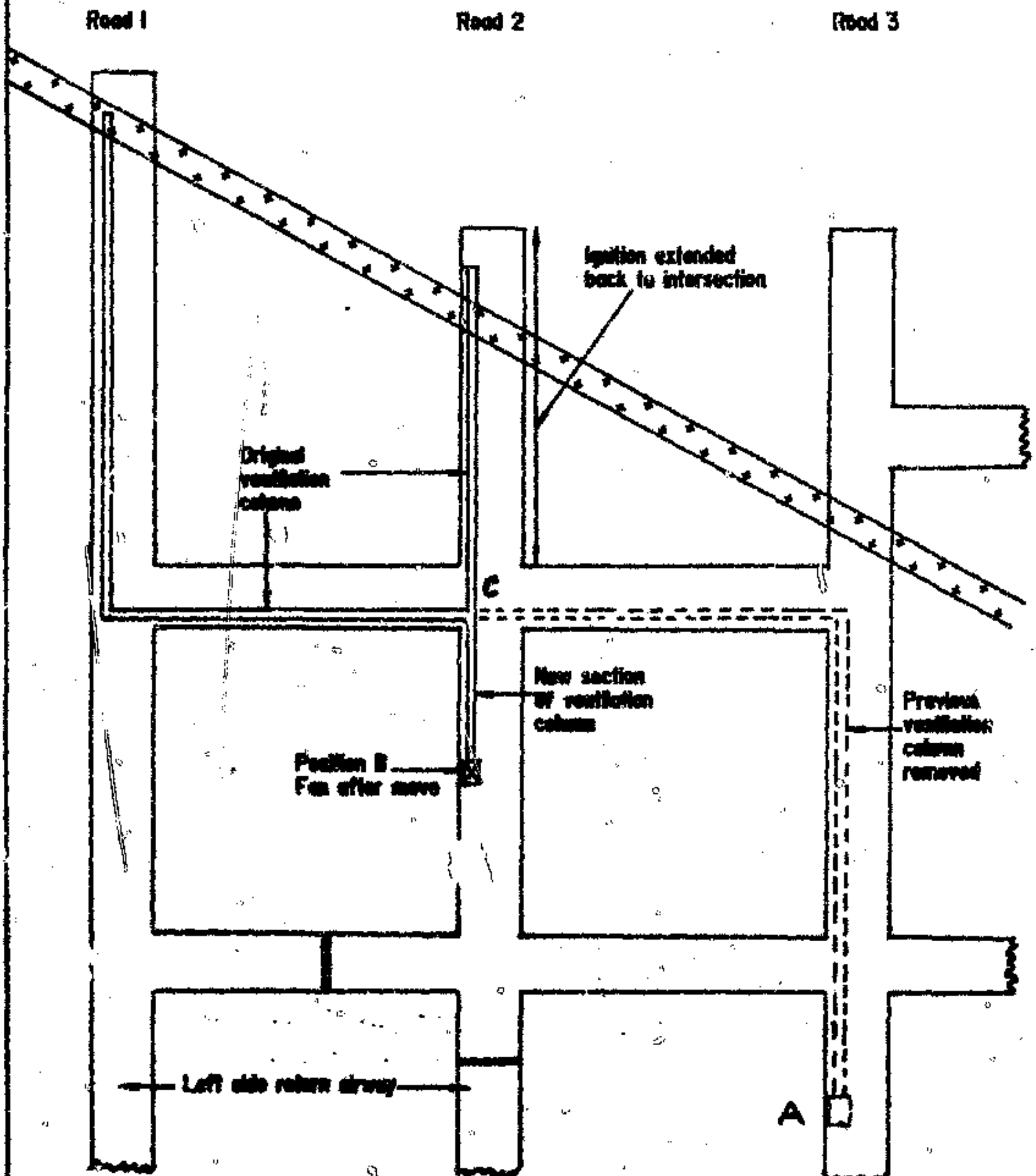


Figure 13

INDUMENI COLLIERY.
DETAIL PLAN OF FACES
WHERE EXPLOSION OCCURED
Not to scale

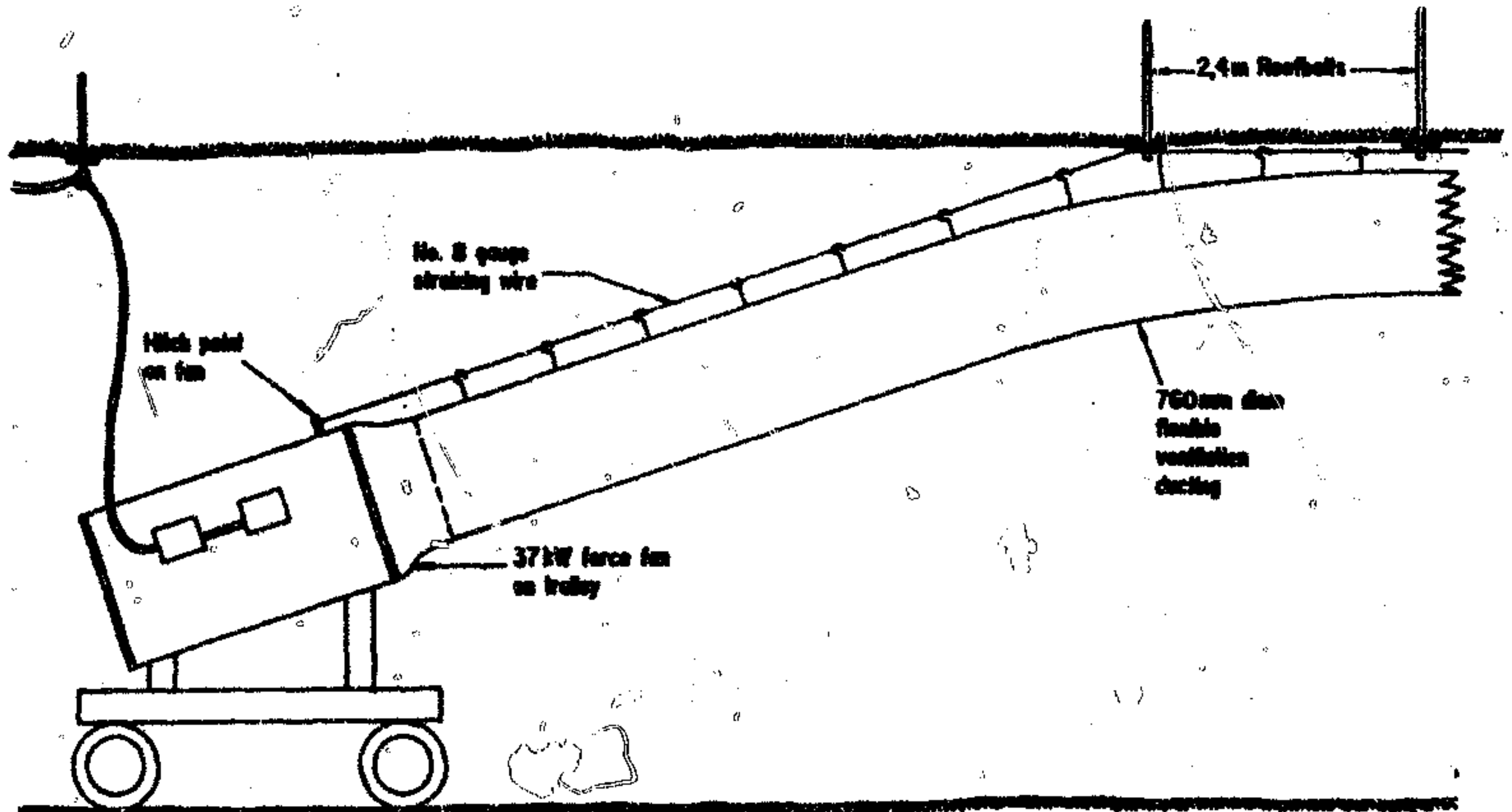


Figure 14

INDUMENI COLLIERY
SECTION THROUGH ROAD No. 2
SHOWING FAN AND DUCTING INSTALLATION

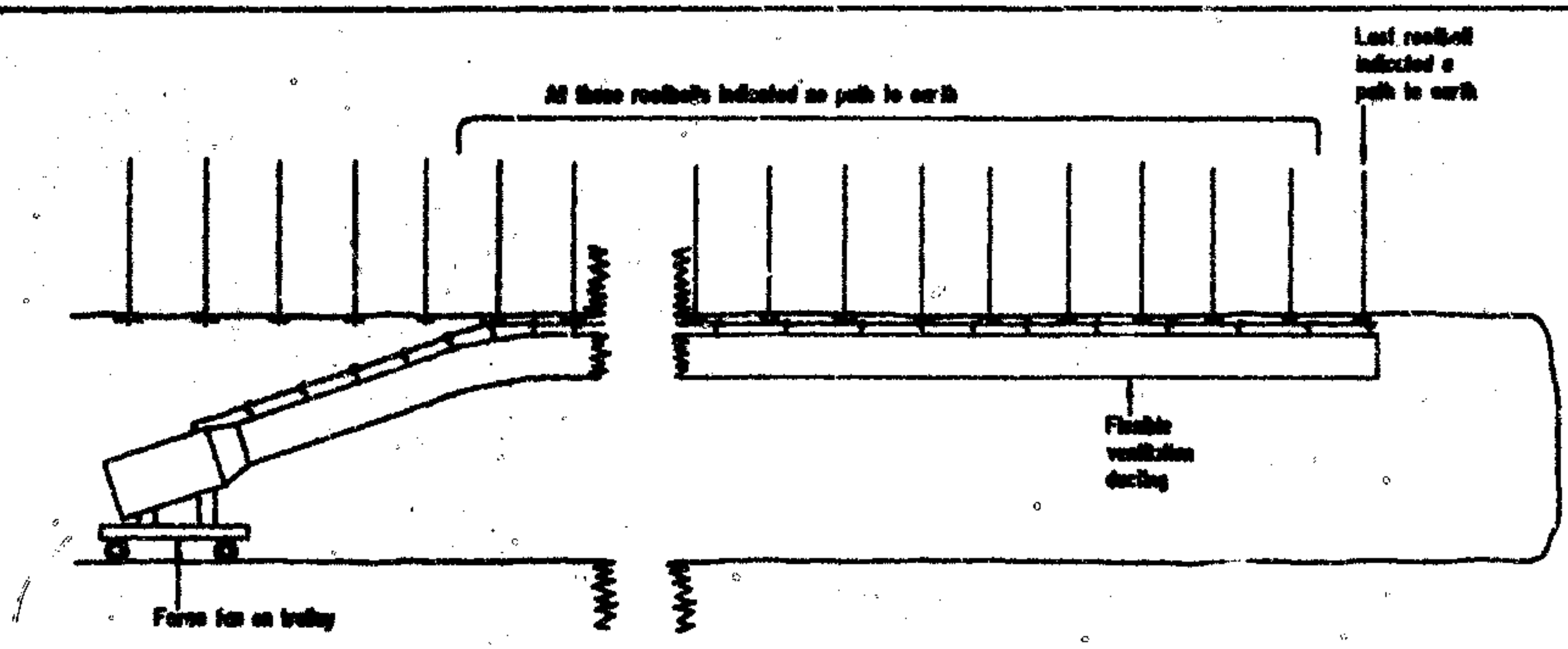


Figure 15

INDUMENI COLLIERY
SECTION THROUGH ROAD No. 2
SHOWING DUCTING ATTACHED TO ROOFBOLTS

The final conclusion was that when the flash occurred in the fan starter, electricity travelled along the suspended wire and flashed across either at the roofbolt washer or inside the roofbolt hole thus igniting the gas.

When electric equipment and cables are moved the system should be examined for damage particularly to cables and glands.

Strict degassing procedures, as standard practice laid down by the Mine Manager, should be employed when starting a fan where there is an accumulation of methane which stretches back from the face and over the end of the last ventilation pipe in the face. For this specific condition the column should be broken at certain points so that the ventilation pipe can be blown clean especially in the first few lengths from the fan before full degassing commences.

Previous experience with ignitions had led management at this mine to have 2 of the 50 kg fire extinguishers plus 8 of the 12 kg extinguishers in a special fire fighting equipment trolley at the Miner's box. This is in addition to other extinguishers at the various usual installation sites. The advantages of the 50 kg extinguisher are its range (± 12,0 metres) and its longer continuous life when in use. Previous experience has also definitely shown that the speed with which such an ignition can be extinguished is vital. In minutes the whole end can become a raging inferno. In this respect it is vital that Personnel be trained and competent in procedures to adopt in such a situation without delay. The maintenance of fire fighting equipment, the availability of stonedust and of water under adequate pressure must always receive high priority.

18. The author quotes from the Report on the Hlobane disaster in 1983 in which the Deputy Government Mining Engineer states:

"The whole subject of the Supervisor structure on the mines is beyond the scope of this report, but it does seem that a revision of roles and responsibilities is needed. In addition it appears that in this case the Miners and Officials had a rather rudimentary knowledge of ventilation. More emphasis on proper training and retraining of Miners and Officials in ventilation practices is recommended. Consideration should also be given to the possibility of giving some more weight to the reports of the ventilation department. It is not considered necessary at this stage to contemplate additional legislation but if a copy of routine reports was routed directly to the Manager, with another copy to the line Officials, more attention may be given to the reports by the latter persons."

The proper training, education and experience of underground Employees is a thread which runs throughout this project report.

19. Signalling wires or other electrical installations which resort to bare wire conductors should be made intrinsically safe (see definition under item 6.5).

20. Booster fan installations underground which operate at higher quantities of air and pressure than the main fan could cause methane-laden return air to be blown through stoppings into the intake airways where non-flameproof electrical equipment is used. Such an incident was recorded at Springfield Colliery in 1953 and it is depicted in Figure 16.

An alert Shiftboss and Electrician discovered over 3% methane in the fan motor house (A on Figure 15) and closed off all power to the installation immediately. An added danger of possible methane ignition were the non-flameproof electric lights on the haulage and bare signalling wires. Stonedusting was not practiced at the colliery.

f. Pressure

In many of the experiments pressure records were obtained both by manometers placed in the gallery walls and by special gauges placed at right angles to the direction of propagation of the explosion. The latter give a measure of the wind speeds that occur during the explosion. In general two types of pressure waves are observed, one preceding the flame and being a pioneering blast wave with a steeply rising front of short duration (which may amount to a shock wave), the other a longer pulse or pulses having a duration of a fraction of the order of a second or more following the passage of the flame front. The magnitude of the peak pressures of these pulses may vary from a few KPa depending on the speed of propagation. If propagation is detonative, on the other hand, these pressures may considerably exceed 10 000 KPa.

7.3.5 The Mine Atmosphere

The initiation of propagation of a coal dust explosion is not appreciably affected by the normal variations in the mine atmosphere, apart from its content of firedamp. The presence of firedamp increases the explosibility of the road dust seriously, in proportion to the percentage of firedamp in the air. The British experimental findings may be most simply expressed thus: for each 1% of firedamp in the air, the percentage of incombustible matter in the road dust must be increased by one sixth of the percentage of coal dust in the mixture just not inflammable in the absence of gas. Whilst the additional percentage of incombustible content required for each 1% of gas in the air is therefore less the higher the incombustible matter required in the absence of gas, it may not always be appreciated that the weight of the additional incombustible matter needed is greater, as is shown in Table 7.6 (Mason and Wheeler, 1931).

Table 7.6 Effect of presence of firedamp in the air on inflammability of coal dust for two coals of widely differing inflammability

Weight of inert dust to be mixed with unit weight of coal dust to suppress propagation of explosion

	No firedamp	1% firedamp	2% firedamp	Average addition for each 1% firedamp
Coal A	0.6 (38)	0.9 (48)	1.4 (59)	0.4 (10)
Coal B	2.2 (69)	2.9 (74)	3.8 (79)	0.8 (5)

(Percentages of inert content in brackets)

Thus a concentration of firedamp in the general body of the air of a magnitude that is just detectable with a flame lamp (about 1.25%) may have a marked effect on the explosibility of the coal dust.

7.3.6 Explosibility of South African Coals

The value of the explosibility of the various coals is given by the co-efficient of explosibility which is the Kex value.

In general, the higher the Kex value, the lower is the optimum dust concentration. For example, for Kex values up to 70 the optimum dust concentration is at least 450 g/1 while all samples with an optimum concentration of 375 g/1 or below have Kex values of 100 or more. All samples with Kex values of 60 or below refused to explode at a dust concentration as low as 200 g/1.

The results of various coal samples are given in Table 7.7 below.

Table 7.7

No.	Area	Optimum coal dust Concentration g/l	Medium size um	Volatile matter content, raw coal	Maximum Kex-value
1	Utrecht	-	19	7,2	10
2	Area -	725	16	7,0	30
3	Anthracite	600	16	6,7	25
4	Seam	600	17	7,4	10
5	Vryheid	-	16	10,4	50
6	Area -	450	21	10,5	55
7	Anthracite	475	16	11,3	60
8	Seam	-	21	10,1	55
9	Dundee	-	15	22,8	105
10	Area -	-	19	20,5	100
11	Coking	-	18	19,6	90
12	Coal Seam	425	17	24,7	105
13		500	18	21,6	100
14	South Rand	450	21	19,2	70
15	Coalfield -	450	20	22,3	95
16	Main Seam	400	16	20,8	90
17		425	21	20,1	95
18		475	17	22,0	85
19	Vaal Basin	475	16	21,5	95
20	Seams	400	16	22,0	90
21		500	19	23,1	95
22		450	17	23,1	90
23	Vierfontein	375	15	20,1	105
24	Area	475	16	21,8	95
25		525	21	20,0	95
26		-	15	13,8	55

Table 7.7 (Contd.)

No.	Area	Optimum coal dust Concentration g/l	Median size µm	Volatile matter content, raw coal	Maximum Kex-value
27	Witbank	300	20	25,4	120
28	Area -	325	21	26,1	105
29	No. 2 Seam	325	19	26,1	100
30		425	17	21,8	90
31	Witbank	475	18	25,1	95
32	Area -	450	17	23,0	110
33	No. 4 Seam	450	15	21,0	80
34		450	17	22,8	90
35	Witbank	400	16	32,3	150
36	Area -	450	18	21,1	90
37	No. 5 Seam	325	17	27,0	120
38		300	18	33,9	155
39	Vryheid	425	21	19,3	110
40	Area -	425	17	20,5	110
41	Bituminous	450	20	19,0	65
42	Seam	450	20	22,7	115
43	Arnot	450	18	22,7	85
44	Coalfield	475	17	23,4	90
45		475	18	23,2	95
46		450	19	22,7	85
47	Witbank Area -	325	18	21,5	80
48	No. 2 Seam	450	21	27,1	90
49	Witbank Area -	450	16	29,7	140
50	No. 5 Seam	425	17	30,7	150

Table 7.7 (Contd.)

No.	Area	Optimum coal dust Concentration g/1	Medium size um	Volatile matter content, raw coal	Maximum Kex-value
51	tbank Area -	400	16	22,3	80
52	No. 2 Seam	450	21	22,5	90
53	Witbank Area - No. 4 Seam	450	18	22,3	65
54	Witbank Area -	450	18	32,0	125
55	No. 5 Seam	325	21	30,0	115
56		325	21	30,6	130
57		350	17	26,1	130
58	Wankie Coalfield	450	18	23,0	120

7.3.7 The Durban Navigation Collieries No. 2 Pit disaster. 8th October 1926

This was the worst coal dust explosion ever recorded in South Africa. The total night shift labour force (including one man on the downcast shaft bank) of 125 men were killed.

The Durban Navigation Collieries No. 2 Pit

The salient points of the evidence are set out below:

1. A methane explosion, caused by a blown-out shot, occurred in Section 10 some 1 300 metres from the shaft bottom (see Figure 4).

2. This explosion developed into a coal dust explosion which traversed the whole mine.

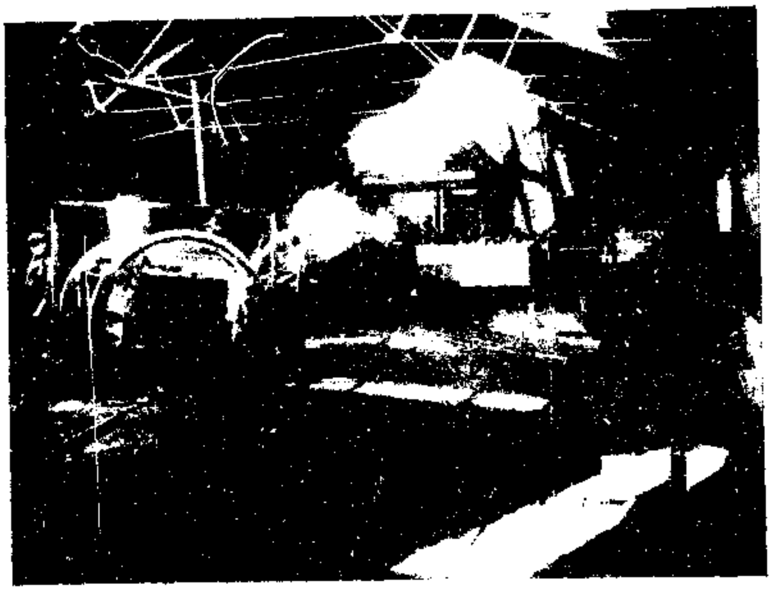
3. The force of the explosion is shown by arrows on Figure 4.

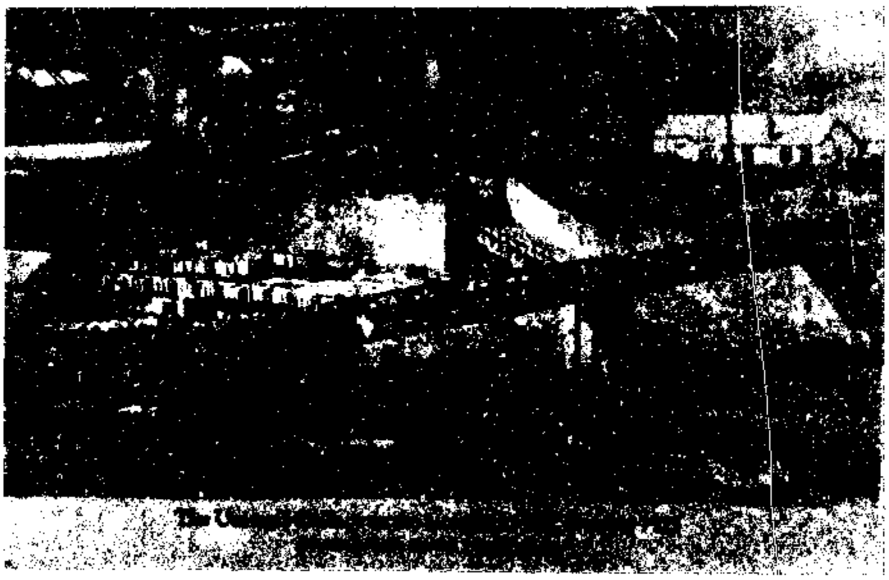
4. The explosion spent itself at the top of the upcast and downcast shafts and in the South main sections.

5. The explosion destroyed the fan drift on the upcast shaft and did considerable damage to the bank of the downcast shaft. The Compound Manager who was about 200 metres from the shaft states in reply to questions at the Official Inquiry:

   "Where were you at the time of the explosion? - In my room.
   Is that far from the Pit? - At Duchen's Store (200 metres from the shaft).
   Did you see anything of the explosion? - I just heard a report.
   Was it a loud report? - It sounded to me like the crack of a rifle.
   No louder? - No. I looked across the square and saw a cloud of dust and smoke.
   You saw no flame? - There was a bit of a flame, yes.
   How high? - It is very difficult to judge.
   Give us some idea? - It was about the height of the top of the headgear."

   The winding engine Driver gave evidence as follows:





The wreckage around the [illegible]

9. There is nothing in the evidence to suggest that, because the colliery was generally dry and dusty, that water was used to allay and settle the dust. Wet cutting was only introduced into the collieries in the mid 1950s. At Hlobane Colliery, two coal dust explosions petered out in wet areas of the sections concerned.

10. Evidence indicates that there was a tendency for dust to be emitted from mine tubs. Without the aid of firedamp it was calculated that 9,01 grammes of dust per cubic metre of ventilation would cause a coal dust explosion. Under the prevailing ventilation conditions, this would equal dust of 3,17 kg per second discharged into the air to maintain a dust explosion. This amounts to 35 dessertspoons of dust per cubic metre - such an amount would produce a dense cloud inpenetrable to the vision. With such a dense cloud it was concluded that a flame or electric arc would have caused a coal dust ignition.

11. That a coal dust explosion travels at high speed and great pressure is confirmed by the injuries to an electrician who was working in the shaft bottom at the time of the explosion.

"What was the cause of death? - The right side of his head was crushed in. Right ribs were fractured. Right femur. Head, face, and neck scorched, skin peeling off hand (palms and fingers) multiple abrasions. Burn on left side of chest and left buttock.
Now, this injury to the head, would you say that considerable force had struck him? - Yes, tremendous force.
You say tremendous force? - Yes.
And the same cause would account for the fractured ribs and the right femur? - Yes.

12. Pressures of up to 1050 KPa have been obtained under experimental conditions in coal dust explosions.

15. The evidence of the Chief Inspector of Mines for Natal regarding the ventilation at the No. 2 Pit is pertinent: Do you not admit it is one of the best ventilated mines in Natal? - It is a splendidly ventilated mine; one of the best in the world, I should say.

16. Evidence of the result of the explosion given by the Lampman is stated below:

"You heard the explosion? - Yes.
Where were you at the time? - Just by the single quarters.
How many metres from the pit head? - Just over 200 metres.
Was it a loud explosion? - Well, it was not too loud, but it was very deep.
Did you hear any sound besides the actual explosion? - I felt a slight tremor.
Did you see any phenomenon after the explosion? - Yes, at he time of the explosion when I locked, I saw flames coming out of the air shaft (upcast shaft).
Mr. Milne: You mean the fan drift? - Yes.
Chairman: That is the only place where you saw a flame? - Yes.
Mr. Milne: Was there any smoke? - The smoke followed it.
At the same place? - Yes, the flame followed it after a very short duration."

17. Good ventilation in a mine is no guarantee that the mine will be safe, as witnessed by the evidence of the Inspector of Mines after the explosion.

"So far as you could see, the ventilation at No. 10 section was sufficient before the explosion. You said the ventilation was quite good? - Oh, yes, when I have been underground I have found the ventilation very good.

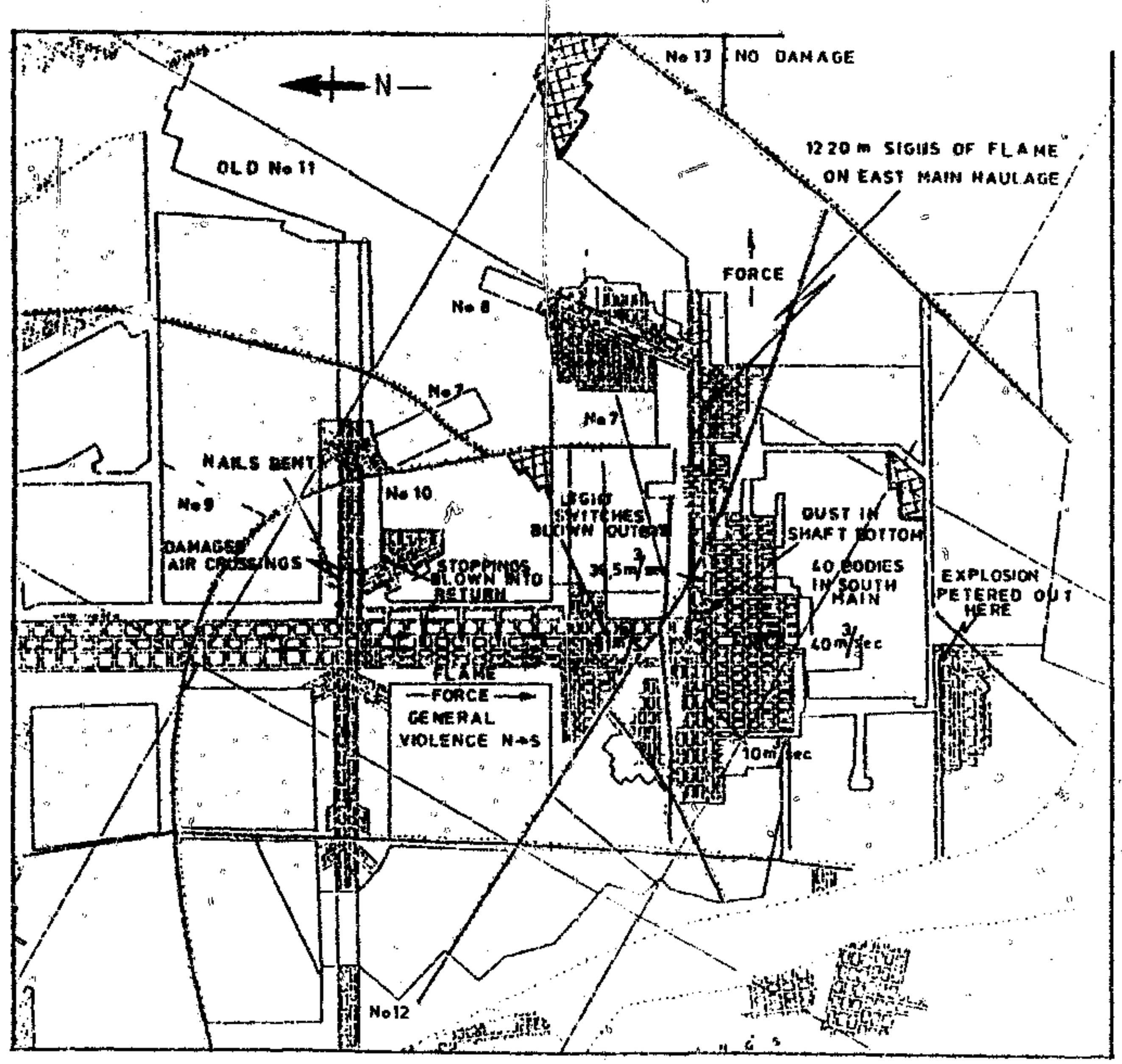


Figure 4
PLAN OF D.N.C No2 WORKINGS
1926

"You saw no smoke? - I was all in darkness and I was covered in dust when the thing took place.
You could see nothing? - No, my view was completely gone.
By the dust? - By the dust and the debris.
That was all caused by the explosion? - Yes.
Were you knocked down? - No.
Your vision was obscured? - Yes.
You saw a flame? - No, I did not.
Were your cages at rest? - Yes, they were.
The one at the top - the right hand one, and one at the bottom, the left hand one? - Yes.
The cage which was at the bottom of the pit at the time of the explosion, what happened to it afterwards, do you know? - I don't know what happened to the cage; it seemed to be jammed and we could not get it away.
Was it still intact? - As far as I know.
Was the rope on the wheel of the cage? - No, it came off.
The Chairman: How long after the explosion were you able to work the cage which was on top of the time of the explosion? - They started to work that as soon as they could get the pit head clear.
What time was that? - I should say about 08h00, between 07h30 and 08h00 but I was not paying any attention to the time; time was no object then."

Photograph No. 2 shows the pit top at the Durban Navigation Collieries No. 1 Pit with the west compartment double deck cage on surface. A coal dust explosion, which travels at a speed between 20 m per second and 2 000 m per second can wreak havoc at the top of such a shaft adding to the difficulties facing management after such an event. Photographs 3 and 4 taken at the Universal Colliery, Sengenhydd, after a coal dust explosion in 1913 support the view of widespread destruction of surface infrastructure following a coal dust explosion.

6. According to evidence, coal dust (together with stonedust) was systematically loaded into bags and taken out of the mine. This practice would have taken place only in roads which were accessible such as the shaft bottom and all main haulages. The return airways which were only 1,4 metres high and totalled some 12 000 metres by 1926 were not swept and cleaned and were also not re-stonedusted. Fine coal dust generated as a result of mining operations (cutting, drilling, blasting, loading and coal transport in tubs) collected in the return airways and was deposited on the roof, floor and sides of these returns. As the pioneer wave of the explosion entered the return airways, this dangerous dust was readily raised into a dense cloud.

What stonedust that did exist in the accessible areas of the mine (the shaft bottom and all main haulages) was being swept up and sent out of the mine thereby further increasing the percentage combustible content of coal dust in these areas.

7. Stonedust sampling and analysis was not carried out with diligence and regularity. The fact that 5 kg of stonedust was applied per ton of coal mined is questioned.

In the return airways, with difficult accessibility, it is likely that any stonedust which did exist was covered by a layer of fine coal dust. Experiments at the explosion tunnel at Klopperbos support the overseas conclusion that such a mixture will give rise to a flame when subjected to an ignition source. No dust ignitions occur when the stonedust and coal dust are mixed.

8. Stonedust dispersed by hand and not by machine is never spread consistently over the whole perimeter of the roadways.

13. Coal dust has been ignited at temperatures less than 600°C.

14. The Chief Inspector of Mines for Natal, under cross examination, stated:

"You know it is a fact that the majority of authorities agree that the coal dust at the face is not so inflammable as the coal dust in the main roads, which have been lying some time and exposed to the drying processes of air? - I do not agree with that statement.

You do not agree with it? - No, I do not.

And you say that the coal dust at the face is the most inflammable you have? - Yes, slightly more liable to explosion than the dust further back from the face.

Now, I want to quote you a passage from Caleb Pamely's Colliery Manager's Handbook, as at Page 875. "The comparatively high temperature which prevails in most underground workings, doubtless assists the fine division of dust particles, dust remaining a long time subjected to this heat, becomes drier, and it may be that it absorbs oxygen or undergoes some chemical change which makes it more dangerous. On the other hand, long exposure to the atmosphere probably causes it to part with some of its volatile gases. It is known that the capacity for absorbing oxygen is greater in freshly worked coal than in that which had lain some time, but it appears that the newly made coal dust is not so inflammable as old dust. The chemical changes which take place would therefore appear to increase the dangerous character of coal dust, but the precise nature of these changes is still unknown and requires investigation"? - Do you agree with that? - Yes, I agree with that at that date. That was in 1898. Pamely says that investigations have to be made and since that time considerable investigations have been made and I have heard the results of some of these investigations. And these investigations are on a small experimental stage.

The Chairman: What is the date of these which you are referred to? - 1907, ten years later. P. Phillips, Bedson;

Dr. Sc. and Mr. Henry Widdas, B.Sc. ignited coal dust clouds in a small chamber by means of an electric current. They tried various types of dust, they soaked them with water, and dried them again, and they selected samples, from the intake, from the returns, from the top of the chamber, and various other places and the final result of these experiments were summed up by Professor Henry Louis in which he said that the experiments recorded would form the basis of any future studies that might be made on the subject; and the data already collected seemed to lead to a number of valuable conclusions. They showed that roadway dust was [illegible] dangerous than dust at the face, and that probably attention would have to be devoted almost exclusively to the dust which was being brought out with freshly hewn coal.

Mr. Crook: What is the date of that? - That is 1908.

And what about the later experiments? - I may qualify the other statements by saying that all kinds of dust were exploded including dust containing over three per cent of moisture.

Have you not looked at any later experiments than that? - But these were definite experiments which were conducted. Under those circumstances, it is certain that my statement as regards those particular dusts was correct.

And your theory, then is that at the mine face this coal dust was more explosive? - Yes, slightly more. It gives the exact amount.

But it was impossible for it to explode without the assistance of the fire damp? - I do not say that fire damp was necessary.

Are you prepared to say that in your opinion this explosion took place without the assistance of fire damp? - I am prepared to say that it could have taken place without the presence of firedamp.

In your opinion, did it take place without the assistance of fire damp? - In my opinion it is more likely to have occurred with the presence of fire damp."

Now, you consider this mine to be a dangerous one, do you not? - I look upon it as a gassy mine, but it has good ventilation and methane is diluted to such an extent that it is difficult to find gas at the face, provided there is nothing wrong with the bratticing or the ventilation system. Do you think it has been made as safe as humanly possible? - I think the mine is well ventilated and stonedusted, too.

7.4 NUMBER AND DATES OF INCIDENTS

1891	Elandslaagte	) These ignitions were probably all
		) caused by the taking of open
1897	Elandslaagte	) lights into workings which
		) contained an inflammable
	Natal Navigation	) atmosphere as a result of
		) inadequate ventilation
1898	Natal Navigation	)
	New Campbell	)
	St. George	)
1899	St. George	4 killed, five injured. Flame lamp opened by a worker
	South African	6 injured. A bonnetted Clanny lamp had been issued without a gauze
	Natal Navigation	Nil injuries. No record of details.
1902	Glencoe	Naked light. No casualties.
	Glencoe	1 injury. Naked light on surface - in the fan drift during fan examination
	Crown	1 injury. Naked light
1903	St. George	2 killed, 2 injured. Naked light
	Ramsay	No casualties. No record of details
	South African	2 killed, 2 injured. Naked light
1904	Ramsay	1 killed, 3 injured. Probably open light
	Elandslaagte	1 injured. Naked light
1905	Crown	1 injured. Naked light
	Glencoe	No record of details. Probably open light
	Glencoe	1 injured. Naked light
	Newcastle Steam	No record of details. No casual
	Dundee Coal	1 killed. No record of details
	South African	1 injured. Naked light

A major feature when studying the incident is the total lack of communication between Supervisors and Workmen. This is supported by the evidence of the Mine Resident Engineer.

"During the week 13th to 18th October, 1941, it was decided to dismantle the 2000 volt switch at the old No. 3 substation and erect it at the new No. 5 substation, Main South West Haulage, Bankfontein Section. As I was under the impression that this type of work could be performed on a Sunday without prior permission having been obtained from the Government Mining Engineer, I instructed the Electrician and the Apprentice Electrician to turn out on Sunday, 26th October, and remove the switch and also connect up the new 2000 volt cable in the sub haulage. At the same time I instructed the Fitter to change the pulley on the main fan on Saturday 25th October, 1941.

On Monday 20th October 1941, I notified the Manager that I intended to do the cable job, that is, the electrical work, in the Main South West Haulage during the coming weekend. I told the Manager that, as I also intended to have the pulley on the main fan changed, I was concerned about using a blow lamp in the haulage whilst the fan was stopped. I understood the Manager to say there was nothing to worry about. I did not ask the Manager to apply for Sunday Labour Permission to do the job as I had been instructed by the Manager to do the job as soon as possible. The Manager did not actually instruct me to do the job on Sunday. Although I had instructed the Fitter to change the pulley on the main fan on Saturday, 25th October, I discovered, after the accident, that he had turned out at 08h30 on Sunday 26th October to do the job. The Fitter had completed the work of changing the pulley by 10h30 and then gave the fan a trial run. The fan was stopped again for minor adjustments, which took about another 15 minutes to complete. The fan started up again at 10h45. The fan is never stopped without permission having first being obtained from the Mine Overseer. I arranged with the Mine Overseer to carry out the alterations to the fan and the cable job during the weekend 25th to 26 October, 1941.

1906	Elandslaagte	18 killed, 2 injured. Outburst of methane from floor. A flame safety lamp was opened to relight others that ha become extinguished
	Elandslaagte	1 killed, 2 injured. Defective safety lamp
1907	Ramsay	1 killed, 4 injured. No record of details
1908	Ramsay	1 killed, 3 injured. No record of details
1909	South African	1 injured. No record of details
1910	Cambrian	2 injured. A flame lamp fitted with a single gauze ignited firedamp in an inadequately ventilated stall inbye of a dyke
1914	S.A. Collieries	1 killed. No record of details.
1915	Hlobane	1 killed. A miner punched five holes in the gauzes of his flame lamp presumably to make it burn more brightly. When testing in a high place in a heading he ignited firedamp and was fatally burned.
	Hattingspruit	1 killed. A worker introduced a naked light into a roof cavity
1919	Durban Navigation Collieries No. 2	1 killed, 3 injured. Explosion took place in the fan drift and extended underground. Coal dust apparently did not play a contributory part
1921	Northfield	1 killed, 1 injured. No record of details
	Hlobane	1 killed. Non-fiery colliery. The miner took his acetylene lamp and his flame safety lamp into a heading, which had just passed through a dyke, and ignited an accumulation of firedamp

1922	Utrecht	2 injured. A worker entered the working face prior to the initial examination by the miner in charge
	Burnside	20 killed. The driving belt of the main fan broke and the fan was not working for an hour and a half. Firedamp built up at the working faces of a section. Most of the flame lamps went out when the workmen returned to the faces when the ventilation current restarted. A worker opened his lamp to relight the others and the explosion resulted
1923	Hlobane	12 killed. Non-fiery colliery. The main fan had been stopped for most of a weekend. On the Monday the miner in charge took an open light into an old end and ignited firedamp that had accumulated
1928	New Tendega	1 killed, 10 injured. No record of details. Probably naked light.
	Tshoba	No record of details. Probably naked light. No casualties.
1936	South Rand Exploration	1 injured. Ignition of methane by open flame lamp
1937	Utrecht	In each of the four cases one worker sustained burn injuries after igniting firedamp at the working face with a naked light. It was after this series of accidents that the mine was declared fiery
1941	Utrecht	Due to the stoppage of an auxiliary fan overnight firedamp accumulated at the working faces of the section served by the fan. When the fan was started at the

		beginning of the next shift a body of inflammable gas was driven out
1941	Consolidated Collieries	2 injured. Methane gas was ignited by a blow lamp in the main road. The mine was stonedusted and there was no evidence to suggest that the explosion was augmented by coal dust
1944	Consolidated Collieries	1 injured. Burning cigarette ignited methane
1944	Tshoba	Non-fiery mine. A worker entered a stall with a naked light prior to examination by miner.
	Hlobane No. 1	57 killed, 6 injured. Although declared a fiery mine, firedamp was of rare occurrence. A heading had passed through a dyke and firedamp was being liberated at the face where an electrical coal cutter was operating. The miner went into the place on an inspection round and the flame of his safety lamp may have been extinguished by an explosion in the lamp. The miner forced the lamp open to adjust the relighter mechanism which he thought to be inoperative. The explosion resulted when he flicked the relighter before screwing the base of the lamp back into the bonnet. No brattice line had been erected to course ventilating air through the heading which must have been filled with an explosive atmosphere. There was insufficient

		evidence to suggest that coal dust contributed to the extent of the explosion
1945	Tshoba	No record of details. No casualties
1953	Northfield	5 killed. Naked light.
1955	Carnavon Anthracite	1 killed, 3 injured. Firedamp was ignited by a cigarette lighter.
1960	Durban Navigation Collieries No. 3	Firedamp was ignited by a faulty safety lamp
1961	Coalbrook Collieries	7 killed. Flint lighter and oxy-acetylene torch ignited methane in an inclined hauling shaft
1962	Ingagane	Ignition on the surface at a pump house of inflammable gas issuing from a borehole
	Wayati	1 killed. Non-fiery colliery. (No firedamp reported during the preceding 35 years). Firedamp ignited by a "cheesa" stick during blasting operations. Some coking of dust was observed which suggested that coal dust may have played a minor role
	Durban Navigation Collieries	Ignition of firedamp. No casualties
1971	H.C.J. Contractors	28 killed, 3 injured. Naked flame.

7.5 DETAILS OF INCIDENTS/PHENOMENON

7.5.1 Consolidated Collieries Limited. 26th October 1941. 2 injured. Methane in haulage road ignited by blow lamp.

Introduction

The colliery is situated in the Ermelo area. The coal seams are found at a depth of between 60 and 85 metres and were known to be gassy. A 2000 volt electric cable was being extended to a new substation site on a Sunday and two Electricians were making a joint in this cable using a blow lamp. The main fan had been stopped for repairs for 3 hours during this period during which time methane built up in the main haulage where the electrical connection was being made. A methane explosion took place seriously injuring the two Electricians.

The Event

The Manager, in his evidence, stated that: "It was reported to me at about 14h00 on Sunday the 26th October, 1941, that the Electrician and Apprentice Electrician had been severely burnt by marsh gas in the Main South West Haulage. On inspecting the scene, I found that the accident had occurred at a point in the haulage where the Electricians were making a joint between two lengths of cable. They had been using a blow lamp for this purpose. The area in which the accident occurred is very disturbed by faults. Some time previous to this accident, gas was found in this area before a connection had been made to No. 3 vertical winze. As the cover in this area varies from 60 metres to 80 metres one can expect to find gas in ends which are not adequately ventilated. When the main ventilation fan is running there is 5 m^3/second passing along this haulage. The fan is situated at the bottom of No. 2 vertical winze and upcasts through it. No. 3 vertical winze is downcast and is situated 2 300 metres from the No. 2 Winze. When I took over the Management of this mine, I realised that sooner or later gas might be encountered. consequently I equipped the mine with flame proof apparatus. Only approved types of flame and electric lamps are allowed in the mine.

Although this mine has not been declared a fiery mine, the regulations relating to fiery mines are being observed. Every working place in the mine is examined at the beginning of each shift for methane and reported on. Only on three occasions in the last 5/1.2 years has gas been found in the workings.

A Miner was delegated to examine the haulage where the work was to be carried out. Figure 5 shows the Main South West haulage, the numerous faults and slips and the position of the No. 3 winze together with the scene of the explosion. The Miner examined the haulage between 08h00 and 09h30 and found no gas; the roof and sides were in order and there was a strong air current in the haulage."

The Electrician, who was injured, gave evidence as follows:

"Sometime during the week 20th to 25th October 1941, the Resident Engineer told me to turn out on Sunday 26th October 1941 to do the joint in the new 2000 volt cable line in the Main South West Haulage. On Saturday 25th October 1941 the Resident Engineer and I discussed what other work had to be done on the Sunday. He instructed me to take my Apprentice underground the following morning and remove a 2000 volt switch from the old No. 3 substation and take it down to the new substation in the South West Haulage. I was also informed that a Miner was going underground to examine for gas and bad roof and to see that everything was in order.

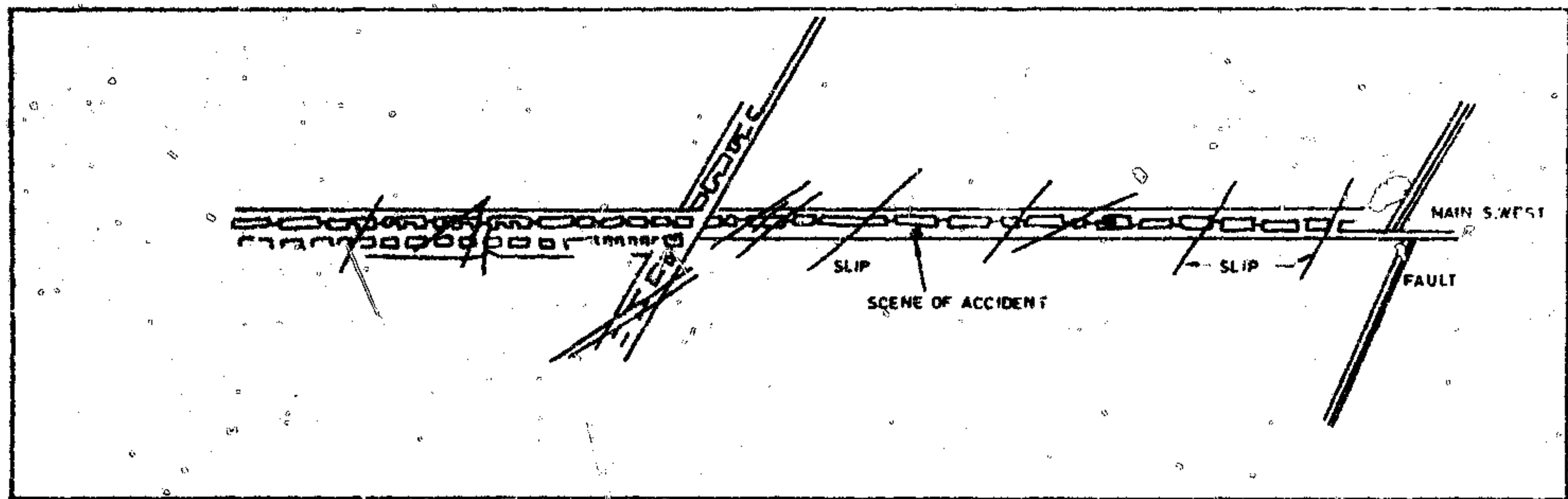


Figure 5

CONSOLIDATED COLLIERIES
PLAN OF ACCIDENT SCENE

On going underground at about 09h00, the Apprentice and I went straight to No. 3 Substation with three workers and took out the switch. The switch was placed on a scotch car and transported along the haulage to the new substation. On the way we met the Miner who informed us that everything was safe. We stopped the scotch car at the point where the junction in the 2000 volt cable had to be made. After preparing the ends of the cable for soldering, the blow lamp was lighted. I started making the joint. It took about 30 minutes to complete the soldering and whilst waiting for this to cool off, we moved back between the tracks to prepare the pitch for the "through box". After the solder had cooled off, I returned to prepare the "through box" for the pitch. The Apprentice was still heating the pitch behind me. Just as I reached the box he shouted look out. I turned around and saw a blue flame on the roof. I jumped across the track and lay flat on the floor. As I was jumping to safety my right arm was severely burnt by the flame.

We had been working with the blow lamp for about 1/1.2 hours before the gas was ignited. I cannot say whether the fan was stopped whilst we were working in the haulage. I did not notice any change in the ventilation current; all I could feel was the heat from the blow lamp.

The flame on the roof seemed to break into two. One portion travelled towards the mouth of the adit and the other towards No. 3 Winze. As the flame travelling towards the mouth of the adit went out after it had travelled a short distance I shouted to the Apprentice to get up and come out. He saw that he was badly injured so I crawled about 10 metres along the haulage to where I could feel and breathe fresh air. I shouted to him again and he got up and came to where I was resting in the fresh air. We then walked along the haulage to the surface. As soon as we got to the surface I placed the Apprentice in my car and rushed him to the doctor."

Had I known that the Fitter had not changed the pulley wheel on the fan on the Saturday and that he intended doing it on the Sunday, I would not have permitted the Electricians to work with a blow lamp in the mine, because from my coal mining experience, I have come to the conclusion that it is not a safe practice in any coal mine to work with blow lamps when the ventilation current is reduced or stopped. I have no doubt in my mind that whilst the fan was stopped a certain amount of gas oozed out of one of the numerous slips in the haulage and accumulated in the cavity in the roof, and when the fan started up again this gas was drawn down and ignited by the flame of the blow lamp.

The Mine Overseer informed me that he could see no reason why a blow lamp could not be used in the main haulage to make a cable joint even if the fan was stopped."

The Fitter who worked on the fan stated that "at about 10h30 on Saturday 25th October, 1941, the Resident Engineer instructed me to change the pulley on the main Fan at the bottom of No. 2 vertical winze. After getting the necessary tools I went underground to do the job. I had previously been told that no job, which would necessitate stopping the fan, could be performed without prior permission being obtained from the Mine Overseer; consequently I went to the Mine Overseer's cabin to ask his permission. He refused to allow the fan to be stopped because there was a shift working underground. I then asked him if he had any objection to the job being done on the next day (Sunday). He said it would be quite alright. I then asked him if it would be alright to stop the fan as there would be a couple of Electricians working underground. He replied that as the Electricians would be working near No. 3 downcast Winze there would be sufficient ventilation and that he could foresee no danger. I turned out at about 08h00 on Sunday 26th October 1941 to change the pulley. I stopped the fan at 08h30. It took me 2 hours to change the pulley. I started up the fan again at 10h30. I stopped it three times between 10h30 and 10h45 for minor adjustments. The fan was fully commissioned again at 10h45. I stayed at the fan until I was satisfied that everything was in order. I went out to the surface at 11h30. I saw the Miner in the Main West haulage; it is possible that he was in the mine when I stopped the fan. I stopped the fan before the Electricians entered the mine. I saw them going along the Main West haulage towards No. 3 winze. I did not speak to them."

Conclusion

At the time of the ignition the Apprentice Electrician was heating pitch with a blow lamp. The pitch caught fire and the blow lamp was withdrawn from the burning pitch. Prior to the pitch catching fire he noticed that the flame of the blow lamp had increased in length. He was not sure whether he had drawn the blow lamp towards the roof when he withdrew it from the burning pitch. The explosion occurred at noon and the Electricians had been working with the blow lamp for at least two hours before the explosion.

The main fan had been stopped for repairs for approximately 2 hours which would have allowed methane to build up in the Main South West haulage.

7.5.2 Northfield Colliery. 28th March 1953. 4 killed and 7 injured. Methane explosions at goaf edge caused by flame from matches.

When firedamp was ignited by a worker striking a match at the goaf edge in Section No. 1, Northfield Colliery, a fiery mine in the Magisterial District of Dundee at 09h15 on Saturday the 28th March, 1953, the resulting ignition fatally burnt four workers and seriously burnt a Learner Miner and six other workers.

History

The section in question, Figure 6, was developed during December 1949 on the bord and pillar system. During development, the area was not gassy. There is a large area of ground immediately

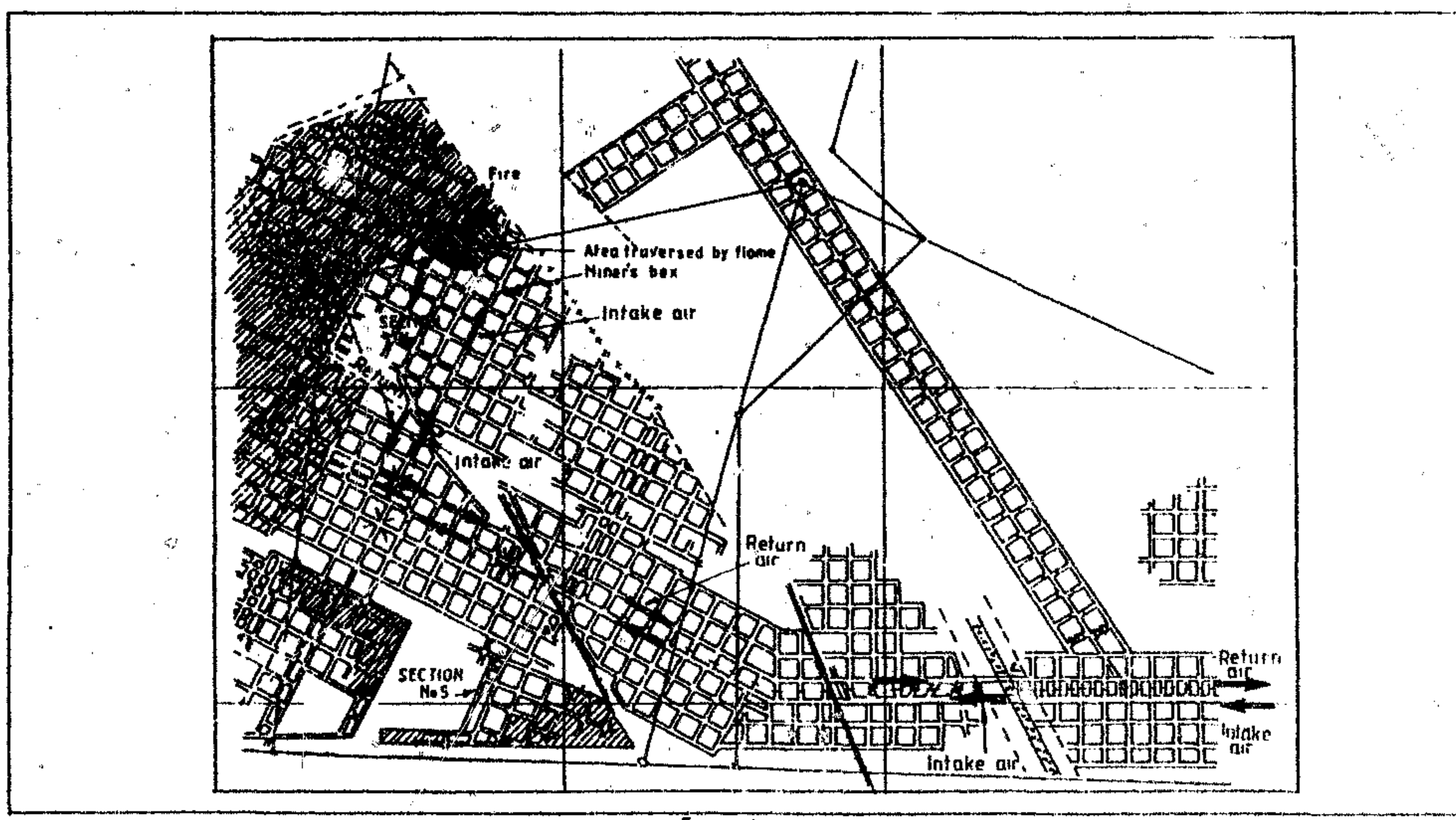


Figure 6

NORTHFIELD COLLIERY - PLAN OF EXPLOSION

beyond the section from which the pillars have been removed. During pillar extraction, for the past few months at least, there had been practically no sign of gas. The Sub Inspector of Mines had visited the section only 12 days before the accident and reported that the ventilation was excellent and no trace of firedamp was found.

Only the bottom seam is mined. The top seam is a metre above it. Both seams are liable to spontaneous combustion. There was a dyke and burnt coal on the right hand side of the section near where the fire appeared to be.

Extent of Explosion

It would not have been wise or practicable to make a detailed study of the scene after the accident as there was smoke and 2% of firedamp in the air some 8 metres outbye from the goaf edge. There was a fire in the goaf. At any moment another explosion could take place. The urgent problem was to ascertain the most suitable points for sealing the area to enable the erection of seals to take place as quickly as possible.

There were no obvious disturbances or signs of burning at brattices and timbers which would indicate the extent of travel of the flame in the section. For this reason it was necessary to make use of the evidence of eye witnesses to determine the approximate area affected. This is shown shaded in red on the plan Figure 6. The ignitions that burnt the various persons were not accompanied by much violence. Persons in some cases were thrown to the ground but were not injured through this cause.

The stonedusting in the section was good. Had this not been so the effects might have been a great deal more serious.

Possible Causes

The coal is liable to spontaneous combustion. A great deal of evidence was led, therefore, to try to establish whether it was not possible that a gobfire in the goaf had ignited gas. There is, however, no evidence of gobstink or heating, nor did anyone complain of any headaches or other ill effects prior to the ignition.

The Manager smelt gobstink in the return air from the section some 6 hours after the ignition and this was confirmed by the Assistant Inspector of Mines.

In view of the evidence of witnesses, the Rail Attendant admitted that he had with him underground, matches, a portion of a match box and a home-made cigarette. It is extremely likely that he lit a match to smoke his cigarette and in doing so, ignited gas at the goaf edge.

The burning gas apparently started a fire somewhere in the goaf. The fire continued to burn giving off great volumes of white smoke and when the oxygen became deficient it gave rise to the odour of gob stink which was not evident immediately after the ignition.

Sequence of Events

The first ignition or ignitions appeared in the form of four air blasts at intervals of a few seconds, each blast being more severe than the last. The first blast was barely noticeable, while the last was strong enough to knock down those in the immediate vicinity.

The Miner, who was only slightly injured, awaited the arrival of the Mine Overseer. On the arrival of the Mine Overseer, they together visited certain of the working faces to see if all persons were out of the workings. Owing to the smoke it was inadvisable to go into some of the working faces.

They then proceeded 350 metres outbye along the haulage and awaited the arrival of the Manager and the General Manager. When joined by these two Officials, they returned to the section and visited other working faces. There was a great deal of smoke but apparently little carbon monoxide as the canaries used and the persons in contact were not affected by this gas at any time.

Apart from the smoke they did not observe any evidence that there had been an ignition. All the brattices were still in order and the ventilation was still good. They decided to go out of the mine and arrange to seal off the section. On their way out along the haulage, they were met by the Deputy Inspector of Mines who returned with them to surface to examine the plans and to discuss the most suitable point for erecting seals. On surface, it was generally agreed that the most suitable place would be the dyke at points B C and D, Figure 6. The plan also showed two other roads entering the area at F and G. It was not known whether there were stoppings at F and G or whether these roads were even accessible.

The Deputy Inspector of Mines and the Manager then proceeded to the section to make certain tests and to enter the old workings at A in order to examine roads F and G. The return airway 350 metres outbye the section contained smoke and 0.0025% of carbon monoxide. There was an odour of gobstink in the return air. At a point about 8 metres back from the goaf in the vicinity of point A on the plan, there was thick smoke and 2% of methane at roof level. Brick stoppings were found at F and G after crawling over falls. In order to effect the initial seal, it was necessary only to wall off at B C and D and to repair the stopping at G.

Some exploration on the left hand side of the section showed that there was a good deal of smoke and the area was uncomfortably hot. The ventilation courses around the section from right to left. All then returned to surface and met the Inspector of Mines, who had recently arrived from Pietermaritzburg. Arrangements were made to prepare for sealing off. The first shift went down at 18h00 to prepare the sites.

29th March 1953. At 12h25 on that day there was an ignition which caused a strong current of air to flow past the wall sites. At 14h05 there was another weaker ignition.

Building of the stoppings commenced at 07h00. At 10h15 when the main road stopping was up some 1 metre there was a severe ignition which knocked everybody down and carried some of them considerable distances along the roadway, fortunately without inflicting any severe injuries. The bricklayers ran out of the mine and all work was suspended.

At 11h15 there was another milder ignition, which, so far as is known, was the last that occurred.

After much discussion and much opposition from the General Manager, it was decided to retreat to a dyke a long way outbye and build walls at points L and K (Figure 6).

The decision was made owing to the extreme danger of continuing building at B C and D and the lack of a suitable site inbye of L and K.

Scores of tubs of building material were then brought back along the haulage to outbye of L and K. Preparation of the wall sites commenced at 21h45.

30th March 1953 The initial seals consisting of 750 mm thick brick walls at K and L were completed at 12h15 on that day. Everyone was then removed from the mine with the exception of an essential Pump Attendant and the Onsetter at the shaft bottom about 1 700 metres outbye of the seals. They were instructed to report any sound from the direction of the seals.

1st April 1953 Twenty four hours later the walls were found to be intact. Stone packs 6 metres thick in one case and 7 metres thick in the other were built against the brick wall and a further 675 mm thick brick wall was built against each stone pack. There was no further incident. Coal getting was permitted once the final walls were completed.

Conclusion

It is clear that one of the workers contravened a Regulation. It is reasonable to conclude that he struck a match to light his cigarette and in doing so ignited the methane, thus resulting in very serious injuries to himself and to others and in the death of four of his colleagues.

In deciding whether or not to take legal proceedings against the Miner, there were two things to be borne in mind:

In favour: the loss of life which resulted from his folly and the constant danger of others doing likewise.

Against: the very severe punishment already inflicted on the Miner by his injuries, for at one time his life was endangered.

It is also clear that there must have been firedamp of sufficient concentration at the point of ignition to ignite and convey the flame into the goaf where the bulk of the gas was almost certainly concentrated.

All the available evidence shows that there was no gas found by Miners or Inspectors for some months past and there was no gas at the time of the Miner's inspection approximately 1.1/2 hours before the ignition. It can only be assumed that firedamp was released by goaf movements and entered the place where the Miner was employed.

Precautions

In addition to the strict precautions taken in the past to prevent persons from entering the mine with contraband, the Manager now offered pithead Guards a bonus of 50 cents for each person found with contraband when about to enter the mine. While this step may be subject to a certain amount of abuse, it was done in the interests of safety.

This incident also highlights the fact that multiple explosions can occur in an area. In many instances it is difficult to sample the atmosphere in the faces in an attempt to anticipate a further explosion. The precautions to be adopted in this case are dealt with in Chapter 5.

7.5.3 Coalbrook Collieries Limited. 25th May 1961. 7 killed Methane explosion in transfer chamber of downcast incline shaft caused by flint lighter and oxy-acetylene torch.

Introduction

The colliery is situated 30 kms south of Vereeniging in the Northern Orange Free State. The two coal seams being worked were relatively deep, gassy and liable to spontaneous combustion. In January 1960 a major pillar collapse entombed and killed 428 Miners on the afternoon shift. Further subsidences in the bottom seam later that year resulted in the workings being sealed off. During reclamation of equipment through the inclined shaft a jammed Sherwin feeder had to be released by cutting girders in the shaft transfer station. A violent methane explosion occurred on the 25th May 1961 killing 7 persons as a result of this act.

The sequence of events which led to this explosion are discussed and analysed.

Stratigraphy

The Vryheid Formation, which overlies the Dwyka Formation, consists of sandstone with shale bands and coal seams in the lower parts (Figure 7). The succession becomes more shaly towards the east where it attains a maximum thickness of 263 m.

Three coal seams are present in the succession and are numbered in ascending order (Figure 7). The Nos. 1 and 2 seams are very intimately associated and are separated by a sandy shale parting ranging from 0,3 m to 8,5 m in thickness.

In the deeper parts of the Coalbrook Basin a thin coal seam, of up to 1,5 m in thickness, is developed locally approximately 1 m below the Number 1 Seam. This coal seam rests directly on Dwyka diamictite.

The average parting thickness between the No. 2 Seam and the No. 3 Coal Unit is 20 metres and consists of interlayered sandstones and shales.

Structure

Over most of the area the coal seams are relatively flat except for minor undulations. An increase in gradients occurs towards the edges of the Coalbrook Basin and is accompanied by the thinning and eventual pinching out of the seams against the pre-Karoo ridges.

Coal Quality

The quality of the coal, on an air dry basis, is shown in Table 7.8.

Table 7.8. Coalbrook Basin: Coal Quality

	Number 1 Seam	Number 2 Seam	Number 3 Seam
Moisture (%)	5,6	5,8	4,3
Ash (%)	24,3	28,9	29,3
Volatile matter (%)	23,4	21,3	19,8
Fixed carbon (%)	46,7	44,0	46,6
Calorific value (MJ/kg)	21,7	19,6	19,9

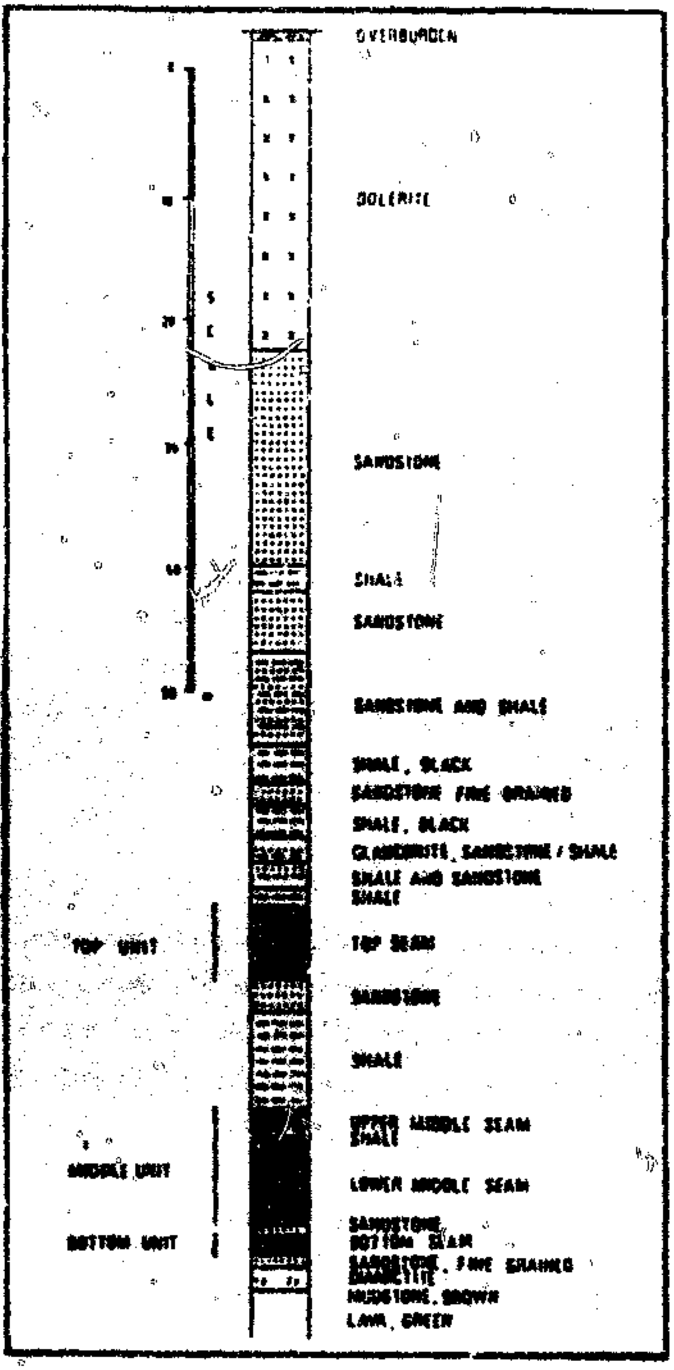


Figure 7
COALBROOK COLLIERY
STRATIGRAPHIC SECTION

Shaft System

The main coal hauling incline shaft (where the explosion occurred) was 24 m^2 in dimension by 500 metres in length at a grade of 1:3. At the halfway point in the sheft was a horizontal chamber which housed the conveyor drives for the first 250 metres inclined lift. This shaft together with a 20 m^2 vertical man and material shaft were downcast. A vertical shaft was used as the upcast and on which was mounted a double inlet Sirocco fan. Dampers were available to reduce the volume of air handled.

Normal air velocity down the inclined shaft was 10 metres per second.

The inclined shaft was concrete lined on the sides and roof but gaps up to 0.3 cms existed between this lining and the normal sides and roof of the shaft.

The Incident

After the major collapse in the No. 2 seam workings in January 1960, severe pillar crush was experienced in the vicinity of the incline shaft. Following a heating in No. 2 seam, the area was sealed off at the beginning of January 1961 and reclamation of equipment was undertaken via the inclined shaft using the shaft trolley. Evidence is that the Mines Department were not informed of the imminent closure of the shaft and the withdrawal of equipment. The workings in the upper No. 3 seam were kept open and, unknown to most mine Employees, an "angle drift" of 1:5 connected the No. 2 seam workings with the upper No. 3 seam workings. The general arrangement of the shaft is shown in Figure 8.

Most underground equipment had been reclaimed and hauled out of the incline shaft by the beginning of 1961 and the No. 2 seam workings had been sealed off near the inclined shaft bottom.

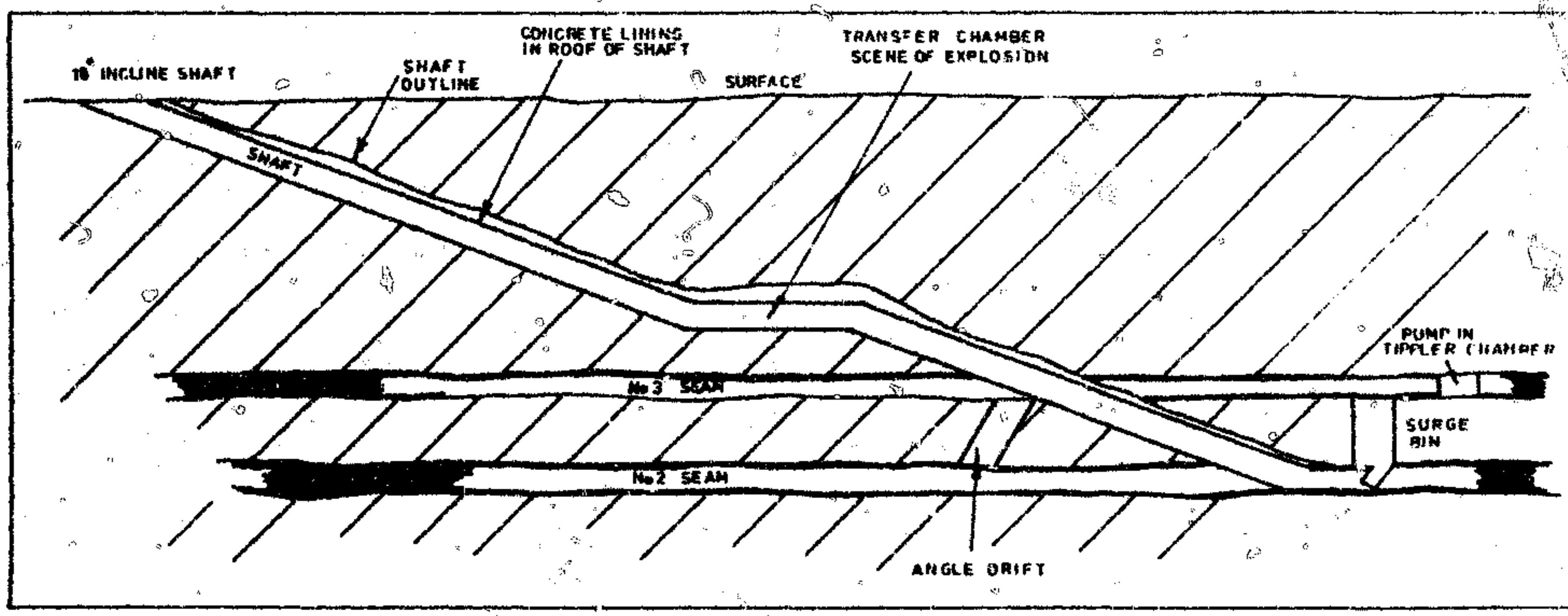


Figure 8
COALBROOK COLLIERY
SECTION THROUGH INCLINE SHAFT
SHOWING SCENE OF EXPLOSION IN TRANSFER CHAMBER

A series of events now took place which finally led to the methane explosion caused by a revolver type igniter or the flame from an acetylene torch used to cut a roof girder in the inclined shaft transfer chamber.

i. Bumping on surface as a result of pillar failure in the old No. 2 seam workings in the vicinity of the inclined shaft bottom.

ii. In April 1961 heating was discovered behind the seals in the shaft bottom in No. 2 seam. Ventilation was breathing through the main seal in and out and high concentrations of methane (in excess of 5%) were being emitted from behind the stoppings.

iii. The air velocity down the inclined shaft was 10 metres per second which was sufficient to remove the methane emissions from the leaking seals in No. 2 seam.

iv. An oxy-acetylene torch was used on several occasions, before the 25th May 1961, to cut steel structures and machines which were being removed via the shaft. Surface guards at the shaft top searched all persons for contraband and tests were conducted for methane by competent persons prior to using the oxy-acetylene torch.

v. Three events now occurred which were to lead up to the ignition:

- Methane gas (± 5%) was detected by the Training Officer against the roof of the tippler chamber in No. 3 seam. He apparently removed this gas concentration by waving a jacket in the air pocket. His evidence is as follows:

"I made tests for gas before the explosion up to ( ) 12th May. I made a daily examination at the incline and the top seam but I did not test for gas daily in the incline. I decided on my own when and when not to test.

I found 4% of methane on the tippler level in a recess above the tippler. This recess was an old de-duster plant recess.

The ventilation was very good. I could test it by feel. There was a strong wind. It was a strong breeze, very uncomfortable to work in.

I reported the gas, once in writing to the Manager.

On my last day, I think it was the 20th, I again found gas in the recess. The recess is about 10 metres above the pump. I reported to the Manager in his office after the 1st May. I do not think he knew of the gas before this. We discussed how to get more ventilation to this place. He said I must remove the plates at the bottom of the orepass. All the coal coming from the top seam ran between these plates. The plates were removed by the Welder. They were gone when I got back the next day.

On Saturday 20th I went down the incline. There was no gas except in the recess at the side of the shaft which is on the same level as the tippler. It is 3 metres wide and 1,8 metres high. Above the concrete there is a 1 metre gap to the mine roof. This gas was above the concrete. I was unable to trace the source of the gas. There was no gas at the seals. I checked above the concrete in the shaft. There was no gas.

I do not know who tested for gas when I left. I never saw the Welder test for gas."

- During the two weeks prior to the explosion, the shaft Guards at the top of the incline were removed and hence no searches were undertaken of people entering the inclined shaft. As is usual, from experience, the Supervisor who removed the shaft guards could not be found.

- The Fitter who was working in the shaft reclaiming equipment complained about the high ventilation velocity in the shaft, the dusty and uncomfortable conditions and the extreme cold (May month).

At his request and with the Manager's and Engineer's approval, the fan dampers were closed to 200 mm which reduced the air velocity in the incline shaft to 1,5 metres per second. The Fitter was apparently satisfied with the reduced velocity.

On the 25th May 1961 a Sherwin Feeder from the bottom of the tipplers which was being removed from the mine, had jammed against a roof beam in the incline shaft transfer chamber. At 14h00 the Fitter and 6 Helpers went underground with oxy-acetylene bottles to cut the girder which would then free the jammed feeder. No tests were made for methane and no one challenged the Fitter on surface for contraband. He was also not competent to test for gas.

Upon using the flint lighter at the transfer chamber to light the oxy-acetylene torch a methane explosion took place killing all 7 men and severely damaging the shaft.

The damage to the incline shaft was great.

The shaft collar concrete had been ruptured and had fallen in.

For a distance of about 30 metres the lining was not affected. The walls are brick, the roof concrete and rail reinforced. From 30 metres to the transfer chamber, the roof and sides showed signs of great violence. In most places the roof was down with the walls on top or the walls down with roof on top. The place was a shambles and it was necessary to crawl down over the concrete roof which had fallen. The concrete roof was mostly in large slabs. The rails were bent but not twisted.

At the transfer chamber there were less signs of violence, and although the brick walls were high at this place there was much less damage than higher up. The walls were still standing except for a narrow vertical strip on the East side of the transfer chamber, which had collapsed - a strip 1 metre wide. Tests for gas with a flame lamp at a few places further down and in places behind walls, revealed no methane.

In several places where there had been electric lights, the wiring was examined. In the upper 300 metres of the incline shaft, the rubber showed no signs of heat. Below this point the rubber had deteriorated either due to old age or heat.

In one place in the shaft 20 metres below the transfer chamber, on one side of the shaft, there was a hole in the West wall, one and a half bricks in size and a brick hole on the East side, presumably left at the time of construction. This indicated that the walls were not gas-tight. Further down the shaft the damage appeared to have increased. The walls had fallen in and the concrete roof had come down.

About 50 metres below the transfer station, there was a heavy fall of the natural roof strata going up into the roof for 2 or 3 metres.

The scene at the shaft bottom is described by the Rescue Team:

"On arrival at the 3 seam, we found an entrance East and West with double doors. At the 1st door on the East side, the frame work had shattered and the small steel door was up against the second steel door. The second door was not damaged.

On the West side, the first steel door was bent and would not close properly. The second door was undamaged and formed a good seal.

We then went further down the incline and after a few steps noticed a strong odour. The Inspector of Mines lifted his lamp and commented on the percentage of gas. We saw about 4% - the lamp went out.

We all went through the West doors to the No. 3 seam workings. In the atmosphere in various places, we detected a small cap of 1 to 1.1/2% of CH_4. The percentage of methane then increased. We went into the left return, examined stoppings and tested for gas from time to time. We could not find anything more than a distinct cap of gas. We examined two bleeding pipes to panels. In each case there was negative pressure. This could account for so little gas being present."

Summary

A Fitter and seven workers were killed by an explosion of firedamp in the Coalbrook North Incline Shaft. There were no survivors, therefore this conclusion is built up from circumstantial evidence.

The evidence shows that after the explosion, and presumably before, firedamp was being liberated into the incline at the No. 2 seam horizon.

A fortnight prior to the 25th May, 1961, a high quantity of ventilating air was being downcasted down the incline. A week before the 25th the ventilation velocity was reduced. On the week of the accident, the velocity and quantity of air were still further reduced and indeed there was probably little ventilation down the incline. At the same time the mine atmosphere was relatively warm and therefore light, and would tend to rise up the incline close to the roof.

The conditions were such that firedamp could readily accumulate outside the concrete lining of the shaft and not so readily within the lining and adjacent to the concrete roof. The lining generally was approximately 600 mm clear of the shaft roof and sides.

Evidence had established that there could readily have been an accumulation of firedamp at the transfer chamber, outside the concrete lining or inside the concrete lining or both inside and outside.

The quantity and concentration of firedamp is open to surmise. Two pump Attendants maintain that on their way up the incline at 14h00 on the 25th May 1961, they did not lose their flame in the safety lamp. This means that the percentage of firedamp in the incline and at a height of less than 1 metre above floor level of the incline was less than 5% firedamp.

Again according to the pump Attendants, the Fitter was conscientious enough to test at the shaft bottom for firedamp, and it is therefore not unreasonable to assume that the Fitter also attempted to test for gas at the chamber. There is, however, a peculiarly significant aspect about the Fitter's flame lamp which was found after the accident. The wick was fully screwed up. This is ominously consistent with a determined and unsuccessful attempt to relight the lamp.

The flint striker of a lamp will not relight a wick if:

a. the striker is defective;
b. there is over 3.1/2% firedamp within the lamp; or
c. the lamp gauzes are sooty.

When the Inspector of Mines tested the lamp on surface, in fresh air, it readily relit.

The inference now is that there was so much firedamp in the general body of the air at the chamber that the flint striker could not relight the lamp.

The Fitter was given the task of recovering material from the incline. One piece of material was a Sherwin feeder pan, which was loaded onto a scotch car at the incline bottom and was then pulled, by winch rope, as far as the transfer chamber brow where it jammed against a roof brow girder.

The Fitter, who was apparently in a hurry, went to the Resident Engineer who illegally gave him authority to take underground an acetylene burning set with a striker; the Resident Engineer alleges that he told the Fitter to wait for an official gas test before cutting was started.

The Fitter borrowed a burning set, discarded the set striker but took his own revolver type striker. The set was lowered down the incline by the winch. The Fitter and his workers went with the scotch car and within 30 minutes, there was a violent explosion in the incline.

At the time of the inspection by the Inspector of Machinery, after the explosion, he found the acetylene circuit valves open to such an extent that an acetylene flame could be produced. The striker, in working order, was nearby.

Evidence shows that the acetylene flare or striker more than likely ignited firedamp, fatally burning the Fitter and others. A firedamp explosion then passed down and up the shaft, blasting four workers out at the incline collar - at the same time wrecking the belt drive house on surface.

Liability

Regulation 74(6) clearly prohibited the taking underground of matches or any other appliance for striking a light, such as a flint striker.

At the same time Regulation 166(1) made allowance for conditional exemption. Had the Manager applied to the Mines Department, he would probably have been granted conditional exemption.

The Manager failed to do this, and as implied by the evidence, he apparently introduced his own system which he presumably regarded as equally safe. This included the use of a gas test signature book and an inspection by a Shiftboss. The book appears to have been lost without anyone becoming perturbed, and it was not replaced. It is doubtful if the inspections were meticulously carried out each time a torch was used.

It may be concluded that latterly, if not always, the safety precautions alleged to have been adopted, were observed in a casual manner, so casual that it exculpates the Fitter and inculpates the Engineer and the Manager to the extent that there was criminal negligence.

The Fitter contravened Regulation 74(6) by taking the striker underground but as it was with the consent, if not at the incitement of the Engineer.

The Engineer may have thought that the Manager had exemption from the Mines Department to take a striker underground but his failure to make sure, or to ascertain conditions, and his consent to the Fitter's action, read in conjunction with illegal previous underground cutting tasks in the incline, establishes a clear and repeated breach of Regulation 74(6).

The Manager had knowledge or should have had knowledge of what was going on underground with acetylene plants. He was therefore also guilty of a breach of Regulation 74(6) read with Regulation 156(1).

The fact that a gas inspection book system had been introduced and had been in use for some time, and that cutting had been carried out after the book was lost and not replaced, merely makes the breach of Regulations more inexcusable.

The fact that the Fitter had a box of matches in his possession is a breach of Regulation 74(6). The matches played no part in the explosion but indicate a casual attitude towards Regulation 74(6).

Regulation 10(1) requires the mine workings to be made and kept safe. The Manager knew that there were vast quantities of gas in No. 2 seam workings and in portions of No. 3 seam workings; also that the workings had been taking weight. However, he had reason to believe that he had made and kept the incline safe by circulating a large quantity of ventilating air down the incline and also that the Training Mine Overseer had made periodical checks to see that all was safe.

But the Manager authorised the main fan dampers to be pushed half way in, a not unreasonable procedure, if at the same time extra precautions had been taken by way of careful inspection to ensure that gas did not accumulate. There are no clear indications that such inspections were uniformly made.

However, on the week starting Monday 22nd May, 1961, the dampers were pushed in still further. This action would readily give rise to a precarious condition and some meticulous inspections should have been made to ensure that conditions had not deteriorated.

Although there is no evidence to this effect, the Manager had authorised the dampers to be half shut down but no more, and he may not have known about the dampers being still further shut down.

It is concluded that the workings on the [illegible] May, 1961, had not been kept safe, as required by Regulation 10(1) due to the additional closing down of the fan by pushing the dampers in still further.

If the Manager had no knowledge of this action, then the Engineer contravened Regulation 10(1) read with Regulation 177(2).

If the Manager had knowledge of the additional closing down and had failed to have special inspections made then he contravened Regulation 10(1).

If he did not know what was going on then he contravened Regulation 156(1) and (3) read with Regulation 10(1).

7.5.4 Springfield Colliery. 1989. Nil injuries. The heat from a jammed return idler on a main conveyor ignited paper and cardboard.

Cause of the Fire

Refer to the attached plan (Figure 9) showing the locality and scene of the fire. The fire was most probably caused by friction created by a bottom roller that jammed. The area where the fire occurred has a cross sectional area of 4 m^2 as opposed to the normal 16 m^2. This restriction was caused by poor ground conditions which necessitated the installation of heavy steel and timber support. As a result of this aspect, a high intake air velocity, over a period of time, had blown fine coal and scrap paper off the conveyor belt at that point. The belt was also poorly aligned with the one side of the structure attached to the steel sets. This caused some coal to spill off the belt. All this coal and rubble accumulated below the belt and eventually caused the roller to become jammed.

The heat generated by the friction ignited nearby scrap paper which was blown into the adjacent timber support. Prior to the incident, scrap paper had been placed between these supports and this paper caught alight and, in turn, set some of the timber

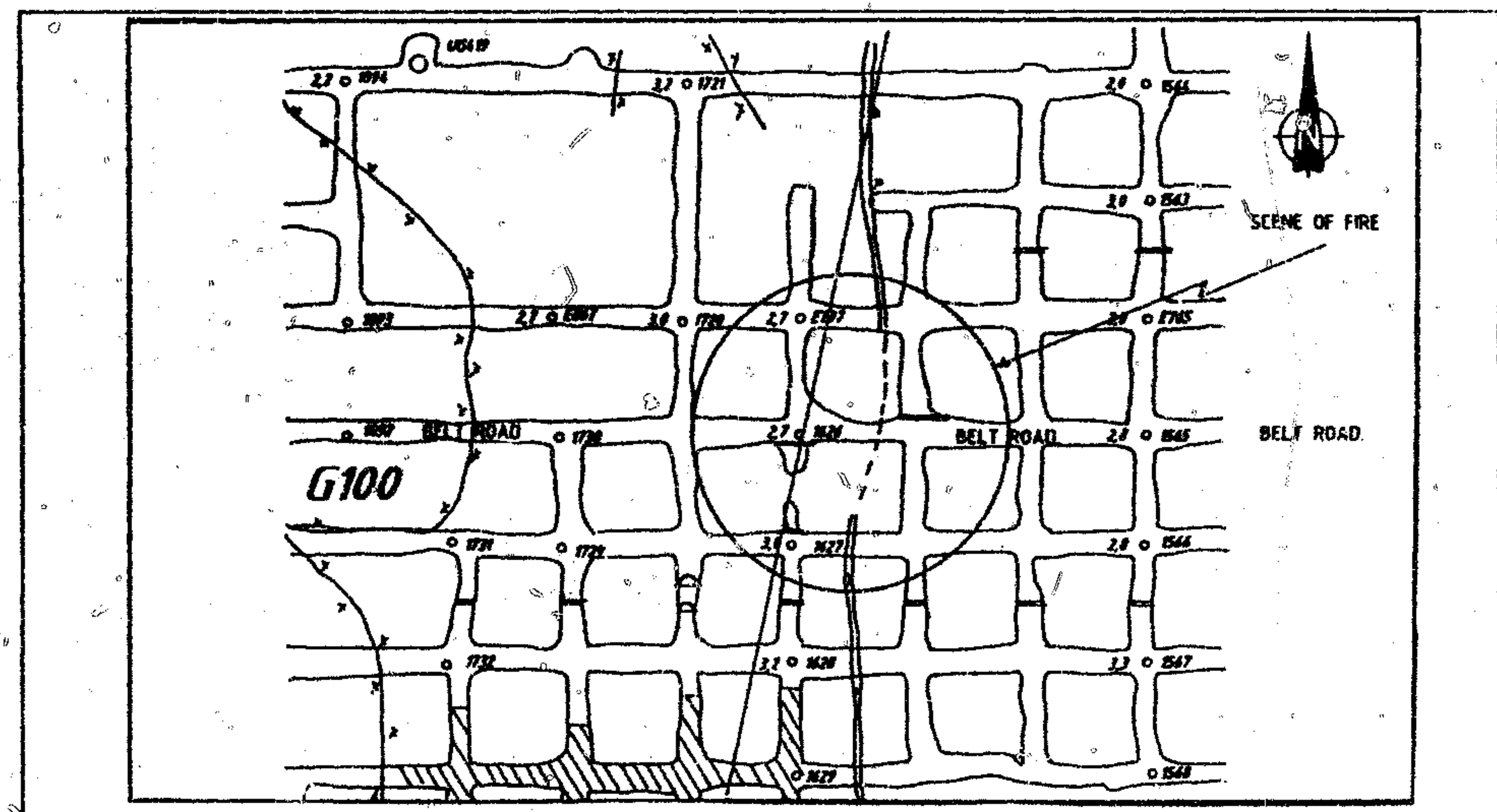


Figure 9
SPRINGFIELD COLLIERY
LOCALITY PLAN OF FIRE

sets on fire. Figures 10, 11 and 12 show details of the affected area.

Detection of the Fire

The fire was detected by a Shift Overseer and other personnel travelling towards the shaft bottom. The persons concerned were alerted by the smell of smoke when they reached section G480. Visibility was good and they continued to proceed towards the origin of the smoke and eventually reached the G201 area where the fire was discovered in the belt road.

The belt Attendant at the belt drive inbye of the fire (W3 belt) noticed burning material coming towards him on the return side of the W2 belt. At that stage he also noticed smoke. He phoned his colleague at W2 belt drive who immediately stopped the belt. Both these workers then started patrolling the W2 belt from either side to establish where the fire was.

They reached the scene of the fire at the same time as the Shift Overseer.

The newly installed Davmon Central Gas monitoring system indicated an increase in carbon monoxide in the G100 return airway, but did not reach the alarm level (which was set at 20 parts per million) due to accidental interruption of the system by untrained personnel. A graph obtained through the system is attached as Figure 13.

Immediate action taken

Workers inbye of the fire were warned of the event, but informed that no immediate danger existed. The surface control room was informed and senior personnel were alerted.

Fire extinguishers and fire hoses were transported to the area by tractor and trailer. The Shift Overseer and other personnel

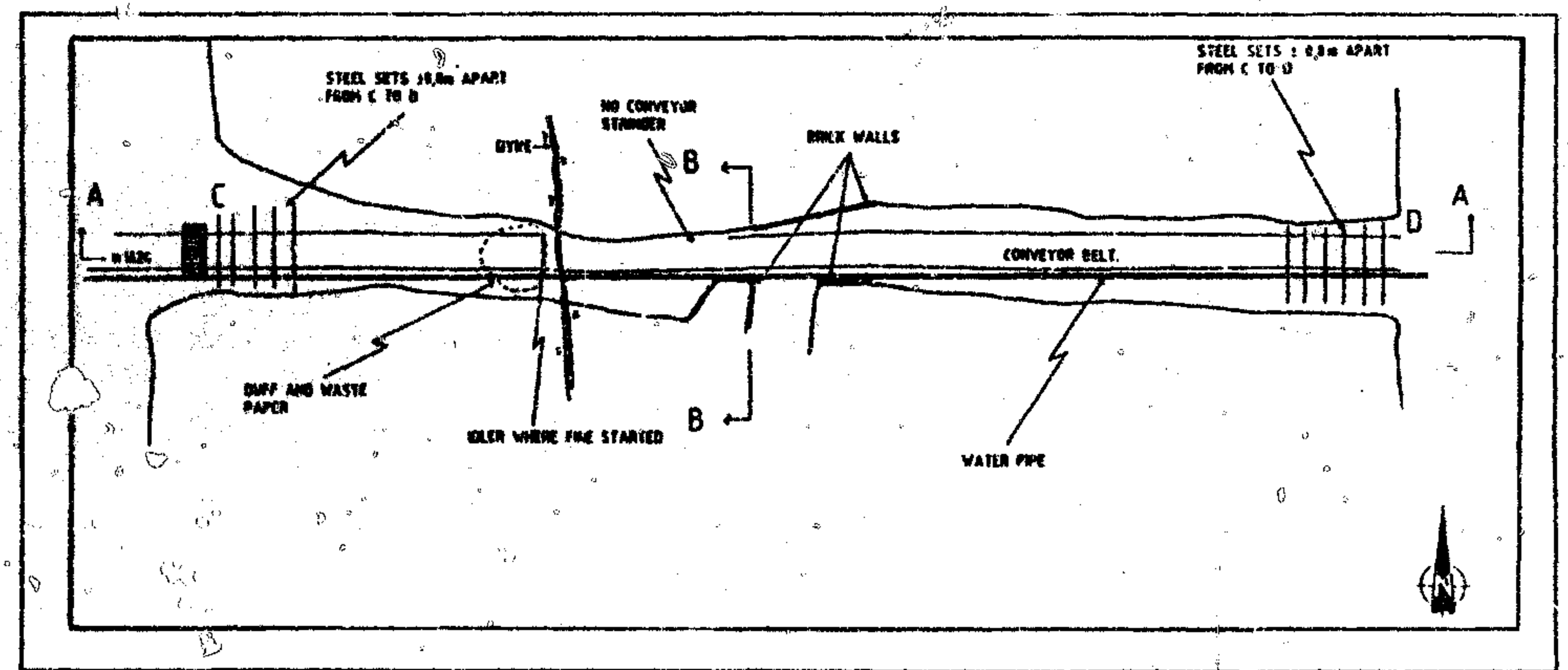


Figure 10

SPRINGFIELD COLLIERY
PLAN OF FIRE SCENE

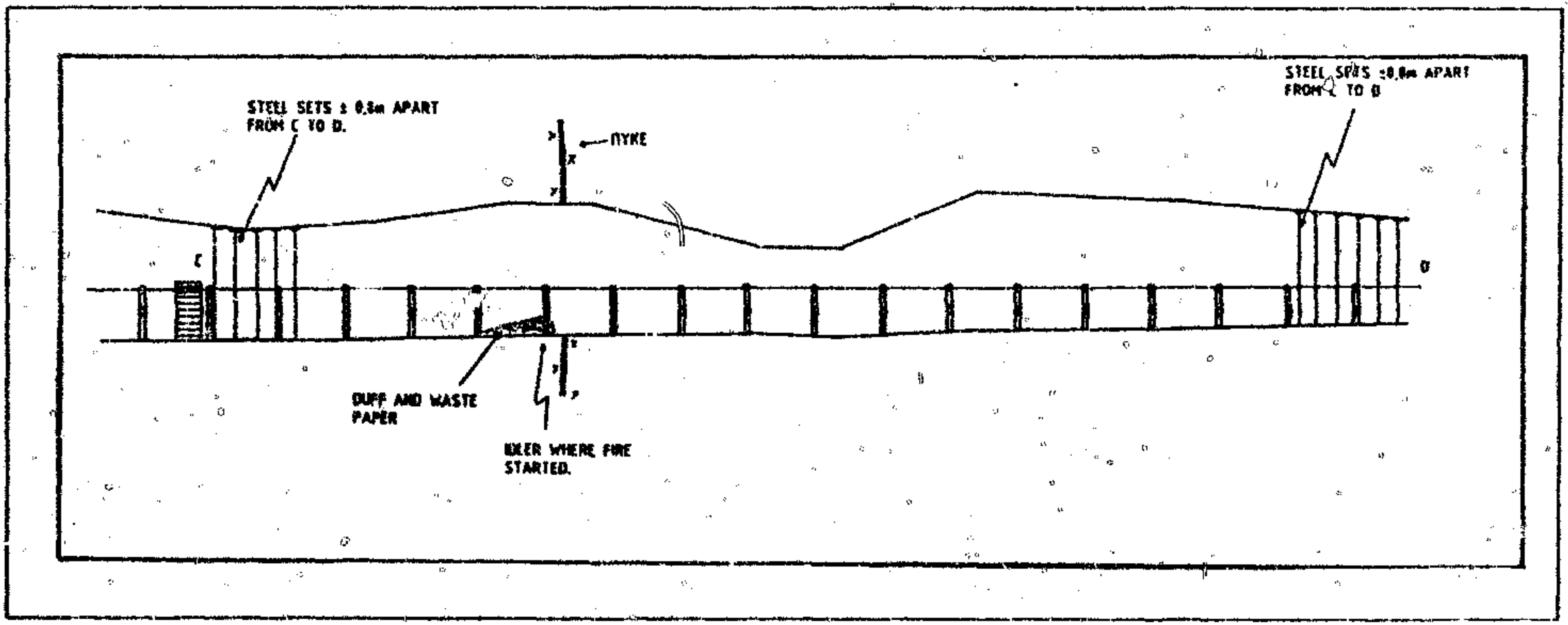


Figure 11
SPRINGFIELD COLLIERY
SECTION THROUGH FIRE AREA

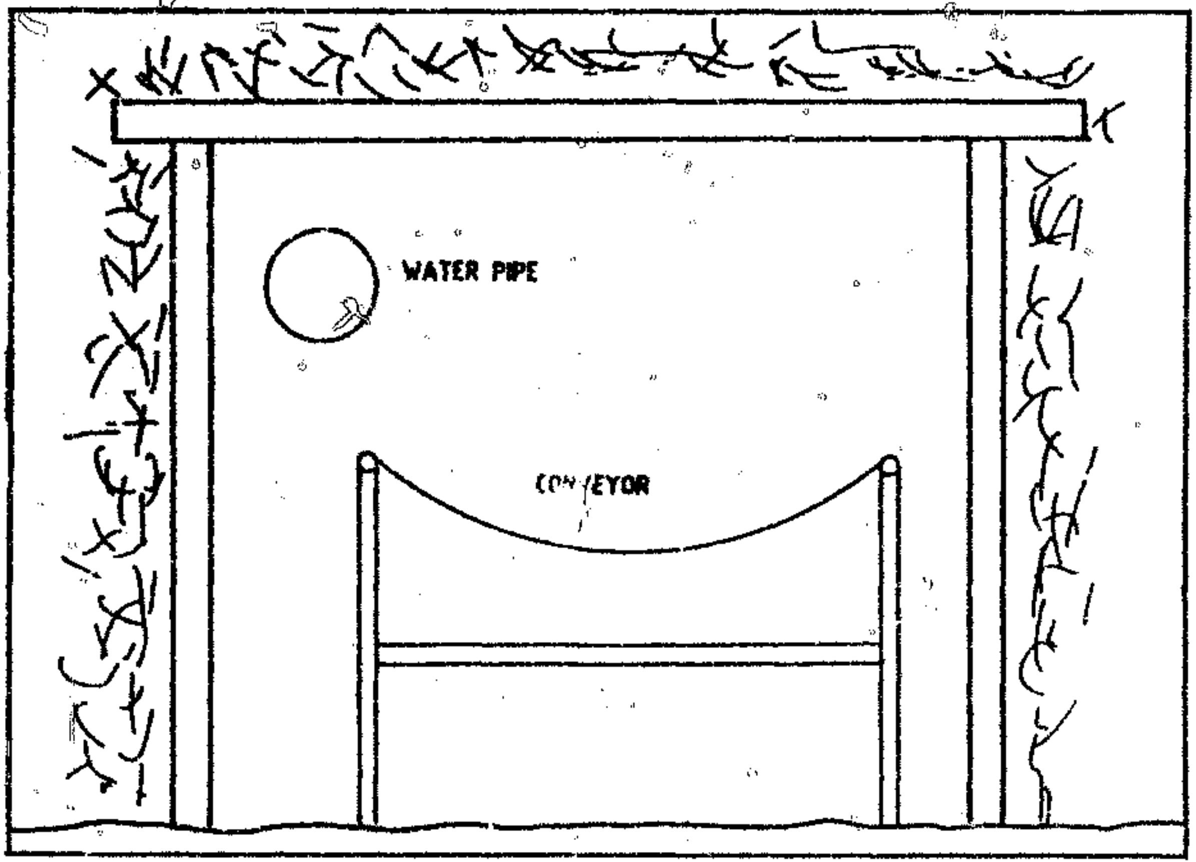


Figure 12
SPRINGFIELD COLLIERY
SECTION THROUGH CONVEYOR

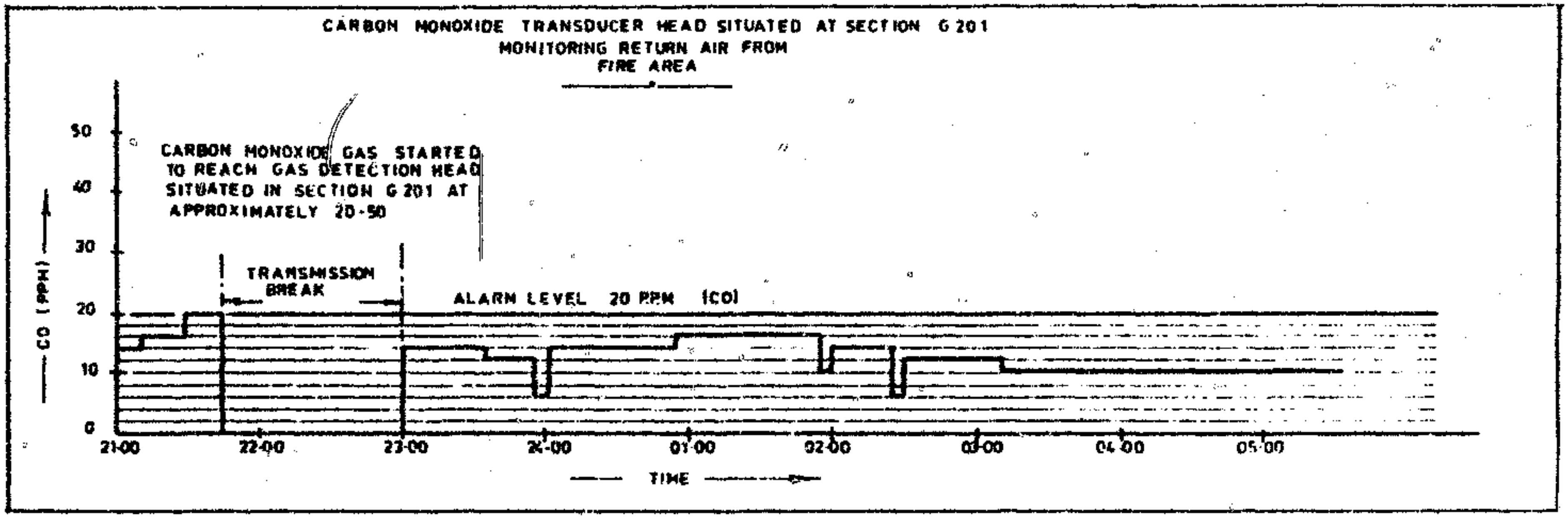


Figure 13

GRAPH FROM SPRINGFIELD GAS DETECTION SYSTEM extinguished the fire, using 20 fire extinguishers and water from nearby fire hydrants. Fortunately, the fire was discovered in an early stage and was extinguished within 20 minutes of being located.

Follow-up Measures

On inspection, it was found that the area behind the timber support was used to throw paper and other material away which was found on the belt. The area was cleaned of any unnecessary combustible material. A thorough belt inspection was carried out in the rest of the mine to identify areas where a similar incident might occur.

Some of the restrictions have been removed to assist in realigning the belt and installation of the stringers as indicated on the plan. This now prevents the spillage of coal.

A non-standard fire hydrant was found and was replaced. All other hydrants in the mine have been checked to ascertain that they conformed to standards.

Another main intake airway was developed to ensure a reduction of air velocity in the belt road.

Proper training of all staff concerned in the use of the Davmon Central Gas monitoring system was done, together with adequate instructions and procedures displayed in the control room where the monitor is installed.

Conclusion

This incident has highlighted the necessity for prompt action and the need to follow the correct procedures in an emergency.

The actions of the Shift Overseer and Belt Attendants indicated that emergency training had been effective, as no panic had arisen and the fire was extinguished quickly and efficiently.

Once again the attention was drawn to:

- regular belt patrols by senior personnel
- special checking of 'problem' areas
- ensuring that no non-standard equipment is used, and
- the importance of correct procedure and training.

7.6 SUMMARY OF THE PRACTICES REVIEWED

- Mines regarded as safe, well ventilated and well managed may be subjected to the most violent disasters (No. 2 Pit of the Durban Navigation Collieries). In the quest to maintain production, details are often overlooked particularly with regard to ventilation.

- Staff shortages result in unqualified persons taking over positions of responsibility for tasks they are not able to perform safely and competently. An apprentice Miner (unqualified) fired the charges which resulted in the coal dust explosion at the Durban Navigation Collieries in 1926.

- Inaccessible areas of a mine such as return airways and old workings are seldom cleaned and stonedusted regularly. It follows that fine coal dust will thus collect on top of previously stonedusted roads thereby rendering the stonedust ineffective and providing conditions conducive for a major coal dust explosion to occur.

- Ineffective supervision and poor communications between Mining Engineers and first line Supervisors can lead to dangerous conditions in sections, coupled with dangerous practices; for example, overloading shot holes with explosives, not using any form of stemming and firing the charges in faces where fine coal dust layers 25 mm thick exist on the floor. Instructions given to have the dust systematically swept and loaded out of the section have been ignored. It is more often than not the case that poor communication between Mining Supervisors leads to misunderstandings. This in turn leads to actions which have caused methane explosions.

- Lack of stonedust barriers to smother the flame of a coal dust explosion.

- The most prolific cause of coal dust explosions are methane explosions and blown-out shots.

- The violence of a coal dust explosion increases the further the distance from the origin of the ignition.

- Coal dust explosions have been known to peter out in areas which are wet (Hlobane explosion).

- The application of stonedust and the sampling of roadways to ascertain percentages of incombustible dust is suspect. Those persons delegated to collect the stonedust samples once a month in a mine are able to salt the sample when it appears to contain a high percentage of combustible dust thereby providing incorrect figures to Mining Engineers. The Author has this knowledge from personal experience. Delays occur from the time of taking the samples to the issuing of the analysed figures. The urgency of having to restonedust an area is thus lost.

- The alleged rate of stonedust application at No. 2 Pit, Durban Navigation Collieries, prior to the explosion was 5 kg per ton of coal mined. This figure exceeds the current day norm which is considered satisfactory (2 kgs of stonedust per ton mined).

- Coals which have a volatile content exceeding 14% will be susceptible to coal dust explosions.

- The lack of adequate dust allaying systems (water sprays and scrubbers) in producing sections and main haulages leads to the make and dispersement of fine coal dust throughout the workings.

- Blown-out shots have similar danger as methane explosions in that the pressure of detonation raises a dust cloud which if sufficient in density and of the correct fineness and volatile content, is ignited by the flame and hot gases of the explosives.

- Coal dust explosions do not always follow the route of intake airways of a mine, as has been shown on the plan of the underground workings of the No. 2 Pit Durban Navigation Collieries. In this instance the force of the explosion also travelled along the main return airways - See Figure 4.

- From eye witness evidence, the dust cloud precedes the flame which is then followed by a cloud of smoke and dust. The roof over the pit head at No. 2 Pit, Durban Navigation Collieries downcast shaft was blown off.

- The systematic clearing up of fine coal dust and stonedust appears not to be a sound idea unless the area is restonedusted immediately.

- Return airways are the life blood of a colliery, and not maintaining a state of cleanliness in these returns on a systematic basis could cause dangerous conditions to arise (in the form of fine coal dust deposited on the roof, floor and sides and timber support).

- Dispersing stonedust by hand is generally not effective, since it is not spread evenly over the roof, floor and sides of roadways.

- Main haulages (tubs and conveyor) are a source of fine coal dust and measures are necessary to allay this dust. The dust arises from the run-of-mine coal which is conveyed in these roadways where generally high intake ventilation velocities exist.

- Inadequate face ventilation and the interruption of the ventilation by stopping the main ventilating fans causes dangerous accumulations of methane to occur both in the sections and on main haulages.

- The use of any open flame in the underground workings of a fiery mine must be treated with great circumspection. Two of the cases investigated, indicate that explosive mixtures of methane existed in main intake haulages far removed from the producing sections.

- Where ventilation to a particular area is reduced in quantity the possibility of explosions occurring as a result of a build-up of methane in such areas cannot be ruled out. The reduced ventilation quantity may not be sufficient to dilute and remove high methane emissions. Paradoxically, tests, by Mining Engineers, for methane are either not carried out or carried out in a dilatory manner, whereas the expectation is that methane tests, in such cases, should be increased for safety reasons.

- Lack of controls at the shaft top to search and question men for contraband can only be regarded as negligence on the part of Mining Engineers. It follows also that lack of discipline underground in not reporting workmen who are in possession of contraband can also lead to ignitions and again this is pure negligence on the part of Officials.

- Uncontrolled use of cutting torches and welding machines underground is a dangerous practice. In many instances those persons charged with gas testing prior to the use of such apparatus assume, incorrectly, that everything is in order and allow the operation to proceed with disastrous consequences.

- Slips and fault planes which exist in roadways, particularly where two or more coal seams overlay one another, provide a natural path for the passage of methane to the upper levels.

- In many instances areas are sealed off and it is later discovered that one or more roads or drifts remain open to allow the passage of methane from the sealed area to working areas.

- In a 1:3 incline, methane will find its way up the incline against an air velocity down the drift of 1,52 metres per second.

- Not all dangerous quantities of methane are properly reported to Mining Engineers by Supervisors who detect this situation.

- In many instances prior to the use of flame cutting underground standard procedures are disregarded generally as a result of haste.

- A lack of systematic inspections of all main transport and return airway roads can lead to the development of dangerous conditions which, if not corrected, have caused fires and explosions.

- Many Miners, especially the unskilled, do not appreciate the dangers of cigarette and pipe smoking in sections. The Mine Overseer, for whom the Author worked as a young Miner, caused a methane explosion in a Natal Colliery ten years later by smoking a pipe underground. He lost an eye as a result of this explosion. He was a man of great experience and one of the recipients of a Bravery Certificate for his rescue work at the disaster at the No. 2 Pit, Durban Navigation Collieries in 1926.

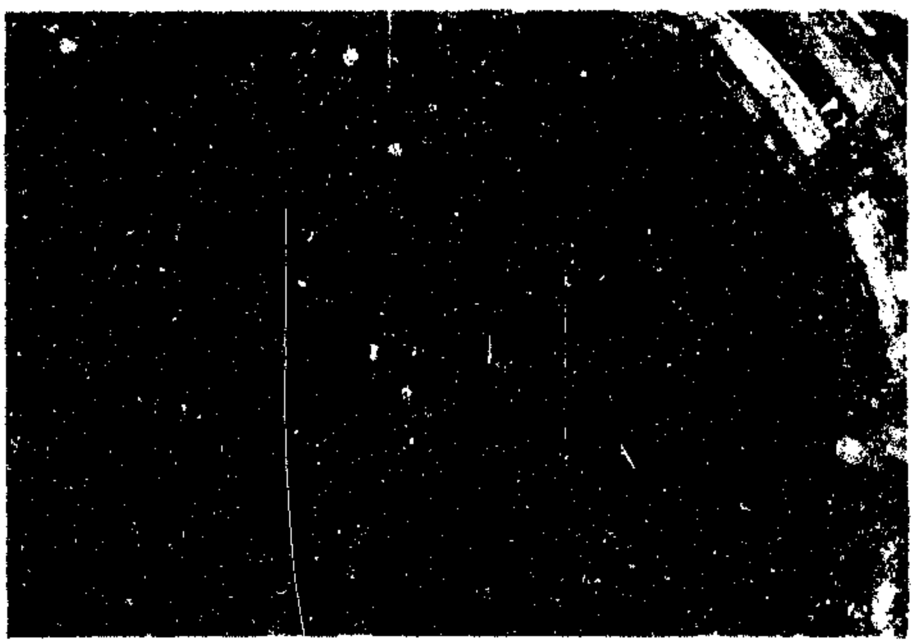
7.7 PRECAUTIONS TO BE ADOPTED

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

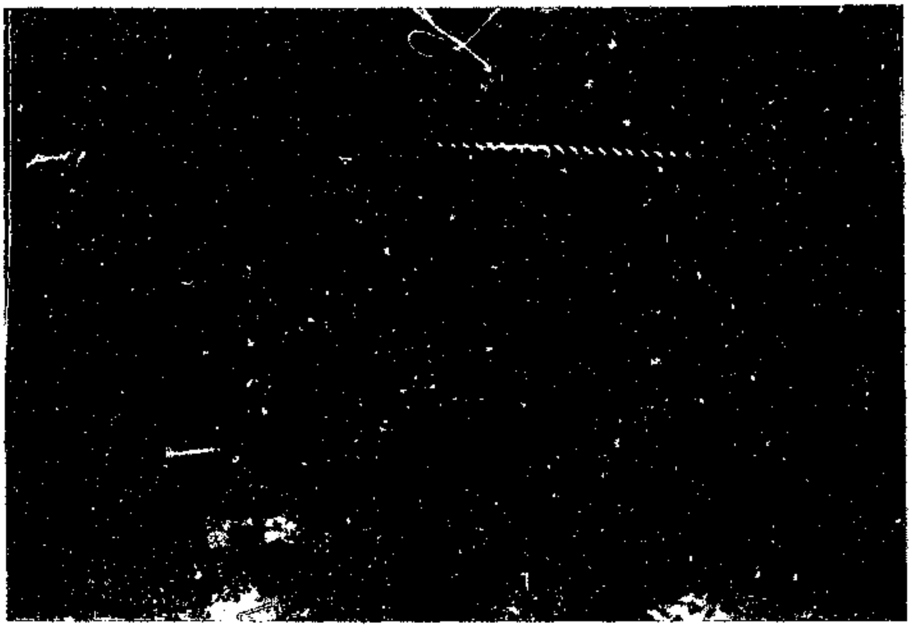
1. All return airways, however difficult to access, should be regularly cleaned and re-stonedusted.

   The acid test of well maintained and stonedusted return airways is that Mining Engineers should be able to inspect these returns in a flameproof diesel Landcruiser from the most inbye sections to the main upcast shafts.

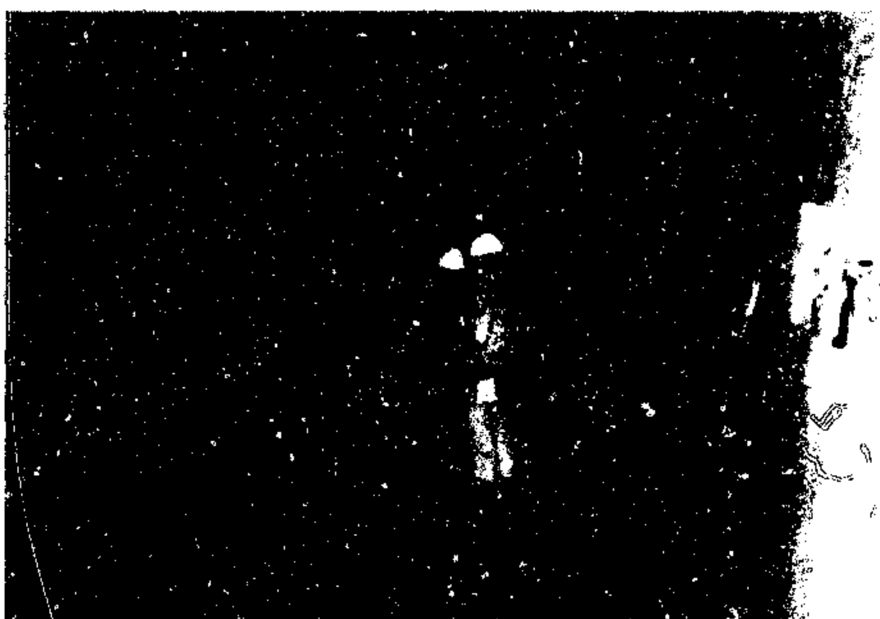
   The four photographs below support this statement.



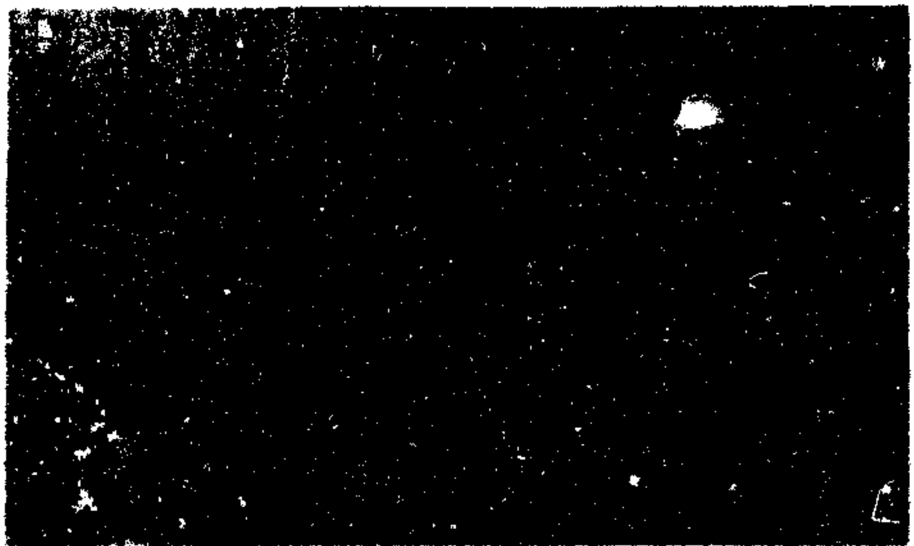
Mining Engineers inspecting a main return airway which has been supported by arch girders.



Mining Engineers inspecting a main return airway which has been cleaned but not stonedusted.



Mining Engineers inspecting a main return airway which has been cleaned and stonedusted.



Stonedust pockets stacked in a main return airway awaiting the arrival of a fan tail stonedusting machine to commence roadway stonedusting.

The percentage incombustibles in such returns exceeds 80% and would not support the propagation of a coal dust explosion.

2. An example of standard procedures used to regulate the use of flame cutting equipment underground is given below. These standards supplement the requirements of the Chief Inspector of Mines permissions, issued to each mine.

- Cutting and welding shall be done in accordance with the Inspector's permission.

- On completion of the work the area shall be inspected for hot materials.

- Approved flash back arrestors and hoses are to be used.

- Low pressure gas should not be used underground.

3. It is a recurring theme in this project report that safety is enhanced when Mining Engineers pay attention to all details of mining particularly as far as ventilation is concerned in the working faces.

4. First line supervisors are able to relieve absentee miners on odd occasions without a threat to safety. However when they continually relieve such absentee miners by running production sections, the following results may be expected:

   - Loss of continuity in planning and supervising.

   - Demoralisation of first line supervisors.

   - A loss of a sense of responsibility since first line supervisors believe that they cannot be held responsible for overall safety within their area.

5. A self-satisfied attitude amongst Mining Engineers regarding high standards of operations invariably and eventually could lead to an accident. Mining Engineers have a right to be proud of high mining standards but it requires continual asserted effort to maintain safe operations. The Author refers particularly to the Durban Navigation Collieries, No. 2 Pit disaster in 1926 where the mine was described as "being well managed, clean and well ventilated". A blown-out shot, fired by an uncertificated Miner in an unventilated rising heading which was described, in the evidence by the Mines Inspector, as being in a terrible condition, caused a major disaster.

6. Mining Engineers should ensure that their Juniors understand all instructions and the reasons why the instructions have been given. Follow-up of instructions is important.

7. In order to prevent a coal dust explosion from propagating and extending from the source of ignition, stonedust

barriers should be erected in every producing section as per the following standards:

7.1 Position of Barriers

- In accordance with Regulation 10.24.9.

- At least one stonedust barrier must be installed in each coal conveyor or endless rope haulage or road.

- The barrier must not be nearer than 140 m from the nearest face.

- The barrier must not be further than 365 m from the furthest face.

- The barrier must be advanced as necessary.

7.2 Type of Barrier

Stonedust barriers shall be of the Polish type.

Each barrier shall have two fixed roofbolt brackets, or alternatively adjustable trestles can be installed, on which shall be placed a frame equal to the width of the roadway (frame width 200 mm, depth 150 mm). Superimposed on the frame shall be boards, placed length-wise to the roadway and spaced across the whole width of the frame, thereby forming a shelf. Such shelf will form the platform upon which the stonedust shall be loaded. See Figures 14 and 15.

Types of Shelf

There shall be two types of shelf.

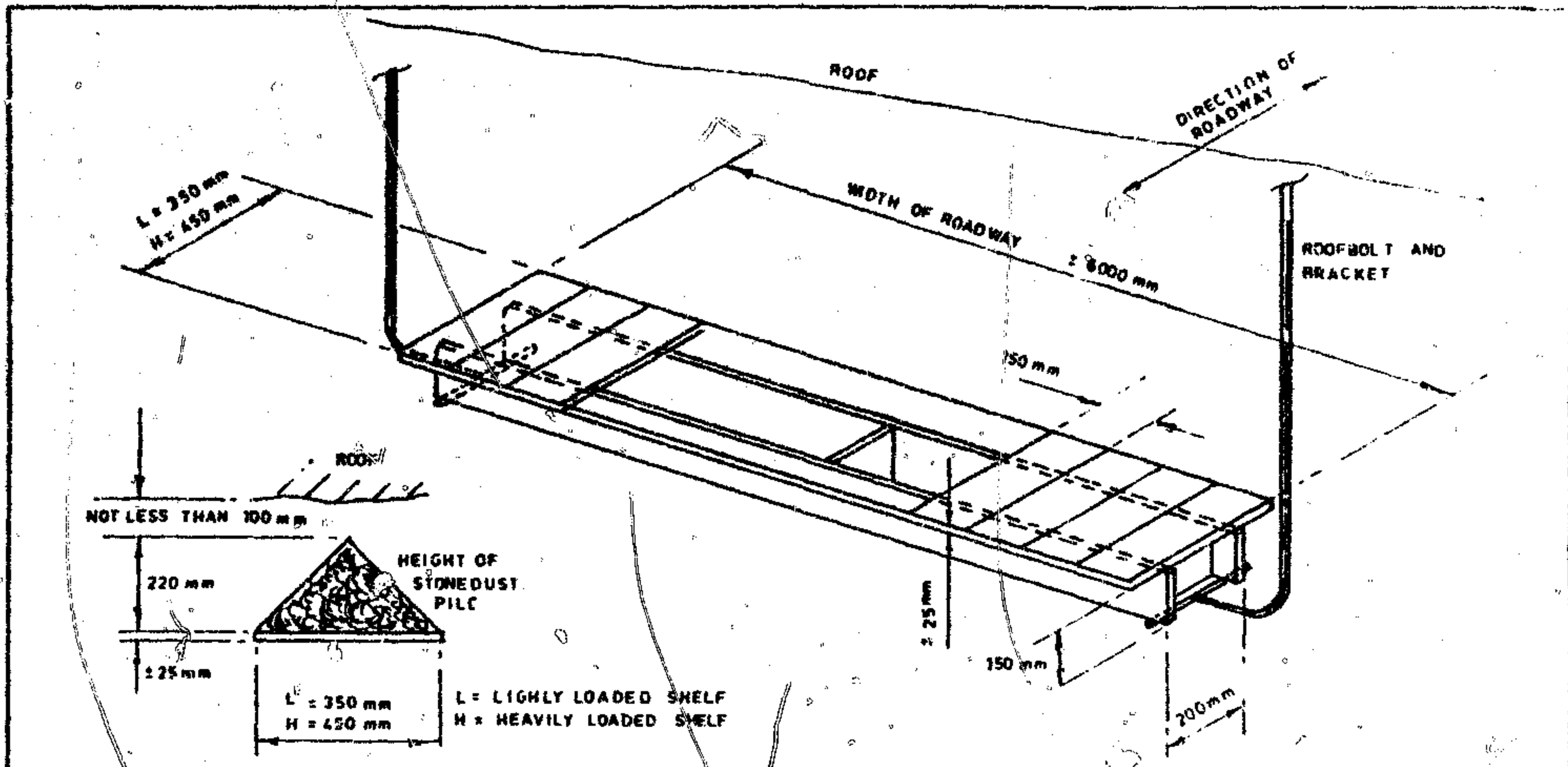


Figure 14
STONEDUST BARRIER
POLISH TYPE SHELF

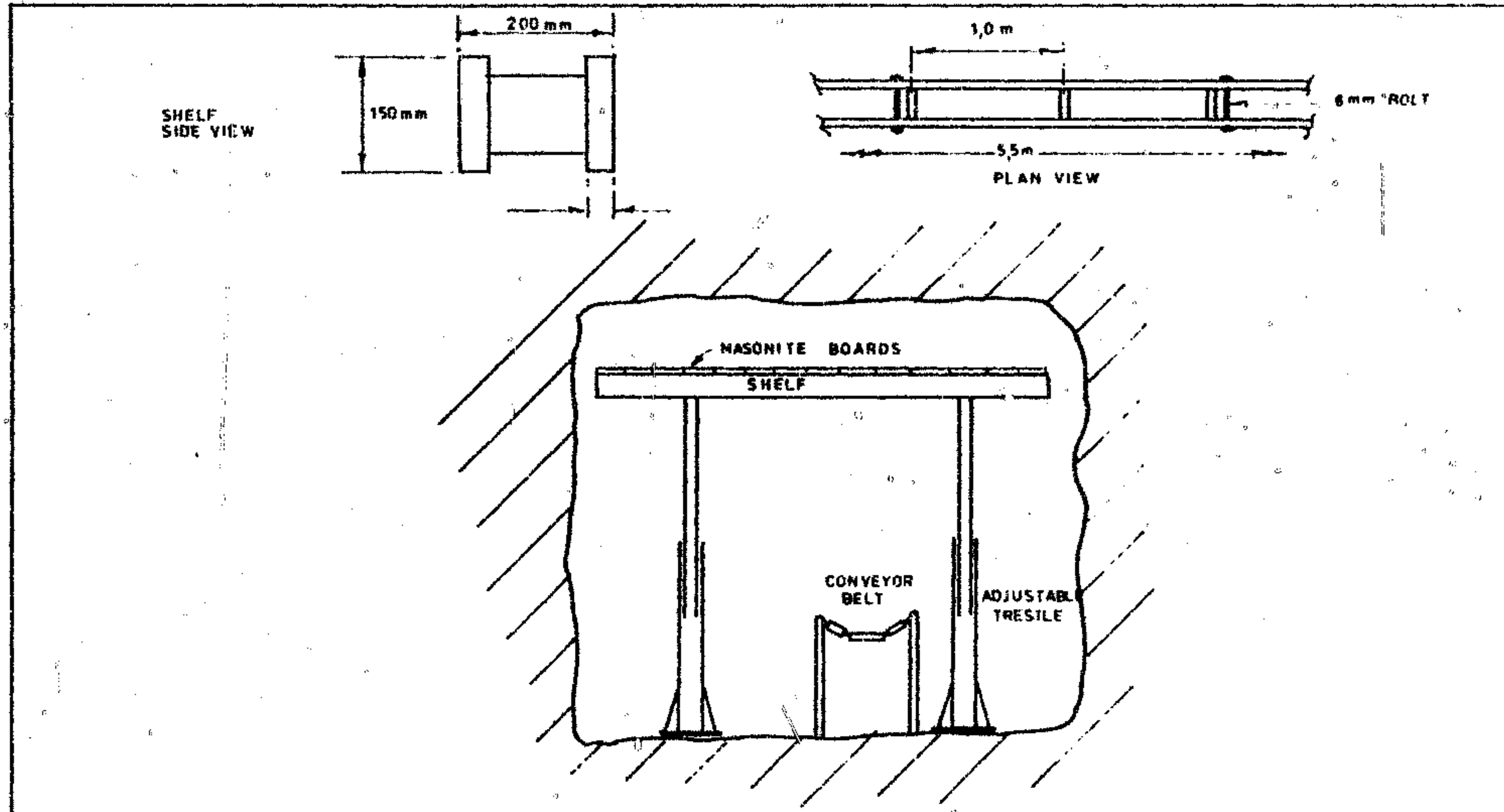


Figure 15
STONEDUST BARRIER - LANDAU-3 COLLIERY

Lightly-loaded shelf: which shall be 350 mm wide and shall be loaded at a ratio of 30 kg per metre of shelf length (average pyramidical dust height to be 140 mm).

Heavily-loaded shelf: which shall be 450 mm wide and shall be loaded at a ratio of 60 kg per metre of shelf length (average pyramidical dust height to be 220 mm).

Types of Barrier

There shall be two types of barrier.

Light barrier: which shall have 100 kg of dust per square metre of roadway.

For a normal 6 m x 3 road, a light barrier shall consist of 10 light shelves of 6 m length and spaced at 1,5 m intervals and not more than 2 m apart. Each shelf carrying 180 kg of stonedust. This is not normally used in the Group.

Heavy barrier: which shall have 400 kg of dust per square metre of roadway. The heavy barrier shall comprise of lightly-loaded and heavily-loaded shelves to a ratio of 1:2. The lightly-loaded shelves shall be in front of the heavily-loaded shelves in relation to the probable direction of an explosion blast displacement. For a normal 6 m x 3 m road, a heavy barrier shall consist of eight lightly-loaded shelves (180 kg each) and 16 heavily-loaded shelves (360 kilograms each). Such shelves to be spaced at 1,5 m intervals and not more than 2,5 m apart.

7.3 Mine Plans/Record Book

- Each barrier must be shown on the mine plan.

- In a record book, the following must be recorded:

Barrier reference number
Total number of shelves and loading thereof
Dispersibility of stonedust
Roadway cross-sectional area
Date of last dust renewal, date of last inspection with signatures of persons responsible.

7.4 Shiftboss - Inspection Barrier

- Each barrier shall be inspected weekly by the Shiftboss.

- Such inspection shall ensure that the barrier conforms to the Standard.

- In particular, each barrier must be examined for caking.

- The Shiftboss shall ensure that the stonedust is renewed immediately after evidence of dust caking is apparent.

- Waterproof stonedust may be used where necessary.

7.5 Abandoned and Semi-inaccessible Areas

Where there are abandoned areas that cannot be sealed off from current workings and that cannot be stonedusted and sampled, the following procedure must be adopted:

- **As much as possible of the abandoned areas must be effectively sealed off with brick stoppings.**

- **Heavy stonedust barriers must be installed and maintained at the edge of the abandoned area to cover all roads connecting it with the working area.**

- **All roadways connecting the abandoned area to the working area must be adequately stonedusted and the level of** incombustibles maintained by regular sampling and redusting as required.

8. While the taking of stonedust samples and analysis thereof on a monthly basis is a legal requirement, the system is cumbersome and does not provide Mining Engineers with immediate on-the-spot results.

Krystolik et al (1985) gave a description of a portable instrument with digital readout destined for quick measurements of solid incombustible content serving as a protection against coal dust explosions.

The United States Code of Federal Regulations determines the basic requirements put to analyzers of solid incombustible content in coal dust.

According to these requirements a portable analyzer should be a self-contained unit suitable for service in underground coal mines and be equipped with a quantitative indicating device that is capable of indicating the incombustible content over the range of 50-100% incombustible and having batteries capable of failure-free operation.

The need for an instrument which quickly determines the incombustible content is great where stone dusting is the basic method for safeguarding against coal dust explosions. This is true for the United States coal mining industry and in a very large degree also for the Polish and South African coal mining industry in which stonedusting, together with anti-explosion barriers, constitute the main protection against the propagation of coal dust explosions.

As a result of years of research work done in the Central Mining Institute, an instrument was designed and its purposefulness tested in mines.

to the Wankie Colliery disaster in 1972, all Transvaal collieries practiced the watering down of roadways as a precaution against coal dust explosions. No stonedust was applied.

At present the Polish mining regulations require that in the Polish coal mining industry 50%-90% of solid incombustible matter be maintained, the percentage depending on the coal dust explosion hazard degree.

The total length of stonedusted zones is given in Table 7.9.

Table 7.9

% of Solid Incombustible Matter	Total Length of Stonedusted zones km
90	35
80	416
70	131
50	600

The maintaining of the zones is becoming more and more difficult since growing mechanisation and concentration of output cause a more and more intensive deposition of new dust layers. The deposition rate is estimated at 2-100 g/m^2/day in return airways and at 2-80 g/m^2/day in galleries driven by heading machines.

To maintain the proper quantity of solid incombustible matter in stonedusted zones frequent checks must be done. The regulations do not state precisely how often the checks are to be carried out, they demand only that the required amount of incombustible matter be maintained continuously.

The quick shifting of the zones poses additional requirements to the stonedusting and the settling of new dust layers calls for smaller intervals between the checks. Due to this situation the determination of the content of solid incombustible matter is becoming a more and more difficult problem for colliery laboratories doing check analyses. These analyses depend on combusting dust samples taken underground during three hours at a temperature of

480°C and on determining the loss of mass. In many collieries more than 1000 samples are to be analysed each month.

The need has arisen to develop a method for quick determination of solid incombustible content without the necessity of combusting, weighing and so on.

In connection with the progress of isotope technique, the idea emerged to take advantage of the radiometric methods - based on interaction of beta and gamma radiation with material - for determination purposes.

Design of the Instrument

The internal view of Inflabar PC is shown in Figure 16.

The electronic system based on integrated circuits ensures correct operation of the instrument and a direct digital readout of solid incombustible matter content.

The switch on the back wall of the instrument has two positions.

Beta radioactive stront 90 is the radiation source. The radiation of stront falls on the dust sample in the container and after back-scattering falls on the semi-conducive radiation detector which is specially made to the requirements of this instrument in the Institute for Nuclear Problems.

The radiation source turns round together with the cover of the container with the sample and therefore during the change of the sample the operator is not exposed to radiation.

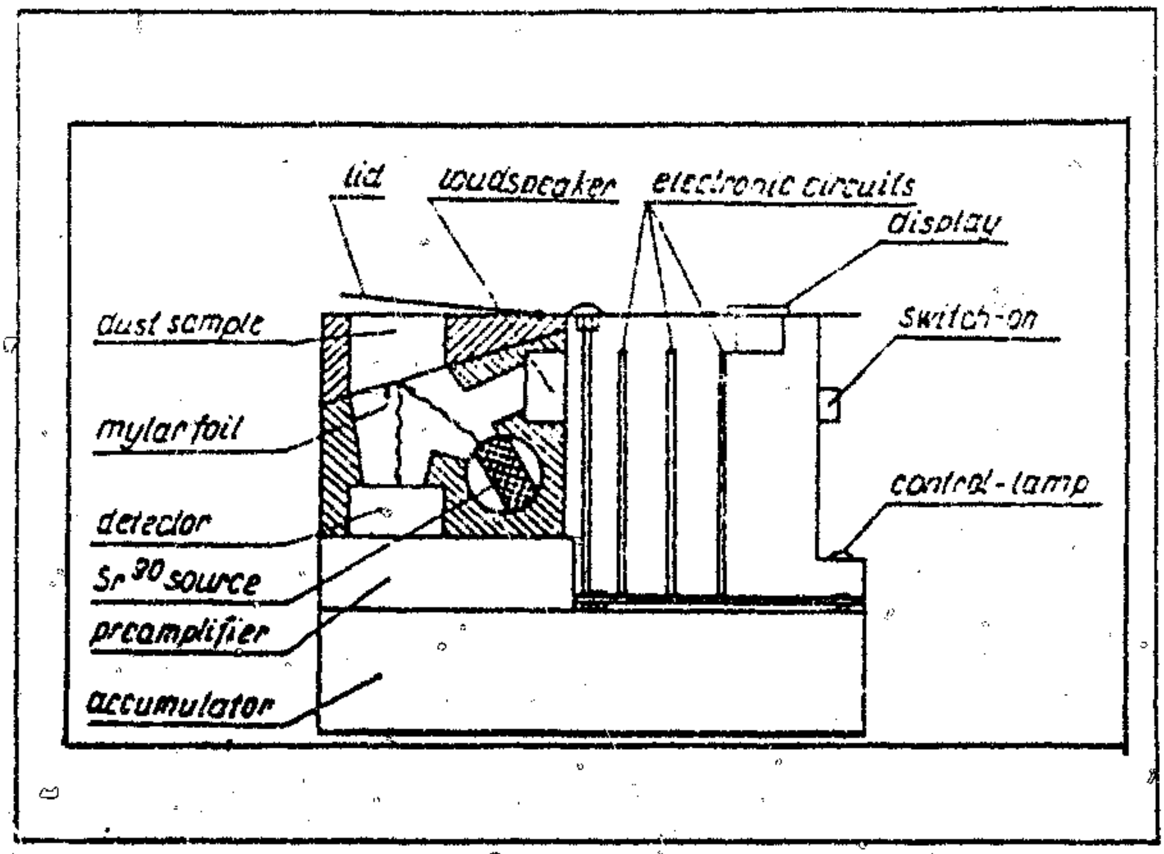


Figure 16
INTERNAL VIEW OF INFLABAR PC

The radiation source, detector and sample container are housed in one block made of organic glass which ensures steadfastness of mutual position of these elements, small dimensions and small distances between the elements.

On the back wall of the instrument there is a switch having two positions. Position "Calibration" serves for regulating zero and the accuracy of the instrument. In this position 9 subsequent measurements are done at a shortened time and at only two significant digits. The tenth measurement has full time and accuracy. In position "Measurement" one measurement without calibration is done. Position "Calibration" enables quick calibration of the instrument. After finished measurement, the readout switches off automatically.

Technical data of the instrument are as follows:

- **Measurement range**
  50-100% solid incombustible matter content
- **Measure time about 30 s**
- **Ambient temperature range**
  178-308K
- **Dust sample mass about 25 g**
- **supply Cd-Ni battery, 7.2 V, 3.5 Ah**
- **Supply autonomy - minimum 250 measurements**
- **Radiation source - stront 90, activity 5 mCi**
- **Maximum radiation intensity - not greater than 2.5 mR/h at a distance of 5 cm from the instrument**
- **Mass of instrument - 3 kg**
- **Dimensions - 215 x 80 x 175 mm**
- **Electric system - intrinsical safely II BI.category i according to IEC and CENELEC/[a].**

Conclusions

1. The Inflabar PC instrument meets the basic requirements

of CFR 30/29 and of Polish regulations and is suitable for use in mines.

2. The instrument can be used for checking the state of stone dusting according to the so far valid methodology of taking samples. It also provides the possibility of quick and easy detection of places in which stonedusting is not adequate.

The Inflabar is very useful for checking during the operation of pouring out the stonedust itself.

The possibilities of the instrument will become evident during its operation on a large scale.

9. All coals with a volatile content of 14% or greater can be regarded as susceptible to coal dust explosions. Roadways in such mines should thus be well stonedusted.

10. The most prolific cause of dust formation is the use of continuous mining machines. The ventilation of roadways in which these machines are used is dealt with in Chapter 2.

Figure 17 depicts the use of fans and scrubbers on the latest generation continuous miner for the safe handling of dust.

11. That coal dust explosions in bord and pillar workings travel up the main returns is a fact and more research is necessary to determine how this phenomenon should be handled. At present the method of systematically and frequently cleaning and re-stonedusting returns is regarded as the safest approach (item 1 of this subsection) to ensure that coal dust explosions are unable to travel along the return airways.

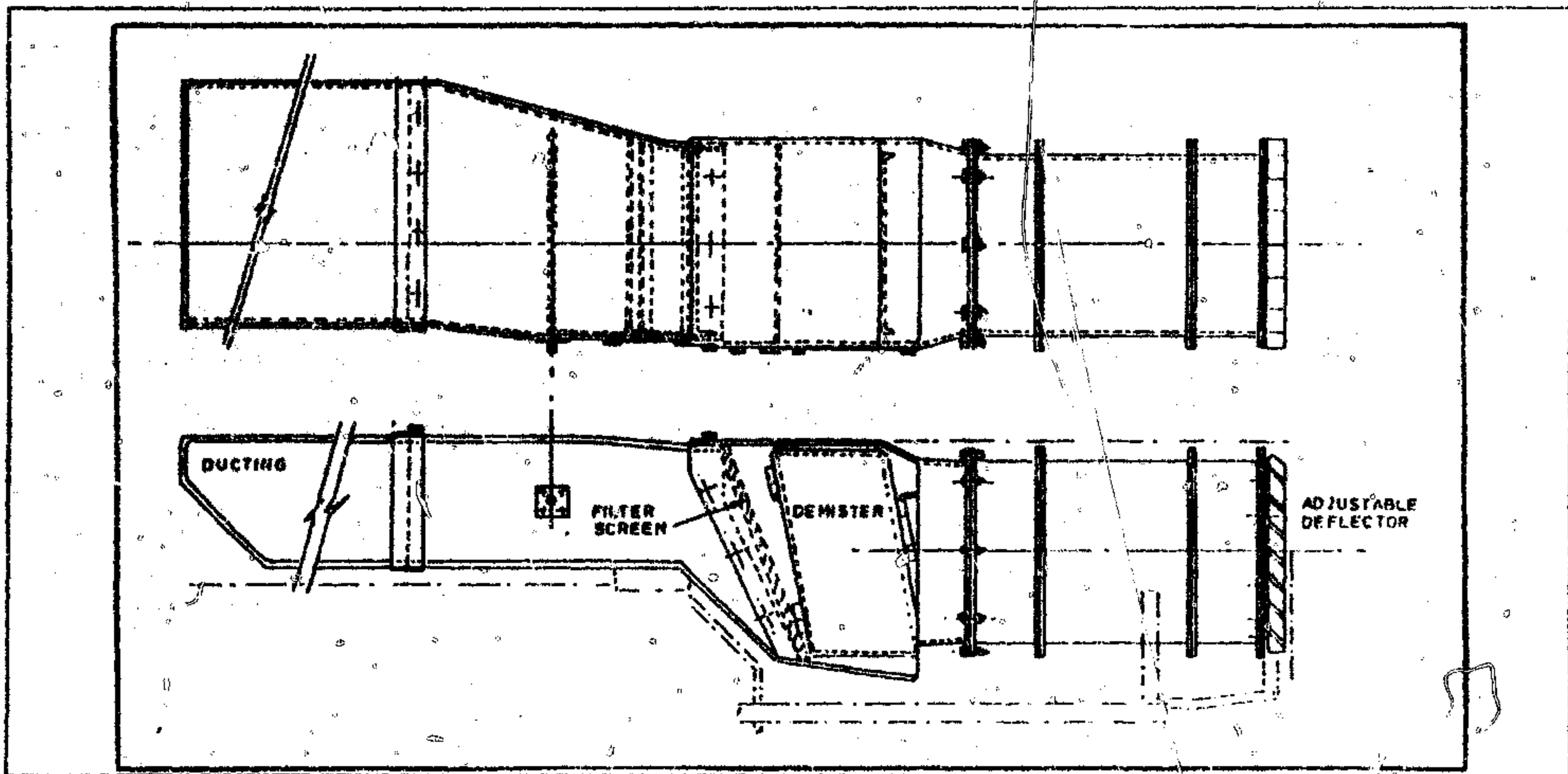


Figure 17
CONTINUOUS MINER DUST EXTRACTION SYSTEM

12. The modern trickle and fan tail stonedusting machines are not only mobile and easily transported but efficient in the distribution of stonedust. The erection of stonedust silos on surface at strategic sites minimises the bulk transport of stonedust underground over long distances.

13. The suggested mine standards regarding main fan stoppages is dealt with in Chapter 6. The stoppage of the main fan in a gassy pit for longer than 15 minutes is a serious event.

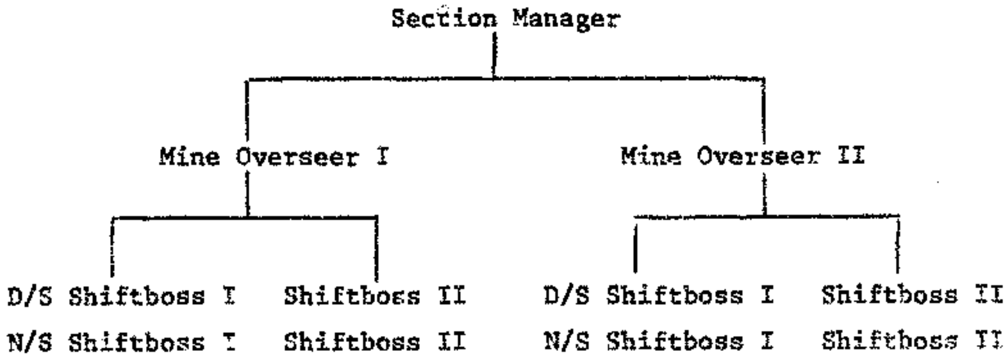
14. No open flames should be allowed underground. Standard procedures for the use of cutting torches and welding underground have been set out in item 2 of this subsection.

15. Senior Mine Management should be made aware immediately of reductions in the ventilating current in any area of a mine. When ventilation is reduced, for example in a shaft, standard procedures should be drawn up which deal with the use of cutting equipment in such a shaft including the testing for methane.

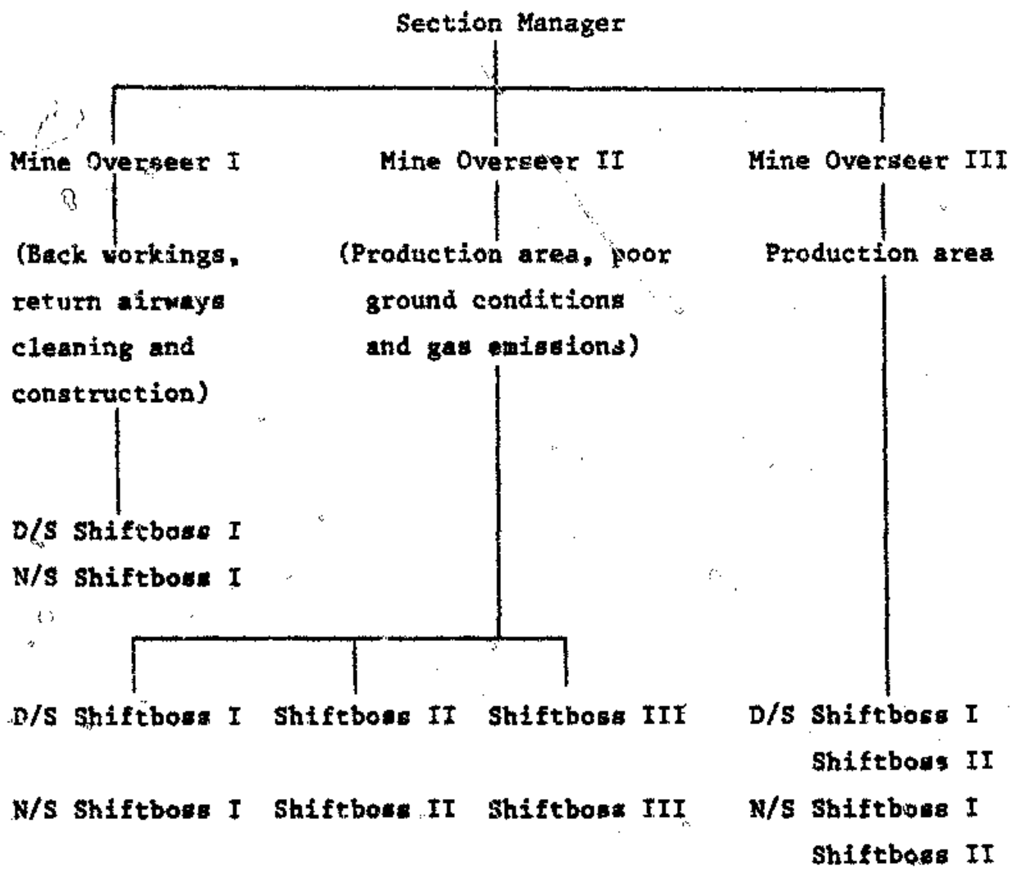
16. Senior Mine Management should ensure that all entrances to shafts where men are to travel are guarded to the extent that persons are searched for any form of contraband. The possession of contraband on persons should be reported to a responsible official immediately.

17. The greater the incidence of geological problems (faults, slips, dolerite dykes, burnt coal zones and poor roof) and methane emissions, the greater the need for increased supervision both quantitatively and qualitatively. This dissertation highlights many serious incidents where supervision was inadequate. Such situations lead to a lowering of standards, the taking of chances and poor communication. Mining Engineers should be aware of changing circumstances and take the necessary action as shown below:

1. New Mine and normal conditions (300 000 ROM tons per month)



2. Same mine as in 1. above with poor roof conditions and gas emissions in one area and attenuated haulages and return airways (300 000 ROM tons per month).



The above organisational charts indicate the additional line supervision required in the second case to deal with attenuated return airways and deteriorating ground conditions in one production area of the mine.

18. Senior Mine Management should ensure that they are fully aware of all roadways which exist when sealing off areas which are experiencing spontaneous combustion or methane emissions. Many instances have been encountered during the cours of this research, where not all roadways were thoroughly sealed because their existence was not known. In the majority of cases, this is due to Mine Surveyors not being made aware of faces which are being advanced and hence not recording these faces on mine plans.

19. Enright (1985) in studies in Australia has concluded that experiments in the 8 and 20 litre chambers showed that Wyvern coal dust at 40 percent moisture content and the Bulli coal dust at 26 per cent moisture content were capable of developing significant explosion pressures and rates of pressure rise. The limiting value of the added water required to inert the dust was considered to be that value which prevented the generation of a dust cloud in the laboratory explosion chambers rather than that value which inhibited a dust cloud explosion. It is possible that dust explosions could occur at higher moisture contents if a sufficiently turbulent dust dispersion system was available.

The majority of dust explosions that occur in underground coal mines result from the ignition of a methane-air mixture. The resultant gas explosion has a high temperature flame front and is capable of developing an extremely turbulent air blast. These conditions may be sufficient to produce an explosion with dust at a higher moisture content than the limiting value found in laboratory experiments.

It follows that wet conditions will not always act in inhibiting the development of coal dust explosions. Prior

7.8 CONCLUSION

Mining Engineers are often lulled into a false sense of safety by having "well ventilated mines" and "clean and well managed operations". Unless they attend to the details of good and efficient face ventilation, the systematic cleaning of return airways, the adherence to proper standards and good day to day communications, methane and coal dust explosions are likely to occur.

Two of the incidents referred to in this chapter resulted in methane explosions as a result of open flames being used in a downcast incline shaft and a main intake haulage respectively. In both situations the possibility of a methane explosion occurring would be considered, by Mining Engineers, to be remote. However, in both cases, the recipe for a disaster was present.

The main ventilation fans were respectively regulated and stopped thereby reducing the ventilation considerably in the respective mines. The act of reducing ventilation immediately allowed the build up of methane to occur in the most unlikely places, the situation being exacerbated by the presence of faults and slips between the different coal seams and the presence of a drift which was not shown on mine plans, and had not been sealed off.

The indiscriminate use of a cutting torch and blow lamp with no prior tests being conducted for methane resulted in explosions.

Making assumptions that areas are free of methane and that the use of open flames is not dangerous is fallacious. Whenever it is necessary to allow open flames to be used the area should be thoroughly examined for methane and adequately ventilated prior to and during the cutting process.

Mining Engineers should ensure that adequate searches are conducted to ensure that no contraband is taken underground.

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CHAPTER 8 INSTANTANEOUS METHANE OUTBURSTS

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(d) Face Preparation
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8.7 CONCLUSIONS

8.8 REFERENCES USED IN CHAPTER 8

8.1 INTRODUCTION

Several serious methane outbursts have occurred at three South African collieries over the past 35 years; however, such incidents are seldom reported on publically or internally between mining companies and are therefore not widely studied. Mining Engineers are thus disadvantaged in trying to deal with this problem.

Methane outbursts occur when workings or boreholes approach an area in a coal seam where methane is entrained under high pressure. At a certain point in time, the intervening strata between the workings/borehole and entrained gas is weakened to a point where it cannot resist the pressure of the gas which then escapes in a violent outburst.

Dangerous quantities of methane enter the workings after blowing hundreds of tons of coal into the roadway, to be followed by clouds of fine coal dust which, due to the shock wave, can be deposited on the roof sides and floor of the section up to 400 metres from the source of the outburst. Such conditions could lead to a methane/coal dust explosion.

This chapter will examine the phenomenon of methane outbursts, where these are likely to occur and with the aid of actual experience at Springfield and Indumeni collieries, describe what happens when the methane gas is released. Suggested methods to be adopted to obtain prior warning of such outbursts are discussed. The characteristics of the coal deposit in the South Rand and Indumeni coalfields is analysed particularly in areas where these outbursts have occurred.

The first real evidence of the circumstances surrounding a methane outburst in the South Rand coalfield included the following account given by a Gang Leader in Section 7 at Springfield Colliery on 17th October 1969.

"Teko Rantho, No. 2271/306597: sworn states:

I am a Gangleader in section 7. I remember the day an outburst of gas occurred in the section. I was in the main near the compressor. I was waiting for the blast to take place. Blasting took place and I heard a storm in the section. I was blown about towards the faces. I realised that something dangerous had happened and I directed my workers to run away in a certain direction. They ran away. There was still a roaring sound going on".

This evidence is corroborated by other witnesses and provides some idea of the force of such a methane outburst.

As the coal excavation process takes place at greater depths and at higher production tempos in South Africa so will the likelihood of instantaneous methane outbursts increase. There is a need to establish why such outbursts occur in massive burnt coal zones which are found adjacent to dykes.

The seriousness of the problem is highlighted by Hargraves who states:

"The Collinsville outburst in 1954 was followed by several years of trial mining and research, but the mine was closed, largely because of the unsolved outburst problem".

8.2 SOURCE OF METHANE

8.2.1 In situ methane content

In the history of coal from vegetation to anthracite, coal undergoes particular progressive chemical changes (Stach et al 1975). One of the results of this is to produce chemically the essential fluid products water, carbon dioxide and methane. These are produced in varying proportions throughout the whole maturation process to the end product, anthracite, in typical proportions as shown in Figure 1 (Hargraves). Concurrent physical changes in the coal include porosity and permeability reduction, increase in density, and darkening of colour. From beginning to end, the amounts of migratory products are vast per ton of matured coal, of the order of 0.75 ton of water, 600 m^3 of carbon dioxide (CO_2) and 600 m^3 of methane (CH_4) at normal temperature and pressure.

Most of the fluids produced are not represented in the seam environment, but have progressively migrated away, apart from some residual fluids retained in coal at any one stage of maturation. This maturation goes on under load of a cover of rock to the surface, rocks generally saturated with water up to the water table - almost to the surface, and the fluids retained in the seam are held at pressure close to hydrostatic head from the water table. Conveniently, the retention of gases by coals is termed sorption to cover all modes of retention. Although coal has a stronger affinity for CO_2 than CH_4, the greater production of CH_4 in the later stages of maturation is able to progressively replace most of the CO_2 retained from earlier stages, so that the seam gas of higher rank - bituminous to anthracite - coals is essentially CH_4.

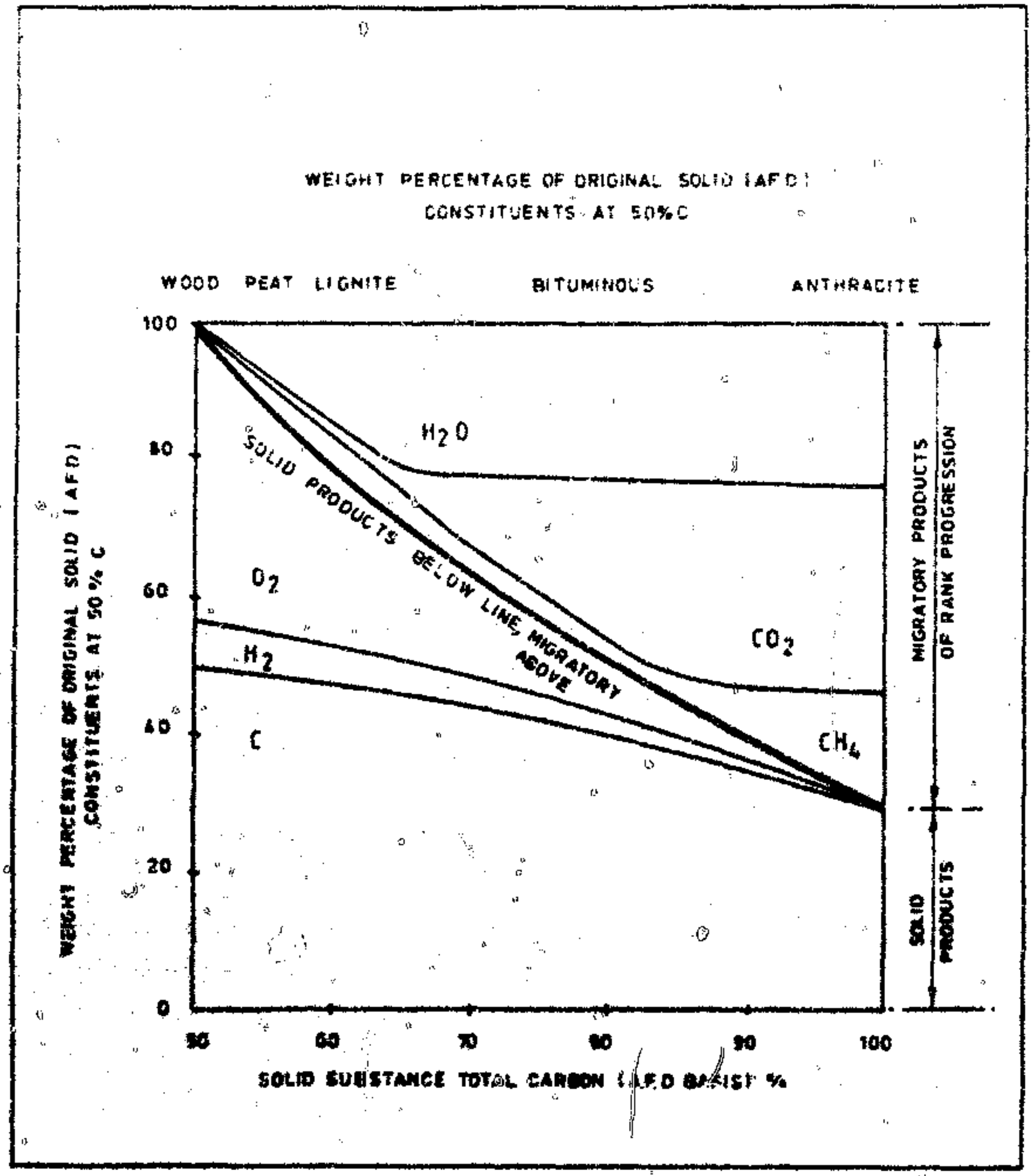


Figure 1

MATURATION OF COAL AND MIGRATORY

When coals, or coal measure strata are intersected by igneous material, the contact metamorphism, largely the heat of intrusion, may increase the rank of the coal, but more importantly gases of pneumatolytic origin especially CO_2 displace the CH_4 formed in-situ in the coal. In some areas this invasion by CO_2 if sufficiently recent, exists to the present time, and the still forming CH_4 has not had the chance to replace the CO_2 again. Such a situation is rather rare on the world basis, but in Australia both the Bowen and Sydney Basins are so affected.

How much gas is there?

The sorptive capacity of coal for gas depends on the porosity of the coal, which in turn is related to rank, to the nature of the gas and to the pressure of the gas. Figure 2 shows typical equilibrium isotherms obtained from 90 samples taken from twenty collieries (Phillips, Waite, et al). Ettinger (1949) investigated sorptive capacities of a number of coals at a range of pressure, Figure 3. The sorptive capacity of coal for CO_2 is two to three times that for CH_4. As a rough guide, virgin seam gas pressure can be taken as about 7 kPa per metre depth, where water tables are shallow. Another factor affecting amount of gas in a seam is seam thickness.

It is convenient to specify gassiness in terms of volume of gas per ton, ignoring seam thickness. Thus, ignoring the effect of stress in reduction of sorptive capacity, an approximation of the amount of gas per ton can be made on the basis of depth, rank and isotherms of sorption for the appropriate gas composition. Alternatively, a direct measurement of gas pressure may be made, replacing depth as the approximation of gas pressure, or the amount of gas may be measured directly from a virgin coal sample, or an index to gassiness may be made on a coal sample on an empirical basis (Hargraves, 1962)."

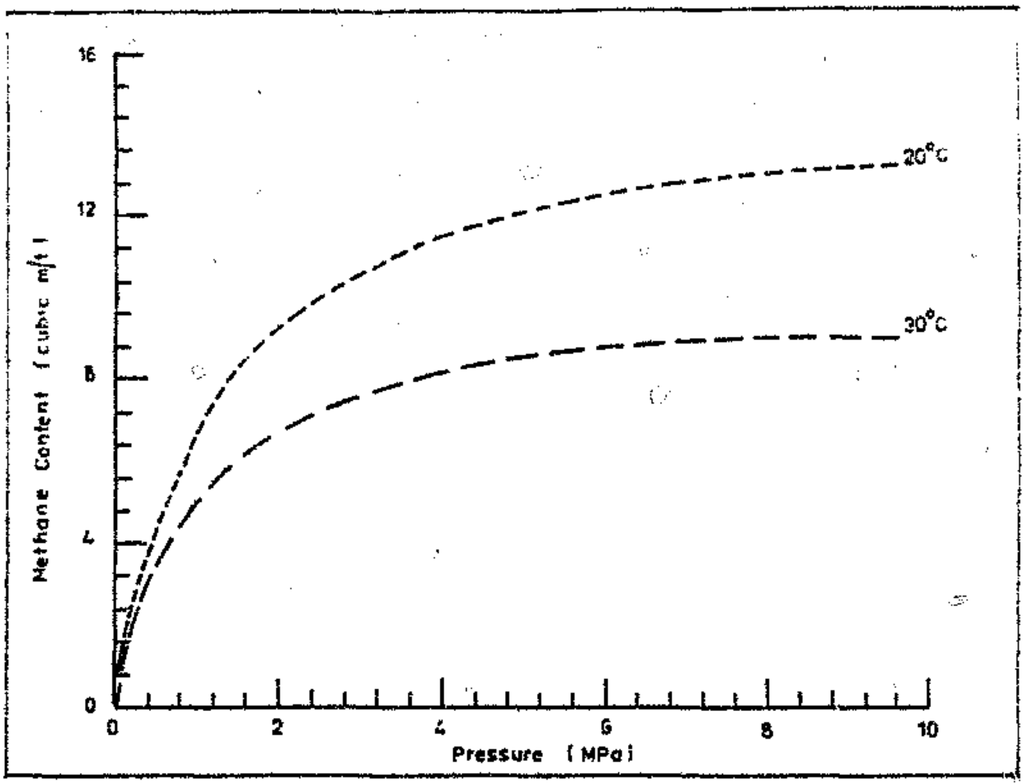
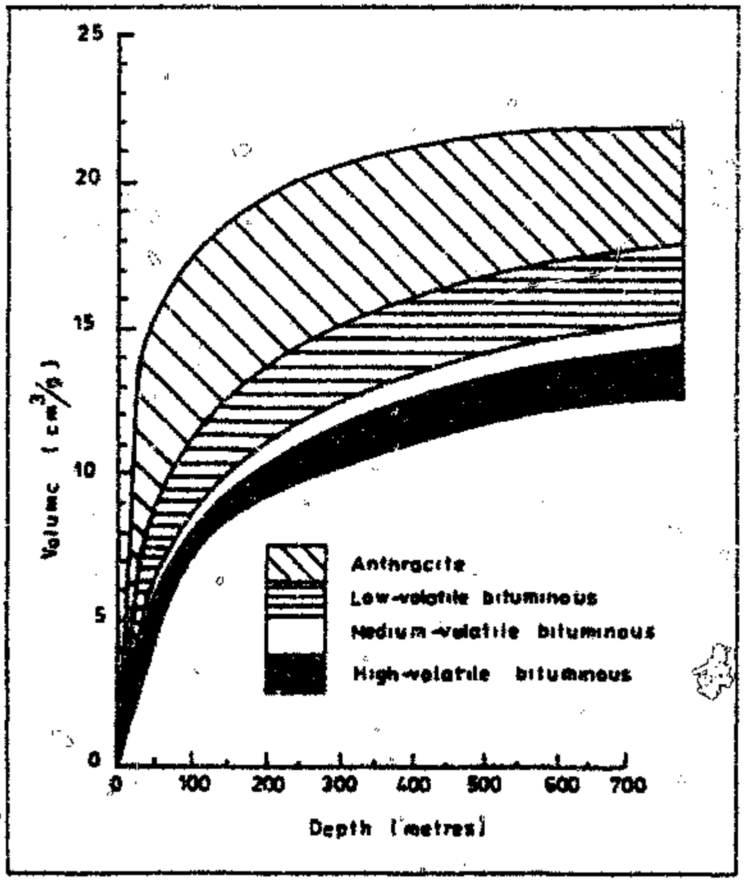


Figure 2

TYPICAL EQUILIBRIUM SORPTION ISOTHERMS



ADSORPTION ISOTHERMS FOR COAL SHOW THAT DEEPER, HIGHER RANK COALBEDS ARE GASSIER

Figure 3

Determination of in-situ methane content

Phillips, Waite et al, have stated that "very little is known of the mechanisms by which methane is released during the coal excavation process and data on the in-situ gas content, permeability and gas pressures of South African coals is virtually non existent."

Two methods are available in measuring or determining the in-situ methane content of coals. These are:

- The direct method which consists of obtaining a fresh sample of coal, enclosing it in a container and measuring the release of gas against time since the sample was obtained. Two disadvantages arise from using this method, namely the quality of the field staff and an accurate estimate of the time that the sample was first disturbed in the excavation.

- The indirect method entails measuring the temperature and pressure of the seam gas whereas the absorption isotherms for the coal seam are measured in the laboratory. In essence this comprises the repressurising of crushed coal samples, at constant temperature and under a predetermined pressure of methane.

Using work by A.G. Kim in the United States, Phillips, Waite, Billenkamp and Stripp concluded that from their tests gas contents of the samples tested contained from about 3 to 10 m^3 of methane per ton of coal. It is of interest to note that Kim has determined that in the United States of America a pressure of 1 MPa of methane represents a typical pressure at a gassy seam at a depth of 150 metres.

Figure 3 showing adsorption isotherms for coal indicates that deeper higher rank coal deposits are gassier. Also at higher pressure, coals can contain more methane. This would tend to explain the gassy conditions leading to instantaneous methane outbursts in the South Rand and Indumeni coalfields.

8.1.2 Permeability

The Concise Oxford Dictionary defines permeability as 'ability to be permeated' - penetrate throughout, pervade, saturate; diffuse itself through, among; in other words travel through.

Geological studies have also established that most coalbeds have a directional permeability, due to sets of interconnected vertical fracture planes within the coal. In coal termed blocky, a set of prominent fractures, or face cleats, are spaced 25 mm to 150 mm apart. A secondary set of fractures, the butt cleats, occur at right angles to the face cleats. In coals that are called friable, the fracture spacing is less than 25 mm, and both sets of cleats are equally prominent. In blocky coal, methane emission rates are higher along the face cleats. Rib stability and roof conditions are also related to cleat orientation.

Surface and photolinear methods have been used to determine cleat orientation for 32 coalbeds in Pennsylvania, West Virginia, Virginia, Alabama, Oklahoma, Colorado, and Utah.

Factors affecting the retention and migration of methane in coalbeds were studied to provide a basis for the more rational design of coalbed degassifiction systems. The following information resulting from the studies has been useful in designing more effective systems for draining methane from coal and essential in explaining apparently anomalous field results. The physical structure of coal, with both very small pores and large cracks and fractures, affects the methane emission rate. Methane flows from the pores by diffusion; flow through the cracks is laminar. In this two-step process, the relationship between diffusion coefficient and fracture permeability, which varies for different coals, determines the methane emission rate.

Methane emission was found to be retarded by the water normally found in coalbeds. As the coalbed is dewatered, methane emission rates increase. The burnt coal zones in South African collieries are known to be dry - at Indumeni Colliery 'a cloud of white dry ash' was blown into a roadway following a methane outburst. Laboratory studies of methane adsorption on coal indicated that coals that are higher in rank are capable of holding larger quantities of methane. Also, at higher pressures, coals can contain more methane. Since both rank of coal and pressure increase with depth, deeper coals can be expected to be substantially gassier than shallow coals. Adsorption data have been used to obtain a rough estimate of how much gas a coal contains.

Figure 4 is a surface exposure of a coalbed showing the orthogonal orientation of face and butt cleats and Figure 5 depicts the flow of methane through coal in two steps; from the solid coal into fractures and then along the fractures into the mine workings.

The gas content of a coalbed is made up of two components - free gas compressed in pore spaces and gas adsorbed in the internal surface of the pores The quantity of free gas depends on the porosity of the coal, pressure of gas in the pores and the temperature. It usually makes up a small fraction, about 5-10% of the total gas content (McPerson, 1975). However, for the determination of total gas content, it is necessary to know this quantity.

The greater part of the gas content in coal in-situ is held on the surfaces of the coal pores and microfractures in adsorbed form as a monomolecular layer. Since the internal surface area of coal can be as large as 90 m^2g^{-1}, the quantity of adsorbed gas can sometimes be extremely high. It is therefore important to be able to determine the amount of adsorbed methane accurately.

Figure 4

A SURFACE EXPOSURE OF A COALBED SHOWS THE ORTHOGONAL ORIENTATION OF THE FACE CLEATS AND BUTT CLEATS

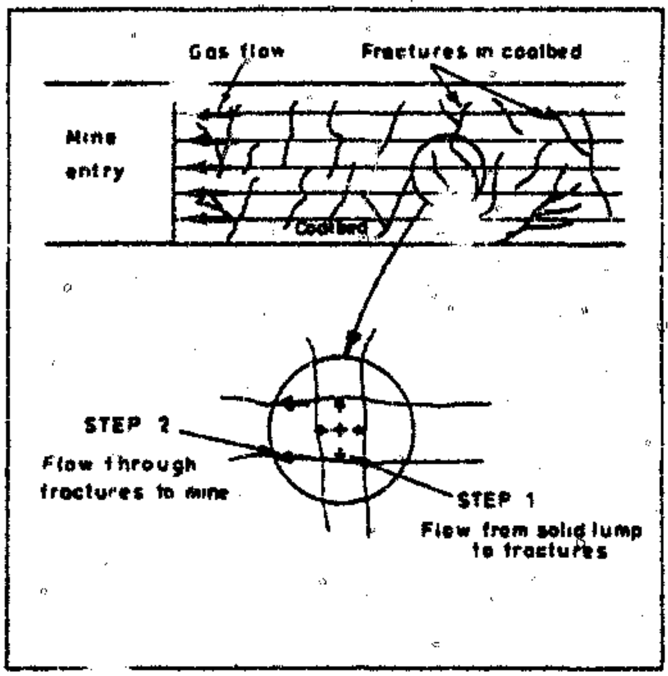


Figure 5

METHANE FLOWS THROUGH COAL
IN TWO STEPS;
FIRST FROM THE SOLID COAL INTO FRACTURES
THEN ALONG FRACTURES INTO THE MINE

Free gas

Free gas is the mobile gas stored in the coal pores. To determine this quantity, the porosity of the coal must first be measured. Porosity is the ratio of void volume to the bulk volume of a porous material. The method of determination of porosity by mercury injection is not suitable for coal, since the surface tension of mercury is much too high to permit entry into the micropore structure of coal. A gas expansion method was felt to be more appropriate for this reason. This method utilizes Boyle's Gas Law for ideal gases. It was, therefore, important to use a gas which has a very low adsorptivity on coal. Helium was found to be suitable for this reason and also due to the fact that the small size and low molecular weight of the molecules ensure that it enters all the pores, most of them being micropores.

The quantity of free gas can be calculated if the porosity and gas pressure are known. It is expressed as the volume of gas, at STP, per unit weight of coal.

Adsorption isotherm

An adsorption isotherm is the curve giving the quantity of gas adsorbed as a function of pressure at a given temperature. Examples of adsorption isotherms are shown in Figure 1. A few points are evident from the figure.

(1) At a given temperature, carbon dioxide is more readily adsorbed than methane, which in turn is more readily adsorbed than nitrogen, indicating that the nature of gas affects this quantity significantly.

(2) At the same equilibrium pressure, the quantity of adsorbed gas decreases as the temperature increases. Starting at 23°C, it falls at a rate of some 0.8% per degree for bituminous coal (Boxho et al, 1980).

(3) The quantity adsorbed increases with pressure and reaches an equilibrium at a pressure that varies with the type of coal. At this equilibrium pressure, formation of a monomolecular layer is complete and the quantity of adsorbed gas does not change with any increase in pressure. However, if the pressure is increased beyond a critical point, a second molecular layer begins to form, causing a further increase in the quantity of gas adsorbed. This critical pressure is normally well above that found in coal seams.

Most important of the other factors is the dependence of adsorption on the moisture content of coal. Gas content decreases significantly when the moisture content of coal increases. However, if the gas content of dry coal is known, the value for moist coal can be approximated using the Ettinger formula (Boxho et al, 1980):

$$\frac{Q_{ads}\ (\text{moist})}{Q_{ads}\ (\text{dry})} = \frac{1}{1 + 0.31\ H}$$

where H = moisture content of coal expressed as a percentage.

Also, the gas content of coal decreases with increasing ash since it is the carbonaceous matter in coal that adsorbs methane.

Adsorption experiments are normally carried out on moisture-free coal. To relate the results to practical conditions, corrections are made for temperature and moisture. Since in coal mining situations, gas pressure seldom exceeds 4.14 MPa (600 psi), experiments were carried out up to this pressure.

Permeability decreases with increasing rank of coal and permeability is markedly reduced by stress.

Permeability refers to the ability of the coal to transmit gas when a pressure gradient exists across it (the coal). Kissell (1975) has reported that the flow of methane is partly controlled by the degree of water saturation of the coal seam. The permeability to methane increases as the water content of the coal decreases thus allowing more pore space available to the gas phase. This assumption is based on the observation that as water flow in an in-seam hole decreased the gas flow increased.

In a seam with low permeability, gas flow rates are very low. High rank coal leads to difficulties in methane drainage.

8.2.3 Seam gas pressure and temperature

Williams (1960) in discussing mine gases states:

"The pressure at which firedamp is contained in coal seams underground has been the subject of many experiments, most of which were largely unsuccessful due to leakage from boreholes, through breaks in the coal and its associated strata, and the low permeability of the coal. About 1880, Lindsay Wood (a prominent mine viewer) and others measured pressures in underground boreholes varying from 116 lb/in^2 (810 kPa) to 461 lb/in^2 (3221 kPa). In later years very few cases of high pressure have been recorded although at Point of Ayr Colliery pressures as high as 300 lb/in^2 (2095 kPa) have been registered. Further manifestations of high pressure are sometimes to be found in the occurrence of outbursts of gas and coal. Recent workers in Holland have recorded firedamp pressures of 15 to 225 lb/in^2 (100 kPa - 1571 kPa) in boreholes. Graham developed a method of measuring the pressure of methane in coal from a small coal sample which gave pressures from 42 to 127 lb/in^2 (293 kPa - 887 kPa)."

The relative importance of knowing the seam gas pressures and temperatures is to be able to use the equilibrium sorption isotherms to predict the quantity of gas held in a certain seam or area of a mine. Phillips, Waite, Billenkamp and Stripp have recently developed a prototype packer for measuring borehole gas pressures. The results of these trials are not yet to hand. These authors also state that results of field trials have indicated that bituminous coals are highly permeable while the anthracites are generally much tighter and have a lower permeability. All the recorded methane outbursts in South Africa have occurred in burnt coal zones which have a low permeability - this fact makes it extremely difficult to determine seam gas pressures and quantities of methane contained in the seam with the final objective of trying to predict possible methane outbursts. Evidence given at Inquiries has all indicated that no methane was detected in the drill holes in the faces prior to the sudden outbursts.

8.3 GEOLOGY OF THE COALFIELDS

Both the South Rand and the Indumeni coalfields exhibit similar geological features, most of which are outlined in this section.

8.3.1 South Rand Coalfield

(a) Structure

The coalfield exhibits varied coal seam flow topography, which is further complicated by two buried pre-Karoo hills within the basin. The attitude of the seam varies between flat-lying in the centre of the basin, to dips of 5^{o} and more around the edges due to compaction settling after deposition. In addition, due to seam downwarping, numerous faults disrupt the seam around both the margins of the basin and the buried hills. Major faults which trend in an approximate East-West direction have throws of up to 40 metres. A significant number of minor faults are also encountered.

(b) Dolerite Intrusions

A coarsely crystalline dolerite sill up to 100 metres in thickness overlies most of the colliery lease area, but is absent in the east due to erosion.

Numerous dolerite intrusions exist throughout the area and vary in thickness from a few centimetres to over 20 metres; they vary in attitude between vertical and almost horizontal with generally no directional trend. Along the strike of dolerite dykes, the thickness, dip and direction vary over short distances and to date exploratory drilling to supply the full advance information needed for mine planning has not been successful.

Dykes and sills cause unstable conditions in thick seam areas where the coal may be burnt for some distance on either side of the intrusion. Roadways driven in burnt coal require heavy support to prevent roof and sidewall collapses.

Surface magnetometry has been unsuccessful in locating dykes due to the masking effect of the thick dolerite sill, but modern in-seam seismic techniques could possibly be used in future.

8.3.2 Indumeni Coalfield

The quality of the coal at Indumeni varies from fairly high quality coking through to anthracite depending upon the extent of devolatilisation that has taken place. Where unaffected by dolerite, the coal contains roughly 23-26% volatiles and produces, on washing, a coking coal with a 5 or 6 swell. When partially affected by dolerite, the volatile content would drop to the range 16-20% and yield only a bituminous steam coal. In the areas where the coal was most severely affected, the volatiles dropped to less than 12% yielding an anthracite coal of moderate quality.

Devolatilisation of coal at Indumeni took place on both a regional and local scale, depending upon the intrusive agent causing the effect. On a regional scale the agent causing the progressive devolatilisation of the coal was the Ingogo Dolerite Sill which occurs above the Main Coal Zone. When this sill occurs at more than 50-60 metres above the coal, no noticeable effect is felt, however, as the sill transgresses the sediments above the coal and approaches the coal seams, the volatile content of the coal is reduced until a point when the coal is effectively a meta-anthracite. Devolatilisation of this nature does not cause the coal to lose its internal structures or its inherent strength, in fact as the volatiles are reduced, the coal effectively gets harder and is at all times capable of being mined.

On a local scale many dolerite dykes were encountered at Indumeni and these naturally had a significant effect on the coal. These dykes varied in thickness and in orientation and three distinct trends could be noted. Significant differences in the effect of the dykes on the coal was not noticeable with regard to the various trends of the dykes but was definitely noticeable in relation to thickness. The thicker dykes invariably caused more problems than the thinner dykes.

Near the major dykes that were encountered in mining, a typical sequence of effect on the coal could be identified. The first noticeable effect on approaching a dyke was that the coal became brighter and harder indicating the first loss in volatiles. Getting closer to the dyke there was an increase in the amount of calcite occurring on the cleat and the coal became progressively duller and more granular until finally the coal lost all its inherent structure and strength becoming an amorphous mass of very dull shaly burnt coal. On contact with the dyke the coal would be totally converted to a grey black ashy material.

8.3.3 Geological similarities

- The coal measures are relatively deep compared with South Africa collieries - ± 200 to 300 metres.
- The seams are gassy.
- The coal measures are overlain by thick (± 60 metres) horizontal dolerite intrusions in the form of sills.

- Dykes and sills emanating from the major sill and varying in thickness from a few centimetres to over 20 metres have burnt the coal seams for some considerable distance and cause unstable conditions which require heavy roadway support. It is in these zones that the methane outbursts have occurred.

8.4 INCIDENTS/PHENOMENON

Outbursts

This is the name applied to the violent evolution of firedamp (usually together with large quantities of coal dust) from a working face. Outbursts are known wherever coal is worked. Some of the largest have occurred in France. In 1907 the Nord d'Alais coalfield an outburst of carbon dioxide blew out 4000 tons of coal. At L'Agrappe Colliery, Belgium, in 1879, 520 tons of fine coal were blown out and firedamp burned at the pit tops with flames over 100 ft (30 metres) high. The largest outbursts in Great Britain occurred at Ponthenry Colliery in the South Wales anthracite area, where in 1920 an outburst involving 270 tons of coal occurred with such violence that the airflow was temporarily reversed in the mine.

Certain features are common to most outbursts:

- The outburst normally occurs in narrow workings or short faces.
- The outburst frequently occurs at or near some geological disturbance, such disturbance usually causing irregularities in seam thickness.
- Before the outburst take place the coal being worked becomes very 'lively', small lumps often being 'shot' off the coalface.
- The outburst of gas always involves an outburst of coal which is invariably very fine.

- The area of the mine where the outburst occurs is dry.

Table 8.1 lists the major incidents of instantaneous methane outbursts known to have occurred in South African collieries.

TABLE 8.1

Location	Year	Seam Gas	Vol matter %	Rank of coal	Fatalities
Natal Cambrian	1942	CH_4	Not known	High	Nil
Natal Cambrian	1942	CH_4	Not known	High	Nil
Natal Cambrian	1943	CH_4	Not known	High	Nil
Natal Cambrian	1943	CH_4	Not known	High	Nil
Natal Cambrian	1943	CH_4	Not known	High	Nil
Springfield	1954	CH_4	12	High	Nil
Springfield	1963	CH_4	14	High	Nil
Springfield	1969	CH_4	10-14	High	Nil
Springfield	1985	CH_4	13	High	Nil
Springfield	1989	CH_4	10	High	Nil
Indumeni	1970	CH_4	9	High	Nil
Indumeni	1970	CH_4	10	High	Nil
Indumeni	1971	CH_4	9	High	Nil

8.4.1 Springfield Colliery

At least five incidents of instantaneous methane outbursts have occurred in the South Rand coalfield.

Figure 6 shows a plan of the workings of the colliery in this coalfield and the area in which the five incidents have occurred. In every case these methane outbursts have occurred in an area where dykes and burnt coal zones are present: the outbursts have also all occurred in the northern section of the coalfield where geological disturbances are more prevalent and where the coal seams are relatively thick.

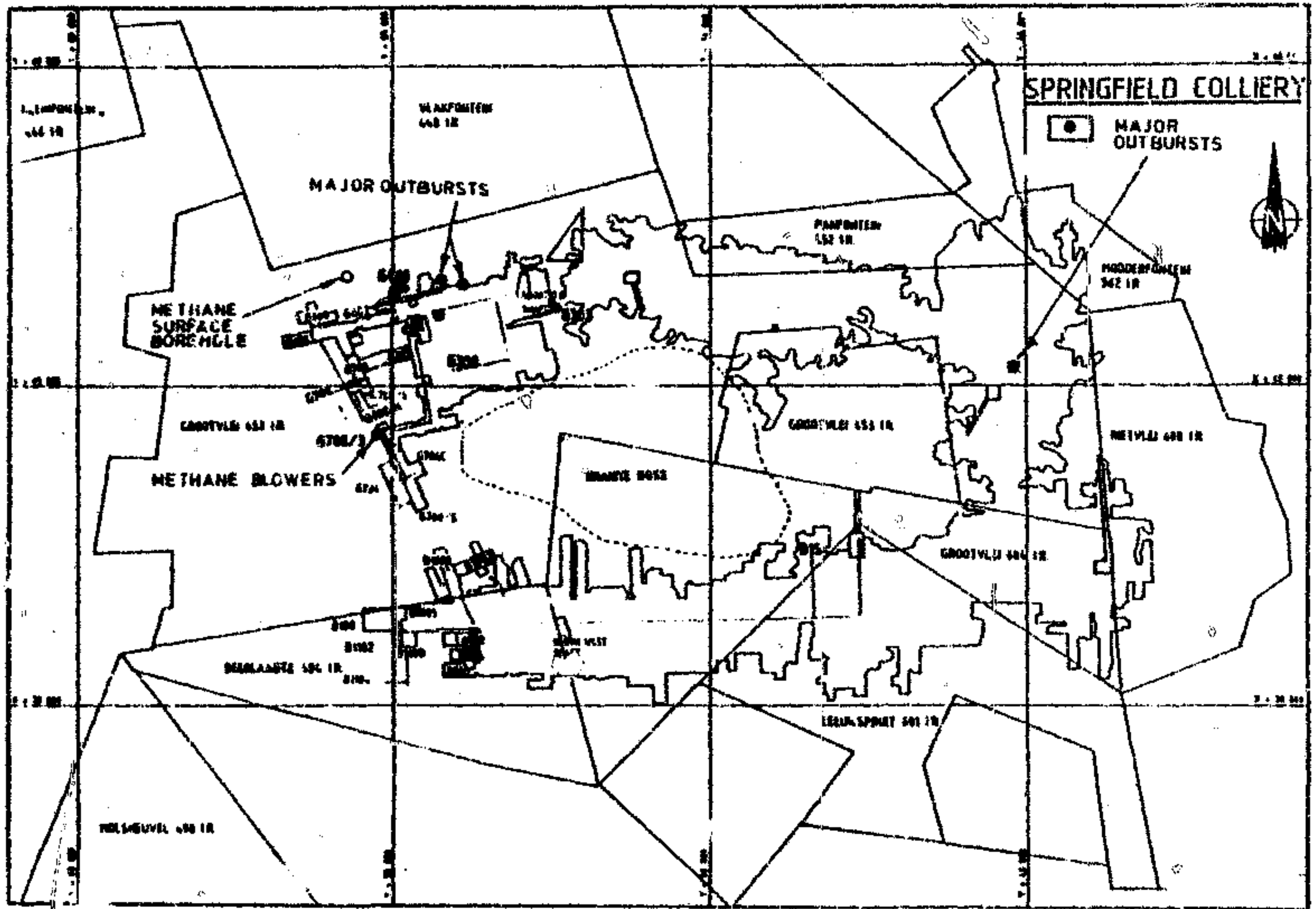


Figure 6
GENERAL PLAN OF SPRINGFIELD SHOWING THE POINTS OF MAJOR OUTBURSTS

Of particular importance are the three most recent incidents, the first of which occurred on the 2nd October 1969 in Section 7 off the Main West haulage and which incident is the most accurately recorded and the second which occurred on the 3rd December 1985 in a surface borehole which was being drilled to determine geological data of the coal seam and lying to the North of the Grootvlei main haulage (see Figure 5). The third incident occurred in a drive north of the Main West haulage in March 1989.

The evidence of a shiftboss who was in charge of a section in the Rietvly area of the mine in 1963, when an instantaneous methane outburst occurred is as follows:

"I am a Training Superintendent at Springfield Colliery. During 1963 I was the shiftboss in Section 5 of the Rietvly area, a part of this mine. We had just blasted through a dyke. The miner put sliping holes into the rock. When he blasted these sliping holes a similar occurrence as this one took place. No people were injured.

The outburst was accompanied by a great deal of methane gas and very fine dust. I estimate about 100 tons of coal was blown into the roadway.

In this instance, all the coal was loaded out and we reached the dyke and blasted the coal between the two dykes.

The roof in the outburst zone after the blast was approximately 5 metres high."

The most detailed documente incident occurred in 1969 in Section 7 off the Main West haulage. Figure 7 shows the plan of the section where the incident occurred and Figure 8 a borehole section of the coal seam and strata in the affected area.

Much difficulty had been experienced in the past in developing the coal reserves lying to the North of the Main West haulage.

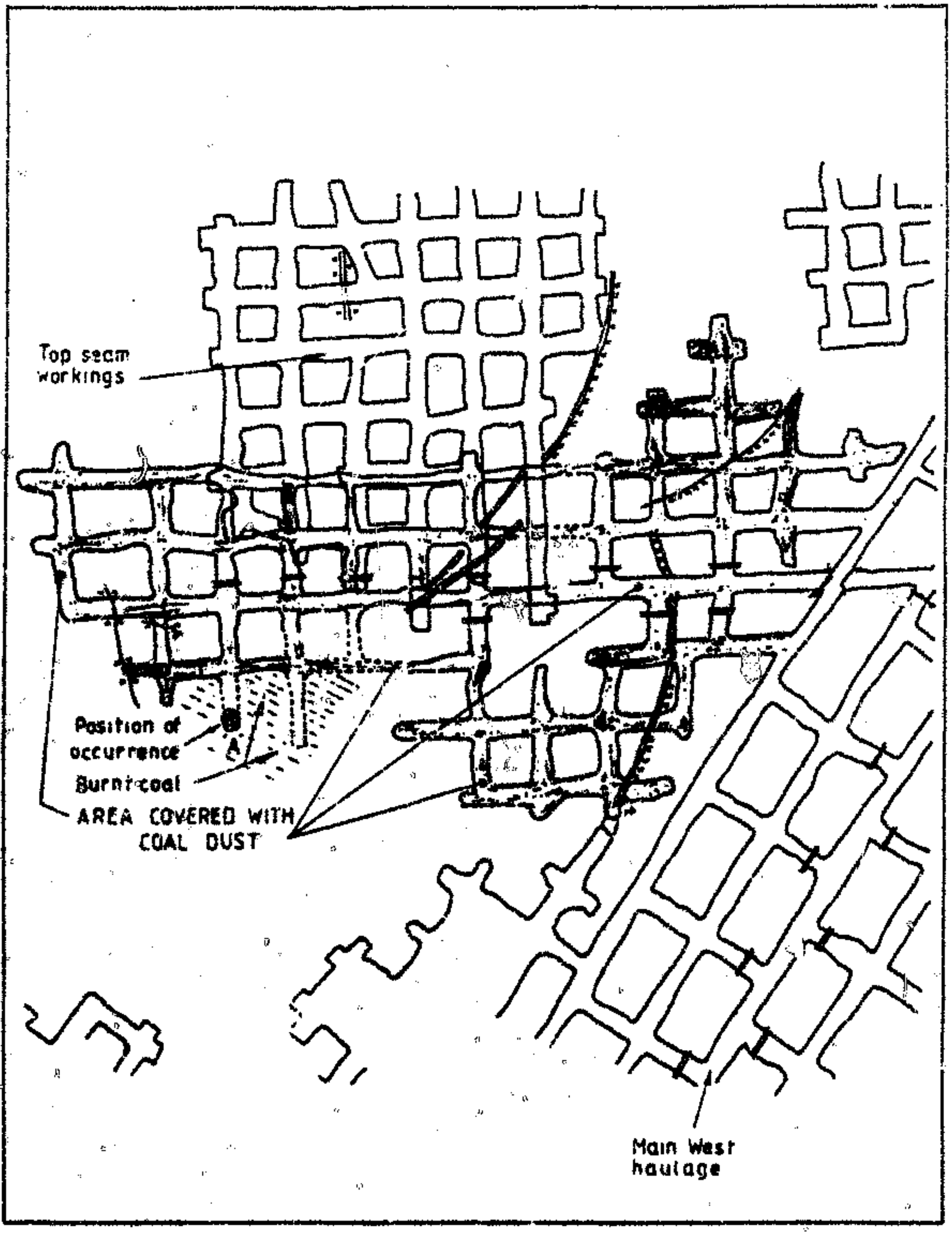
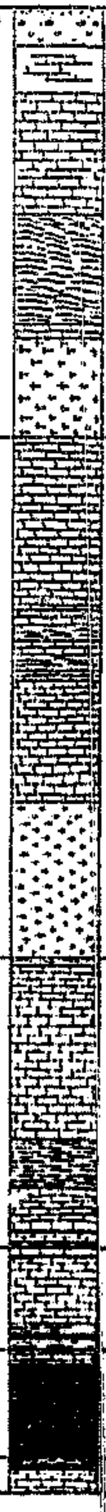


Figure 7

PLAN OF THE SECTION SHOWING THE AREA AFFECTED BY FINE COAL DUST AS A RESULT OF AN OUTBURST



Surface	Depth Ft	Width Ft	Description	From Ft	To Ft
-935·20	15	15	Soil, Pebbles & clay	0	15
	35	20	Sandstone - decomposed	15	35
	85	50	Sandstone - Firm	35	85
	130	45	Shales - Sandy micaceous	85	130
	136	6	Sandstone	130	136
-1113·20	178	42	DOLERITE - Broken	136	178
	230	52	Sandstone - Coarse White	178	230
	250	20	Sandstone - mica streaks	230	250
	280	30	Sandstone & Shales - laminated	250	280
	330	50	Sandstone	280	330
-1330·20	395	65	DOLERITE	330	395
	397	2	Shale	395	397
	467	70	Sandstone - dark mica bands	397	467
	489	22	Shales - muddy, light & dark bands	467	489
		18	Sandstone	489	507
	508	1	Shale - streak of coal	507	508
		4	Sandstone	508	512
-1449·20	514	2	COAL & Shale leader	512	514
	546	32	Sandstone - ?coal at 517?	514	546
-1492·20	557	11	Shale & COAL - Carbonaceous - Cored badly	546	557
	562	5	Sandstone - Coarse	557	562
		40	COAL	562	602
-1537·20	602	1	Sandstone - Gritty	602	603
	605	2	Shale & Coal bands	603	605
	615	10	Sandstone	605	615

Figure 8

BOREHOLE SECTION IN VICINITY OF OUTBURST

See Figure 8a for detailed seam section

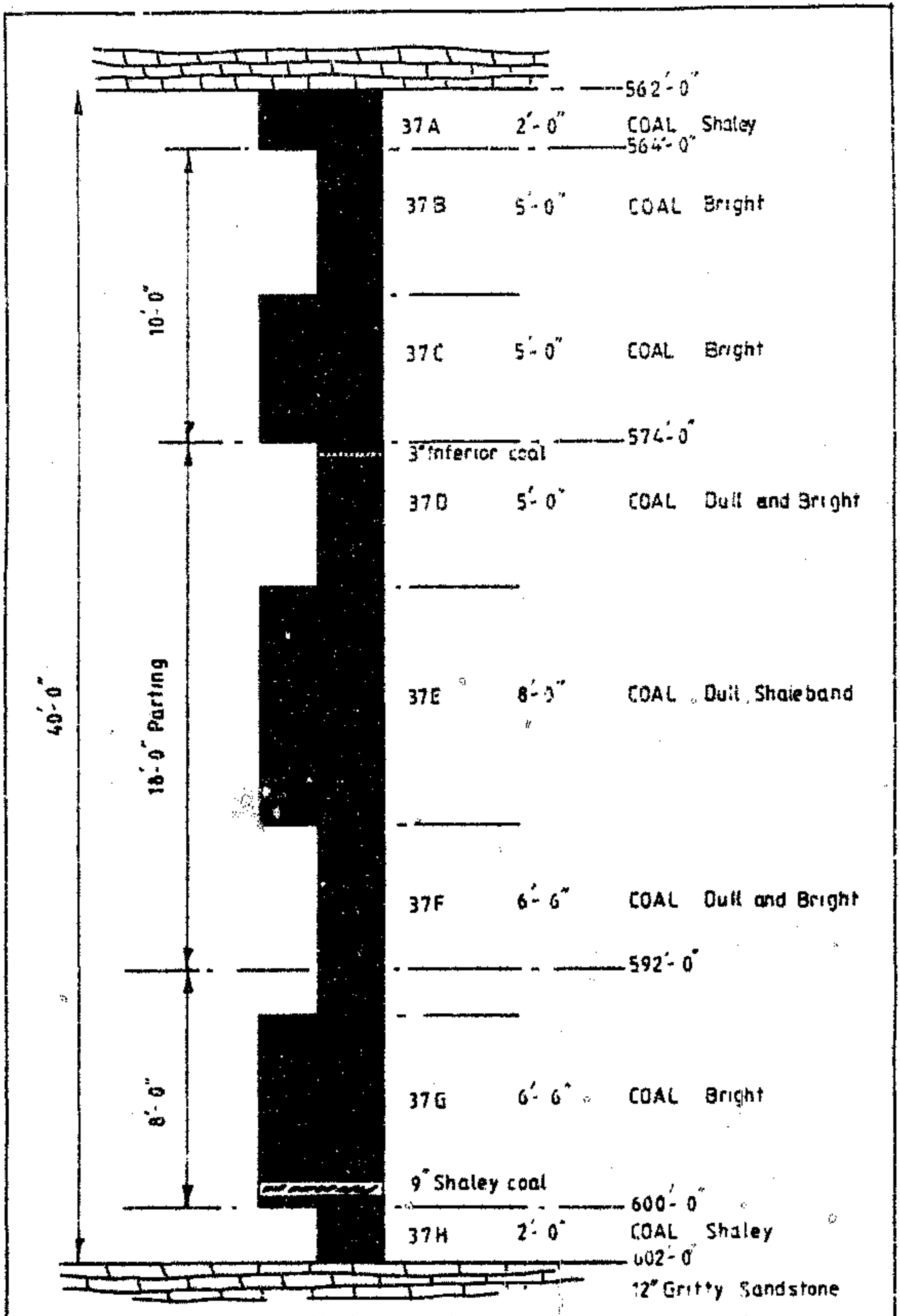


Figure 8a

DETAILED SECTION OF COAL SEAM

In 1969 a decision was taken to attempt to develop these reserves by driving drifts to the floor of the coal seam under previously worked areas in the upper section of the seam - from the floor of the seam the plan was to then develop to the north and establish an area of reserves.

The roadways off the left hand companion encountered a massive burnt coal zone and it was necessary to develop these drives to establish a return airway.

While blasting a cut in roadway "A" shown on plan Figure 7 an instantaneous methane outburst occurred. The author's evidence of this phenomenon is as follows:

"I am the manager of Springfield Colliery, a fiery coal mine in the district of Balfour. On 2nd October 1969, it was reported to me there had been a outburst in section 7 at about 1.30 p.m. on the same day. It was reported to me that Miner F.J. Schuler had been overcome by gas but that he had been brought to safety by Shiftboss van Zyl. Miner Schuler did not stay off work at all.

Likewise the section Gangleader gathered the section men and brought them out of the section. By 3.30 p.m. it was reported that all persons in the section had been accounted for. The electric power to the section was isolated and the section was fenced off.

Arriving in the section the next day there was evidence of a considerable amount of fine coal dust deposited on the roof, sides and floor of the main and also on various pieces of equipment in the section. Apart from a pump in the left-hand companion which has been buried by the loose coal blown out from the face and a ventilation tube which extended into the face where the outburst occurred, there was no evidence of any damage to any of the equipment or appliances in the section. Samples of the coal dust found in the main drive were taken with the object of discovering its explosiveness, its ash content, B.T.U. and particularly its volatile content." Figure 9 depicts details of the roadway where the outburst occurred.

With particular reference to the first left split off the main where the outburst occurred, loose burnt coal had been blown back to within 11 metres from the main drive. All timber and girder supports had been blown out and there was evidence of falls of roof in the affected drive.

There was also evidence that the roofbolts had been damaged.

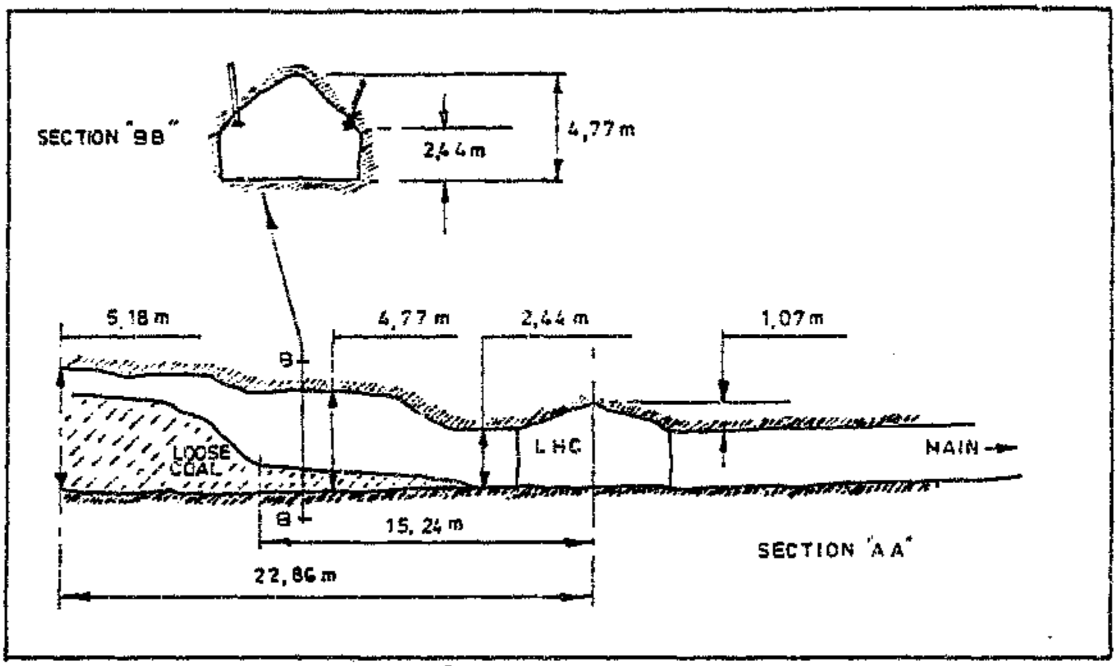
The left hand companion constitutes one of the return airways of the section. There was evidence of a 0,6 metre layer of methane gas being emitted from the face where the outburst occurred and this gas was bleeding at a speed of about 10 metres per minute into the main return airway.

A dräger test in the layer of contaminated air revealed about .001% of Carbon Monoxide. There was no evidence of coal dust in any of the workings approximately 15 metres north of the drive shown on section A-A. In my opinion this revealed that the force of the blast obviously spent itself out into the main and return airways.

On the morning of the visit to the section, instructions were given for reclamation of the section immediately with the following objectives in mind:

a. To load out all loose burnt coal in order to prevent a fire.

b. As a result of a. to establish the geological features which gave rise to the outburst of gas which produced about 300 tons of burnt coal.

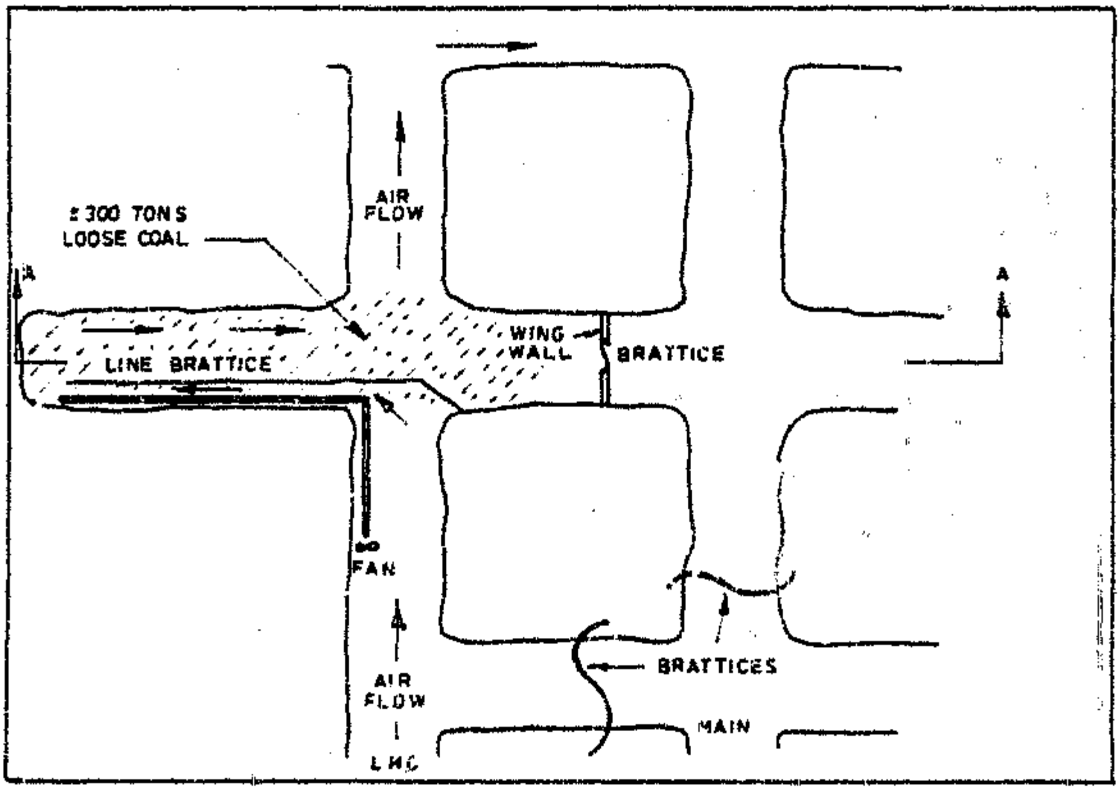


Sections

Figure 9

PLAN AND SECTIONS OF ROADWAY SHOWING METHANE OUTBURST OCTOBER 1969

Plan



c. To establish exploratory work and formulate instructions to anticipate such outbursts in massive burnt coal in the future and thus prevent such an occurrence again.

Accordingly, with the above objectives in mind the section was recovered by Monday the 13th October, 1969.

The roof inbye of the left-hand companion appeared to have been blown down as its height was now between 5 metres to 5,5 metres. Some of the height could be attributed to damaged roofbolts which resulted in small falls of roof taking place. Methane gas was still being emitted from the top of the coal."

No signs of an ignition of methane gas was evident but the area over which fine coal dust had been deposited is drawn on the attached plan of the section Figure 7. Power to the section had been isolated after the outburst. The miner who blasted the face where the outburst occurred was overcome by the methane in the section and was rescued by the shiftboss.

Relatively large quantities of methane were being given off from the drive where the outburst occurred after the incident. From the evidence it may be concluded that approximately 2,7 cubic metres per minute of methane entered the airflow in the section of some 240 cubic metres per minute. Assuming that the methane content in the flow from the pocket was 100% the methane content in the return airway would have been 11% - an explosive mixture. In fact no methane was detected in the return airway an hour after the blast thus indicating that the bleed of gas from the outburst site was not 100% methane. Figure 9 depicts a section through the drive where the instantaneous outburst occurred. Some 300 tons of burnt coal was deposited in the drive.

Attempts to load out the loose burnt coal were eventually halted when spontaneous combustion took place and by the 13th October 1969 the split had been sealed off with a fire stopping just inbye of the left hand companion. Burnt coal areas at this colliery are known to be liable to spontaneous heatings.

It was customary at this mine to cut the burnt coal faces using a coalcutter - the depth of the cut normally varied according to the roof and sidewall conditions. In this particular instance, the cut was 1,8 metres deep and three holes were drilled above and below the cut. Each hole was loaded with 600 grams of Ajax permitted explosive and the face blasted all holes simultaneously.

The roadway where the methane outburst occurred was 3 metres wide and 1,8 metres high; such dimensions reflect the poor ground conditions and the need to keep the roadway narrow to prevent falls of roof.

Normal face ventilation practice at the colliery was to use 37 kW fans delivering air to the faces through 60 cm diameter ducting which was required to be at least 1 metre from the face which was the case in the affected heading. In addition, line brattice was installed to within 2 metres of the face.

No methane gas had been detected in the section prior to blasting the affected heading.

Stonedusting was not practiced at the colliery until after the Wankie disaster in June 1972 and the general method of controlling dangerous dust was to water down all roadways.

It was fortunate that no igniting source was present during and after the outburst.

Explosibility tests of the burnt coal taken from the affected roadway are recorded as follows:

Minimum ignition temperature determination - 850°C which is approximately 200°C higher than that for normal bituminous coal.

Proximate analysis

Moisture %	3,7%
Ash %	24,3%
Volatile matter %	10,3% - 14,1%
Fixed carbon %	62,0%
Calorific value	23,19 - 24,07 MJ/kg

The comparatively low volatile matter of the sample explains the insensitivity of the material.

The Kex index for coal from the South Rand coalfield is between 70 and 100, while the Kex index for burnt coal is approximately 50. Joubert (1981) has stated that coal samples with a Kex value of 60 or below refused to explode at dust concentrations as low as 200 g/l.

The fact that massive burnt coal at the Grootvlei shaft presents a danger due to instantaneous methane outbursts is supported by another incident in September 1982. The third left hand companion (Figure 10) of the section was being worked backover to hole with the advancing heading which had been stopped on a dyke. The holing was necessary to secure a return airway on the left hand side of the section. A section through the coal seam is shown in Figure 11. The roof, floor and sides of the roadway were being worked in burnt coal.

Mining in the No. 2 road to the east, was advancing towards a dolerite sill. Arch girder sets were being used to support the roadway roof and sides.

In order to install the arch girder set No. 36, the miner blasted three drill holes, using 1 200 grams of explosive in total in the bottom left hand corner of the face at 08h30. No methane was present before blasting. After waiting for 10 minutes for the blasting fumes to clear, the miner returned to inspect the results of the blast. When he was 20 metres from the face, there

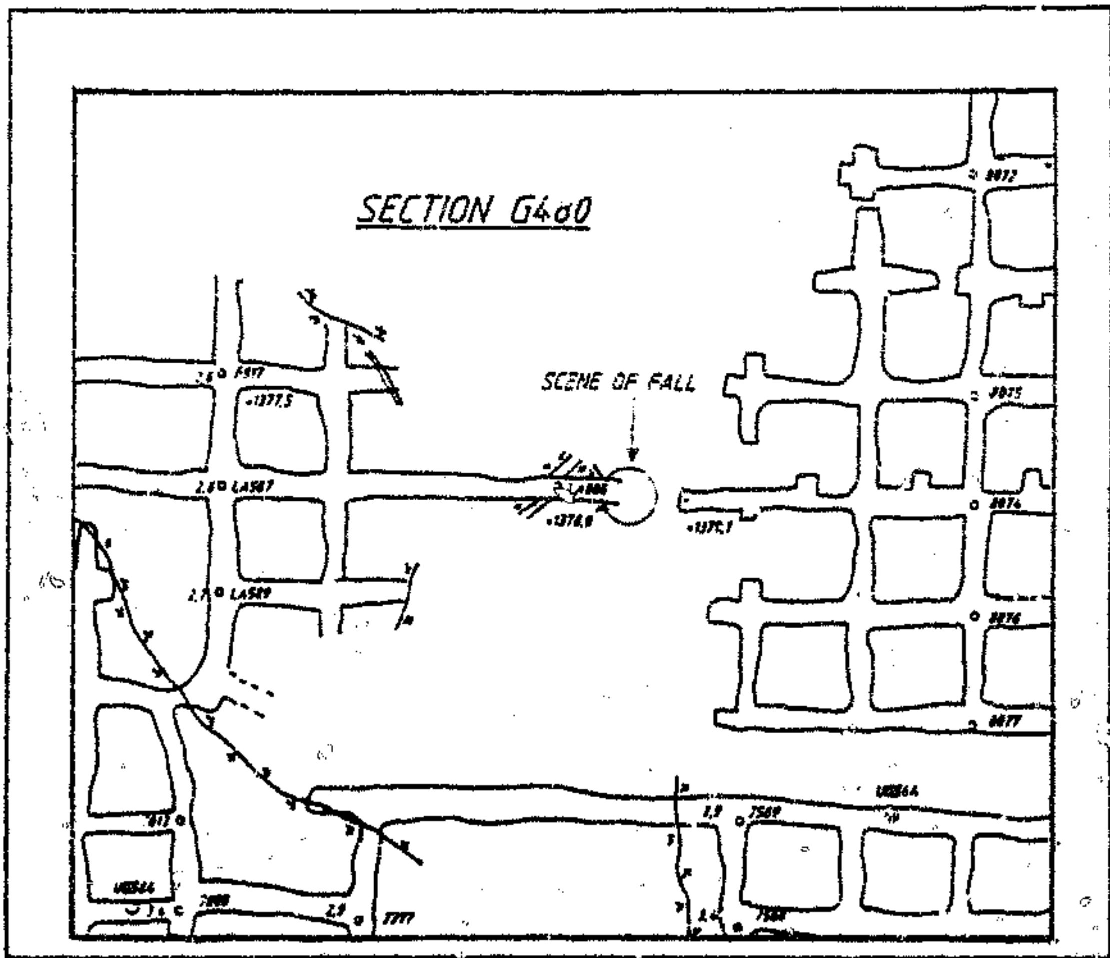


Figure 10

PLAN OF GROOTVLEI SHAFT-SECTION G480
SHOWING LOCATION OF OUTBURST MARCH 1989

It will be observed that the gas concentrations behind the seal passed into the explosive range during samples 6-7-9 and all persons were withdrawn from the ventilating district during this period. The area had been adequately stonedusted.

8.4.2 Indumeni Colliery

On the 24th February 1971 at 18h14 a violent instantaneous methane outburst occurred in a burnt coal zone in Section 700 at Indumeni Colliery - 10 kms south of Dundee.

Figures 13, 13a and 13b show a typical borehole section and seam section adjacent to the affected area.

Figure 14 shows the 700 main haulage in relation to the shaft and also the status of the ventilation arrangements at the time of the incident.

Some of the evidence given to the Chief Inspector of Mines at the Inquiry was as follows:

"On the 25th February 1971, it was reported to me that a large emission of inflammable gas had occurred during the afternoon shift of the 24th February 1971 whilst blasting was taking place in the second left companion of Section 700 at Indumeni Coal Mines Limited. It was reported that there were no casualties.

At the time of the emission the following persons were in charge of Section 700:

J.S.B. Erasmus	Mine Overseer
W. Henning	Shiftboss
H.J.H. Wessels	Certificated Miner

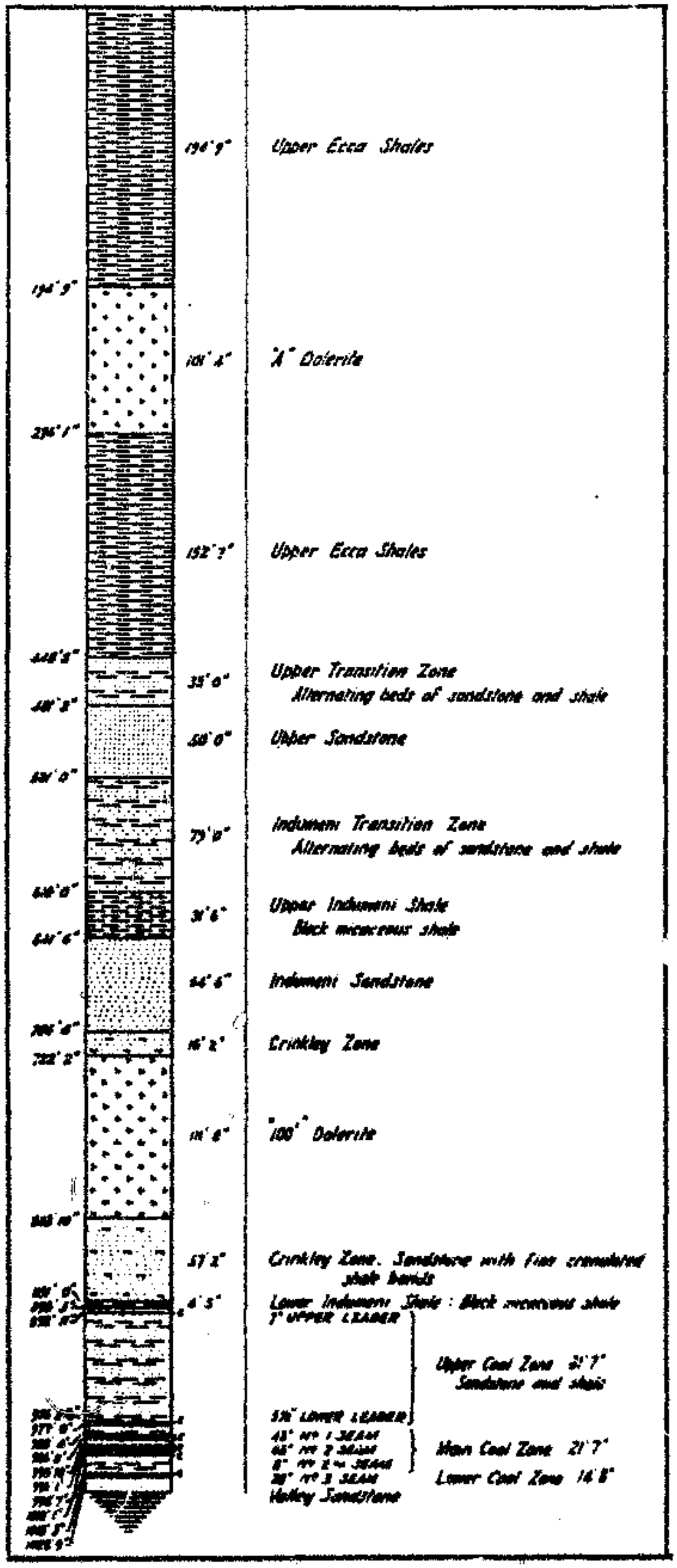


Figure 13a

TYPICAL BOREHOLE SECTION ADJACENT TO
METHANE OUTBURST
INDUMENI COLLIERY

See also Figure 13b

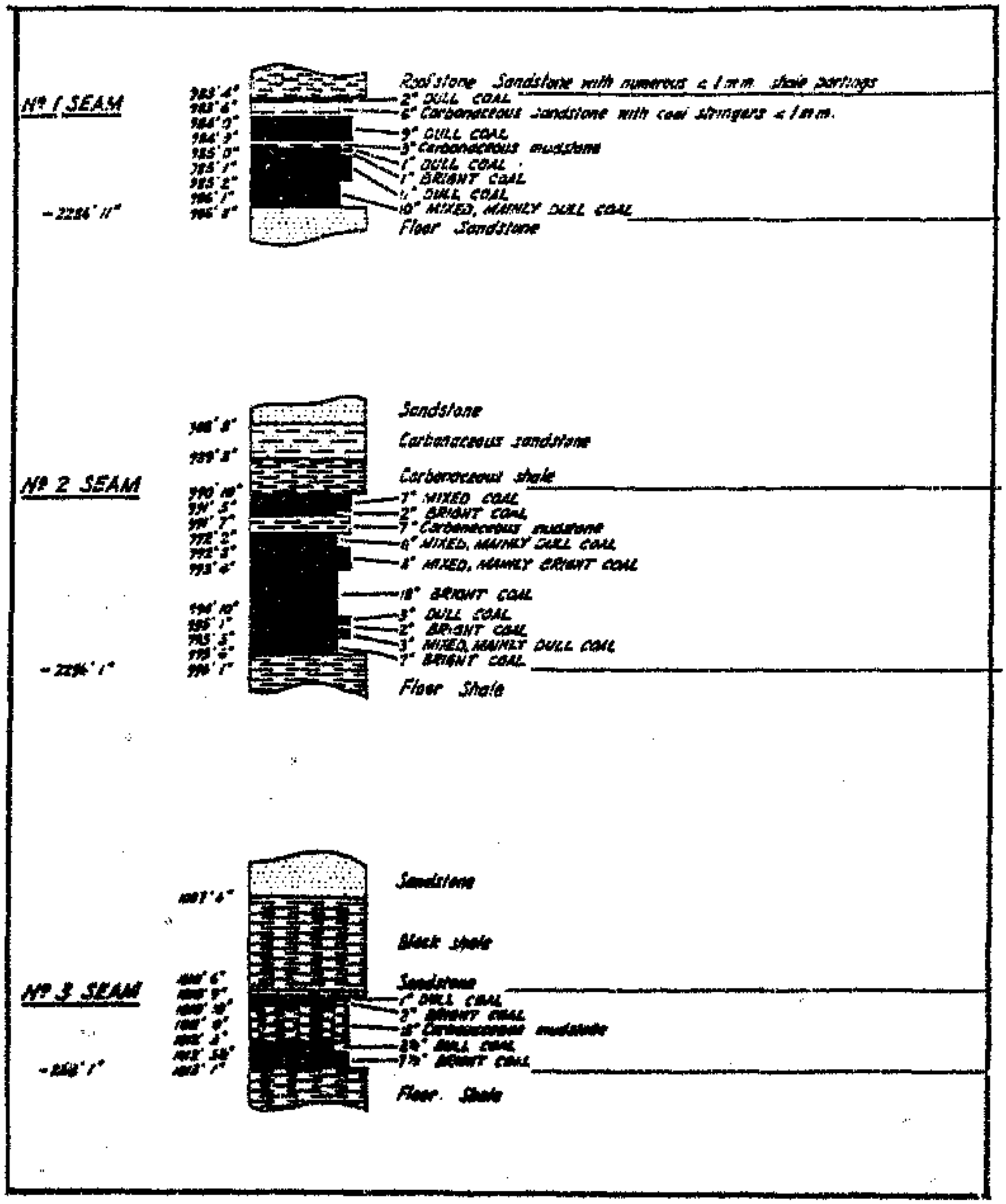
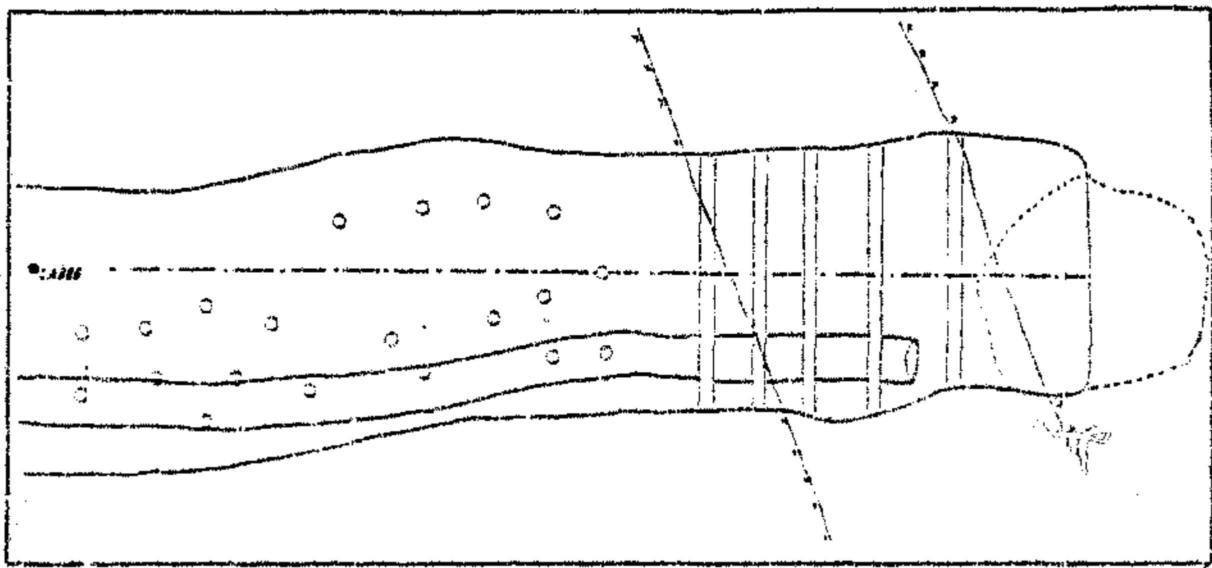


Figure 13b

COAL SEAM SECTION INDUMENI COLLIERY

See also Figure 13a

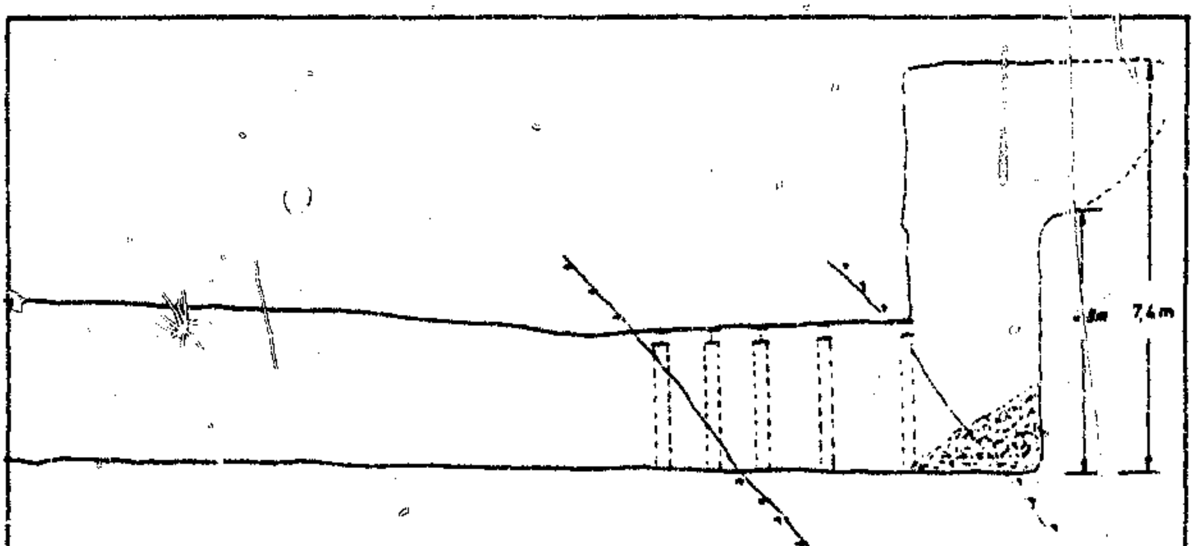


Plan

Figure 11

PLAN AND SECTION OF OUTBURST IN SECTION G480

Section



was an outburst of methane gas from the roof which dislodged approximately 200 tons of burnt coal. This coal lay 16 metres back from the face. The miner was not injured.

Within 30 seconds of the incident occurring, the Jeffrey continuous miner was tripped out by its methane detector at 2,5% methane. This machine was operating ± 240 metres from the outburst.

The miner tested for methane in the rest of the section with his MSA D6 methanometer and found ± 5% methane as far back as the section electrical switchgear and ventilation fans. Electrical power was immediately switched off at the switches and the substation.

To clear the section of gas, the shiftboss increased the ventilation flow from 26 m^3/s to 40 m^3/s by opening the section regulator. Within five minutes of opening the regulator, the gas concentration at the electrical switchgear had reduced to 1,5% methane. The section was cleared of gas by 11h15, some 2 hours 30 minutes after the incident occurred.

On investigation of the area after the incident it was noted that a cavity some 10 metres high had been formed above the last arch as indicated in Figure 11. The debris - mainly burnt coal - had been blown out into the roadway. The distance to hole into the No. 2 road remained at ± 4 m and it was decided that before any further excavation should be attempted the following work should be carried out:

- Drill exploratory holes to ascertain whether other pockets of methane existed between the two drives.

- Re-stonedust the area.

An analysis of the methane/oxygen ratios in the roadway where the outburst occurred at the time of the outburst is clearly not practicable. On the assumption that at the instant of the outburst the roadway, Figure 7, is filled with an atmosphere of almost 100% methane and that at a point in the return airway 300 metres from the outburst the air analysis is as follows:

Oxygen		20,93%
Nitrogen	78,1)	
Argon	0,83)	79,03%
Carbon Dioxide		0,04%
		100,0%

then it may be concluded that at a point in time, as the section ventilation dilutes the methane, and at some point in the section the percentage methane/air mixture will pass into and through the inflammable and explosive limits.

Samples of the mine air in a section after an outburst revealed the following situation:

Table 8.2

Time after Outburst	Position in Section: Point of Outburst	100 metres from Outburst	250 metres from outburst - return airway
00h00	79% methane*1	5% methane*2	Not measured
02h00	5% methane	± 2.1/2% methane	Nil
20h00	60 centimetre layer of methane 5% CH_4 against roof (6-9 metres per minute velocity)		Nil

The absence of one or other sufficiently strong ignition source at some point in the section where an explosive atmosphere existed ensured that a methane/coal dust explosion did not occur.

*1 Firedamp blower

CH_4 - 79%
CO_2 - 0,9%
N_2 - 18,6%
O_2 - 1,5%
100,0%

*2 The atmosphere had become extinctive by diluting of the airflow with nitrogen and inflammable methane thus decreasing the percentage oxygen. Evidence was that tne miner, although not unconscious had collapsed and was dazed and weakened by the reduced oxygen content in the air.

Figure 12 depicts the result of a recent methane outburst in the South Rand Coalfield in March 1989. The outburst occurred 18 hours after a floor roll in the roadway had been blasted and while the mechanical loader was loading out the blasted material. Four men were affected in the face by the methane and made use of their self rescue appliances to return safely to the fresh air intake.

Photograph No. 1 shows the mechanical loader almost totally buried by fine coal which after 24 hours showed signs of spontaneous combustion (80 ppm of CO was detected under the boom of the loader).

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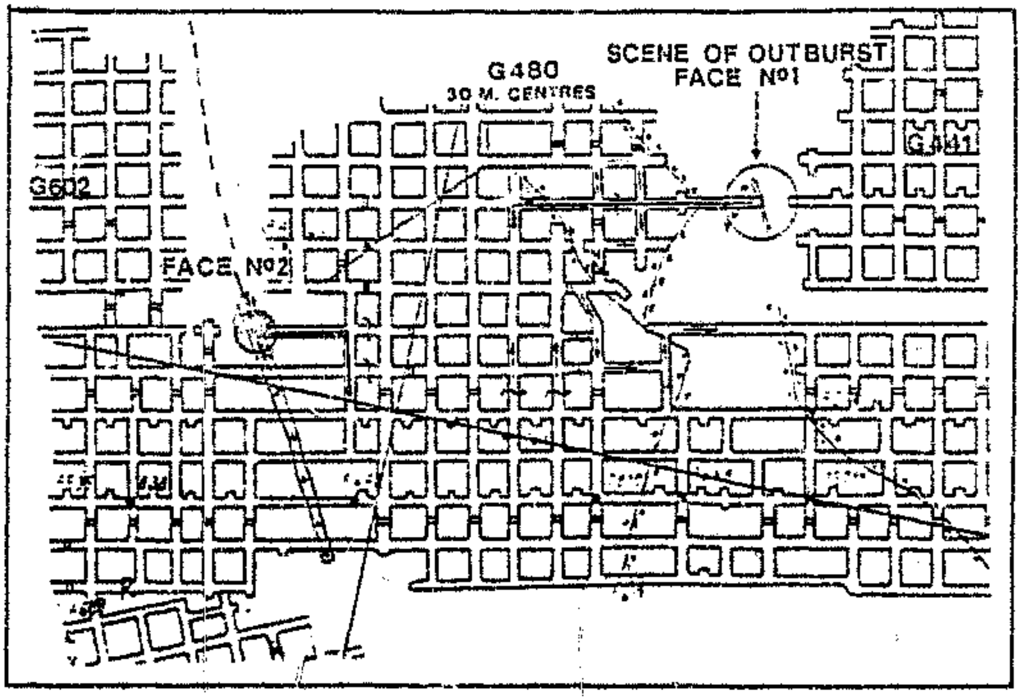
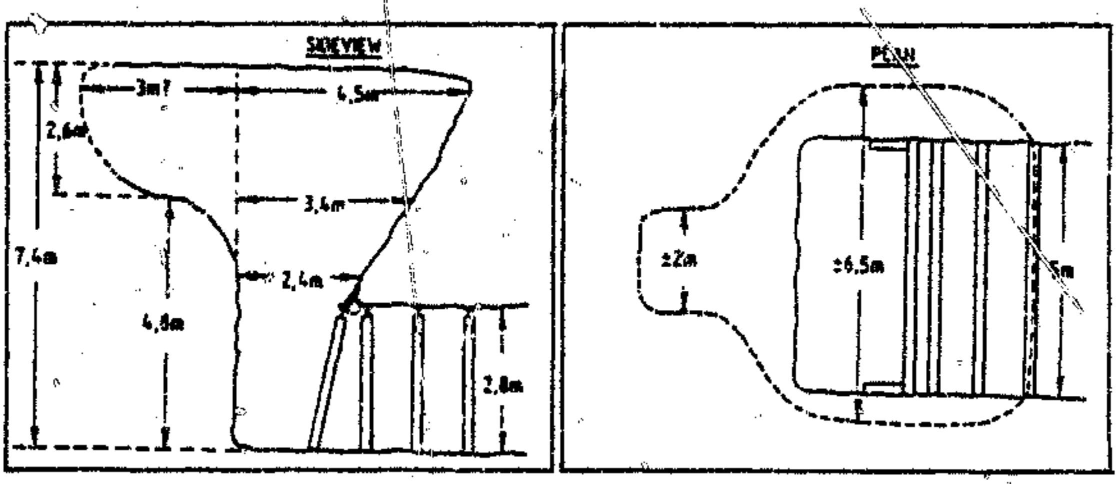


Figure 12

PLAN SHOWING AREA OF MOST RECENT OUTBURST
AT SPRINGFIELD COLLIERY



ENLARGED VIEWS OF SCENE OF OUTBURST

Table 8.3 shows the successive gas analyses after the roadway had been sealed.

TABLE 8.3

RECORD OF ATMOSPHERIC GAS ANALYSIS

Sample Number		1	3	6	7	9	10
Sample Date		890311	890311	890313	890314	890315	890316
Sample Time		16h26	09h15	14h17	11h55	13h13	12h59
Cylinder Press	kPA	550	500	550	500	550	700
02	%	20,6	19,6	16,7	16,0	13,7	12,5
CO	%	0,0015	0,0073	0,0107	0,0084	0,0072	0,0075
CO2	%	0,11	0,03	0,05	0,1	0,14	0,16
CH4	%	0,0	1,5	6,9	8,5	15,5	19,1
H2	%	0,0	0,0	0,0	0,1	0,0	0,0
N2	% Bal	78,5	78,0	75,5	74,5	69,8	67,4
R		0,0	1,0	1,0	1,0	1,0	1,0
Graham's Ratio		1,0	0,7	0,3	0,2	0,2	0,1
Young's Ratio		62,2	2,9	1,5	2,7	2,9	3,0
Willet's Ratio		15,2	0,6	0,2	0,4	0,4	0,4
X - Co-ordinate		0,8	4,0	12,5	14,2	18,25	20,4
Y - Co-ordinate		0,0	1,5	6,9	8,6	15,51	19,1

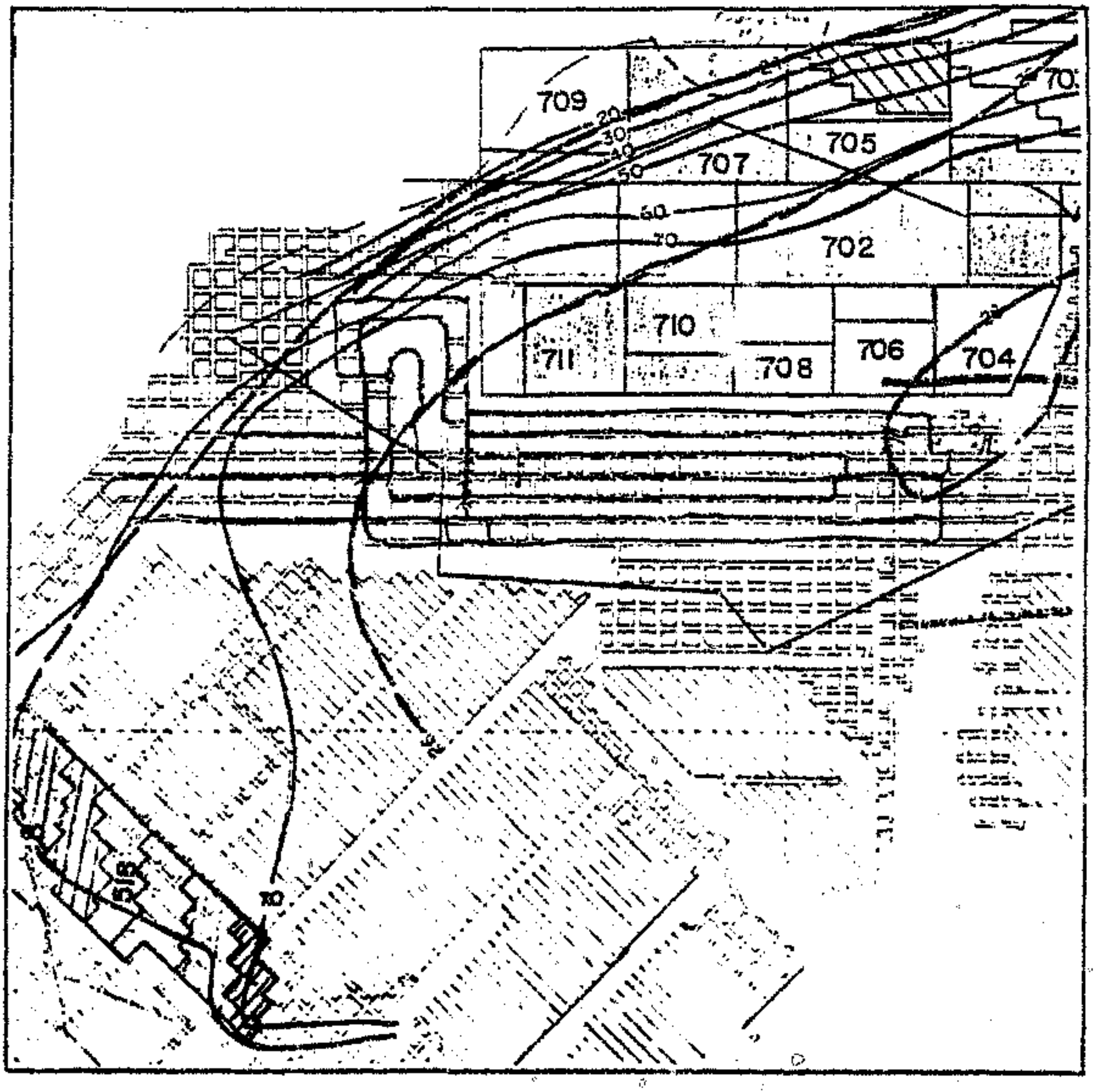


Figure 13

GENERAL LAYOUT OF UNDERGROUND
WORKINGS

INDUMENI COLLIERY

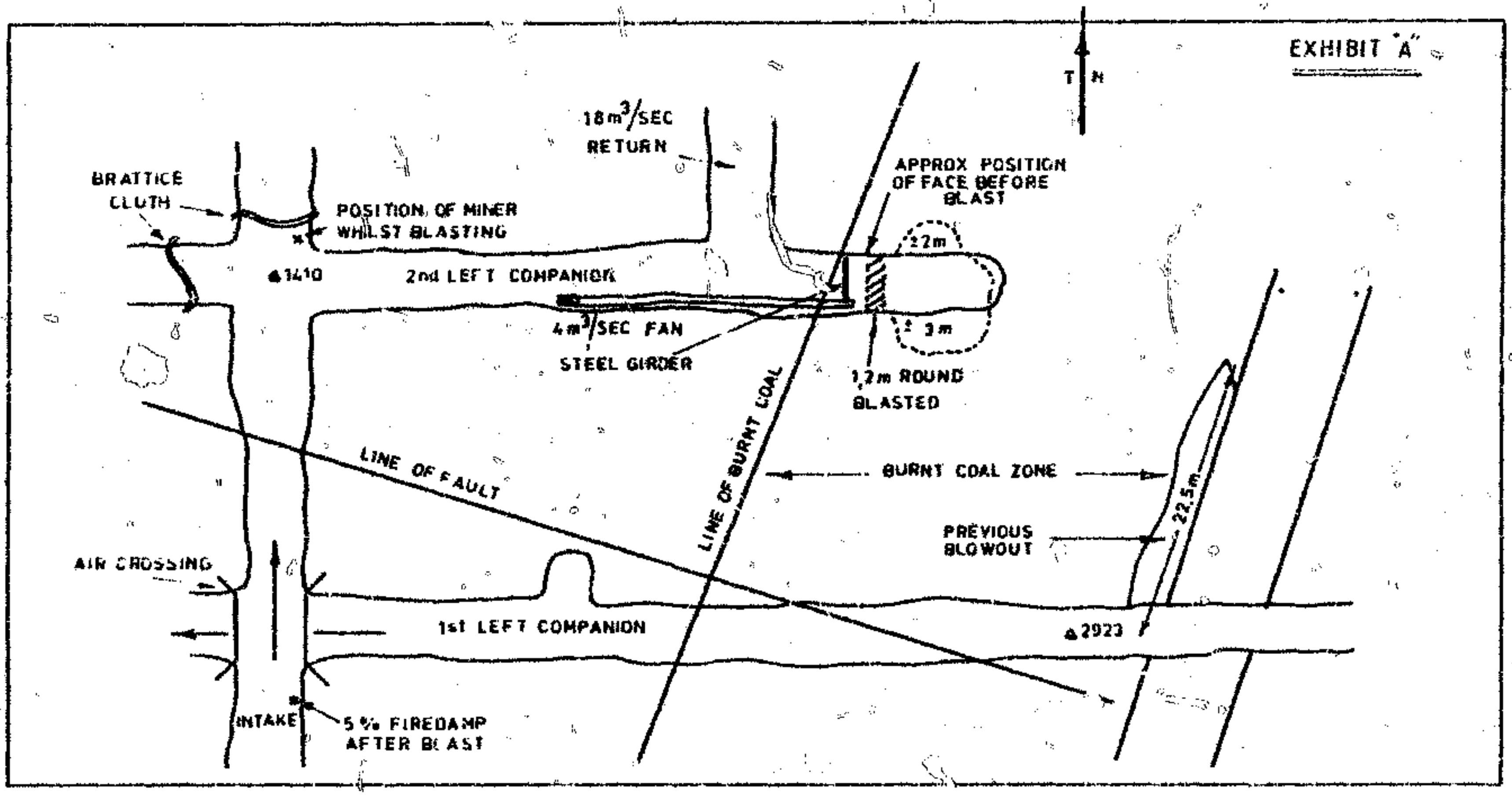


Figure 14

PLAN SHOWING TWO OUTBURSTS AND VENTILATION ARRANGEMENTS IN SECTION 700 OF INDUMENI COLLIERY

- The ground conditions in which the development roadways have been driven are highly stressed and tectonically disturbed; roof conditions have been poor and systematic installation of steel sections has had to be adopted to prevent serious falls of roof as the faces are advanced.

- The incidents have all occurred in relatively deep collieries - 150 metres to 230 metres below surface.

- The volatile content of the seam sections has averaged 9,6% with a range varying between 9% and 14%.

- The lower permeability of the coal is evidenced from the fact that prior to and after the outbursts little if any methane has been detected during normal operations in the vicinity of the stressed zones (in burnt coal).

- The affected areas are generally dry.

- As much as 900 m^3 of methane can be released in an instantaneous outburst and pressures of methane as high as 85 Bar have been recorded in the outburst zone.

- The ventilation system in the section is disturbed and temporarily reversed by the force of the blast.

- All outbursts have occurred almost immediately after a round in the face has been charged and blasted. (explosively induced outbursts or volley blasting).

- No methane ignitions have as yet occurred probably due to the absence of an ignition source.

- The outburst has been followed by lar[illegible] clouds of very fine coal dust which in all cases has proved to be below the explosive range.

- Workmen in the vicinity of the outburst have suffered from anoxia.

- Both roof and sidewall coal has been blown into the roadway - up to 300 tons and steel roof support has been blown down thereby encouraging further falls of roof.

- Spontaneous combustion of the loose and fine burnt coal has resulted.

- Little progress has been made to date in adopting anticipative and coal winning precaution measures with the exception of drilling pilot holes in the face prior to drilling the shot holes - in an endeavour to tap the entrained gas which is under high pressure.

On the 25th February 1971 I accompanied the Senior Inspector of Mines to Section 700 and investigated the incident and I had a plan of the area drawn up, a true copy of which I submit as Exhibit A. The air quantities were measured by the surveyor and the percentages of methane were those recorded by the shiftboss and miner on the night of the emission; ± 40 m^3/second intaking of which nearly 20 m^3/second ventilated the second left companion.

From my investigation into the incident it appeared as though the heading holed into a zone of ash and inflammable gas - the extent is shown in dotted lines on Exhibit A as it was not possible to measure the extent due to the state of the roof.

The roadway was being developed in burnt coal towards a 9 metre thick dyke which had been penetrated by four other roadways in the section. The face of the second left companion was about 24,4 metres from the dyke when the emission occurred.

During the inspection in loco water was being sprayed on the burnt coal, the exposed surfaces of which were at atmospheric temperature but when broken the surfaces exposed were warm.

Mr. Wessels told me that he had fired 16 x 1,22 metre long charges on the face of the roadway in two separate operations and as he blasted the last hole he looked along the roadway and saw a white cloud rolling towards him so he made his way towards the haulage but was overtaken by the cloud. The cloud travelled against the intake air for nearly 91 metres before stopping.

Mr. Wessels tested for inflammable gas at the point shown on the plan and found + 5,0% - he used a flame safety lamp. The cloud was quickly dispersed and when Mr. Henning arrived on the scene, he tested with a methanometer at the same point and recorded 3,0% CH_4. Mr. Henning went into the face and tested for gas with the methanometer and recorded 5,0% in the roadway with more than 25,0% in the face. The ventilation fan was restarted and when I inspected the roadway on the following afternoon, I found 0,2% CH_4 in the face. Normally these roadways are free from inflammable gas.

Mr. Wessels told me that after the cloud had been dispersed the place looked as though it had been freshly stonedusted - this was evidently ash.

From a normal 1,22 metre long round about 40 tons are broken but from this particular blast 216 tons were loaded. The area has been successfully negotiated and no more inflammable gas has been detected.

A similar emission of inflammable gas occurred about 8 months ago in the first left companion when the face was close to the dyke. The position of the cavity exposed after this emission is shown on Exhibit A. Both emissions occurred on the top seam.

A recording methanometer on surface recorded a rise from 0,8% CH_4 in the upcast air to 1,5% CH_4 over a period of 9 minutes - the percentage methane gradually fell back to normal over a period of another 12 minutes. The quantity upcasting is ± 400 000 c.f.m.

As a precautionary measure, 3,65 metre pilot holes are drilled with each round - this will be done until the burnt coal has been negotiated.

Hans Jurie Hendrik Wessels: Sworn states:

"On the 24th February 1971 I was the miner in charge of Section 700 at Indumeni Coal Mines Limited.

At about 6.40 p.m. on the same day after I had blasted 16 x 1,22 metre holes on the face of the second left companion, I looked towards the face and saw a white cloud approaching from the face so I told my assistants to leave and as I followed them to the haulage I switched off the electric current to the fan.

The cloud enveloped us and I smelt something like benzine in the air. There was nothing wrong with the ventilation and when the cloud reached the haulage it stopped and then dispersed rather rapidly.

When I tested for inflammable gas at the point shown on Exhibit A with a flame safety lamp the lamp was extinguished. The ash in the air settled and the place looked as though it had been freshly stonedusted.

Shiftboss Henning arrived on the scene about 10 minutes after the emission and he tested for gas and recorded 3,0% where my lamp was extinguished. After the gas was cleared we started up the fan and en route to the face detected 5,0% CH_4 in the general body of the air and +25,0% in the face. Shortly afterwards this gas was cleared and since that time until now not more than 0,2% has been detected.

I have worked in the 700 area for the last six months and have never found gas in high percentages."

In June 1970 an instantaneous methane outburst had occurred in the first left hand companion of section 700. This roadway had developed to within a metre of the dyke shown on Exhibit A. It had become necessary to install a steel girder set in the face one metre beyond the last set since roof conditions were poor in the burnt coal zone. Three sliping holes were drilled on either side of the face as shown in Figure 15 to square off the face in preparation for the girder set installation. The six shotholes were blasted simultaneously and immediately thereafter there was a further blast followed by a large dust cloud and methane gas which enveloped the miner and several of his crew who were standing in the second left hand companion at peg 1410 on Figure 14 Exhibit A. The section ventilation was temporarily reversed and the crew make an attempt to escape eastwards into the intake airway.

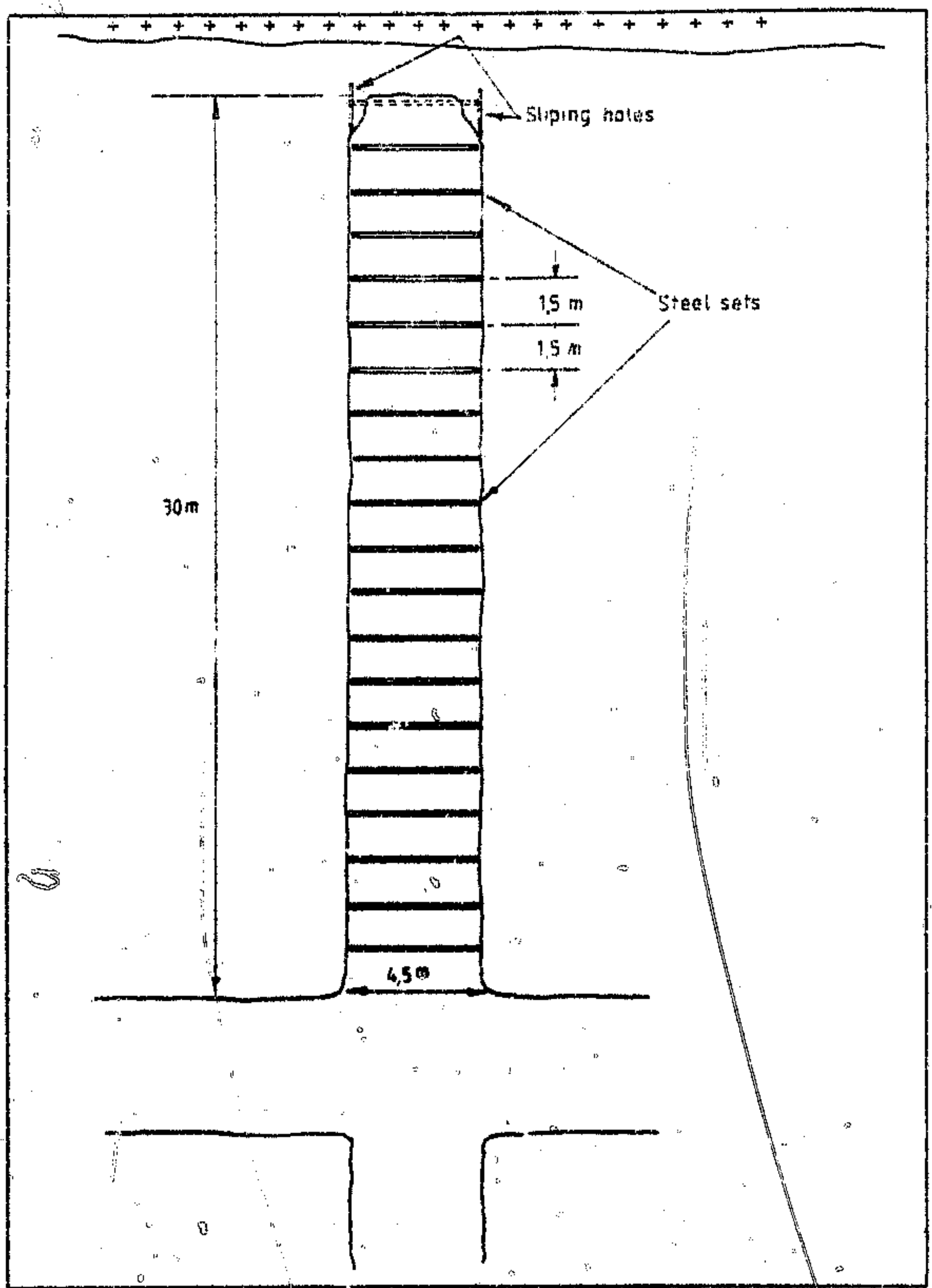


Figure 15

PLAN OF FACE SHOWING SLIPING HOLES
AND GIRDER SETS PRIOR TO OUTBURST

An examination of the first left hand companion some time later revealed that approximately 230 tons of burnt coal had been deposited in the roadway for a distance of 25 metres from the face, all the steel sets in this length of face were blown out by the force of the blast and the roadway roof had collapsed to a height of approximately 5 metres.

It was fortunate that, unlike the incident in the South Rand coalfield, the fine loose burnt coal did not spontaneously heat and all the debris was eventually loaded out and the roadway was resupported.

Shortly before this event a similar outburst had occurred in the first right hand companion of this section and yet no written records exist of these two incidents. The author has been fortunate in obtaining first hand evidence of these two methane outbursts from witnesses who were present in the section at the time. No precautionary measures were adopted until the third event occurred on the 25th February 1971; the total lack of systematic reporting and disseminating of technical details of such a serious nature to the mining fraternity is highlighted in this example.

The third incident which occurred at Indumeni Colliery is the only incident in South Africa where reliable measurements are available of the quantity of methane emitted by the outburst.

Figure 16 depicts the percentage methane in the upcast shaft prior to and after the outburst. Based on an upcast quantity of 189 m^3/second the area shown hatched on the graph indicates the total volume of methane desorbed from the face, roof and sides of the second left hand companion.

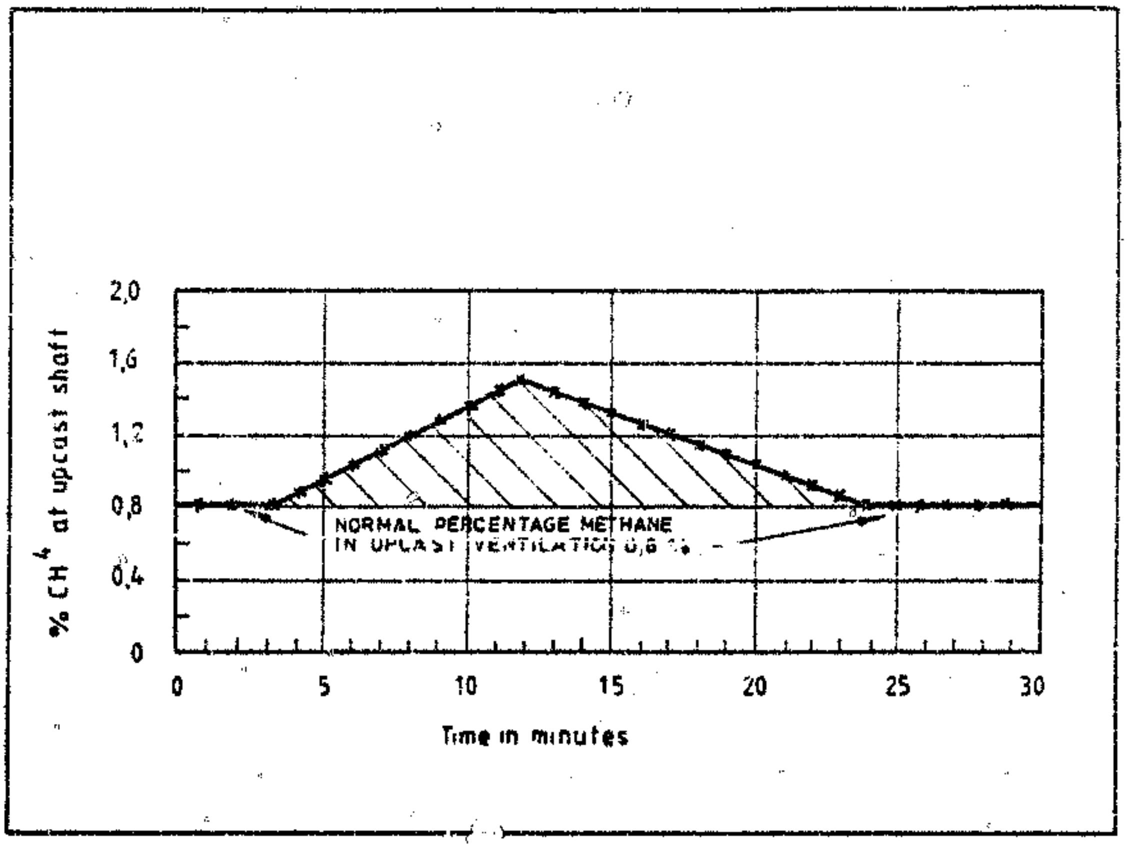


Figure 16

GRAPH SHOWING PERCENTAGE OF METHANE RECORDED ON AN AUTOMATIC METHANE RECORDER AT THE MAIN UPCAST SHAFT BEFORE AND AFTER AN OUTBURST

Quantity of air circulating in upcast shaft = 188,78 m^3/sec

Normal % CH_4 in upcast shaft = 0,8% CH_4

Normal quantity of CH_4 circulating in m^3 per second

$188,78 \times \frac{0,8}{100}$ = 1,5 m^3/sec

Average % of methane in upcast shaft from point A to point B on graph = 1,17%

Increase in % CH_4 over normal (1,17% - 0,8%) = 0,379%

Quantity of methane in m^3/sec as a result of outburst = 0,715 m^3/sec

Total quantity of methane emitted

$188,78\ m^3/sec \times 21 \times 60 \times \frac{0,397}{100}$

= 944 m^3 of CH_4

8.4.3 Conclusions which may be drawn from the practical evidence

The following conclusions may be drawn from the incidents discussed:

- **Instantaneous methane outbursts have occurred in South African collieries.**

- **They have all occurred in burnt coal zones near igneous intrusions.**

8.5 PRESSURE OF METHANE GASES IN OUTBURST PRONE AREAS

The pressure at which firedamp is contained in coal seams underground has been the subject of many experiments, most of which were largely unsuccessful due to leakage from boreholes, through breaks in the coal and its associated strata, and the low permeability of the coal. About 1880, pressures were measured in underground boreholes varying from 810 kPa to 3220 kPa. In later years very few cases of high pressure have been recorded although at Point of Ayr Colliery pressures as high as 2095 kPa have been registered. Further manifestations of high pressure are sometimes to be found in the occurrence of outbursts of gas and coal. Recently workers in Holland have recorded firedamp pressures of 101 kPa to 1572 kPa in boreholes. Graham developed a method of measuring the pressure of methane in coal from a small coal sample which gave pressures from 293 to 887 kPa.

In June 1986 a vertical prospecting borehole drilled in the North West area of the South Rand coalfield from surface to the coal seam at a depth of 170 metres intersected a pocket of methane gas under pressure. Just prior to reaching the sediment zone the drill was changed to a core drill. At the first intersection of the coal zone the driller believed he had struck a pocket or cavity as he could not apply down pressure to the rods. Realising that something must be wrong he commenced withdrawing the rods.

As each successive pull was commenced the pressure of gas in the hole pushed the drill rods up into the drill chuck. As the last three rods, each 6 metres in length, were being withdrawn the gas pressure pushed the rods out of the drill hole to a point come 3 metres from the rig. Following this final incident, water, gas and vapour rose to a height of 20 metres above the surface. An initial smell of hydrogen sulphide (H_2S) gas was noticed and the hole continued to emit methane gas for some two years. While the main core was lost at the intersection point, sediments recovered were indurated (baked by the proximity of dolerite sills and dykes). Unfortunately no gas analysis from this hole was ever undertaken.

Figure 17 shows a seam section from Borehole number 92 situated 300 metres to the east of Borehole number 2005. The coal measures are overlain by a dolerite sill some 80,9 metres thick and lying at a depth of between 68 and 148,9 metres from surface.

The upper 3,16 metres of the coal seam including the immediate roof is badly faulted and disturbed (slickensides). The volatile content varied between 14,2% to 20,8% indicating a degree of burnt coal. A 4,98 metre dolerite sill has split the lower portion of the seam into two distinct bands both of which are badly burnt (volatiles of 6,4%).

The technical details of the drilling gear when the core barrel struck the methane pocket were as follows:

Depth of hole	170 metres
Core barrel diameter	7,57 cms
Weight of rods suspended in hole	2125 kg

Assuming that the gas pressure was able to just support the weight of rods in the hole then the pressure would be as follows:

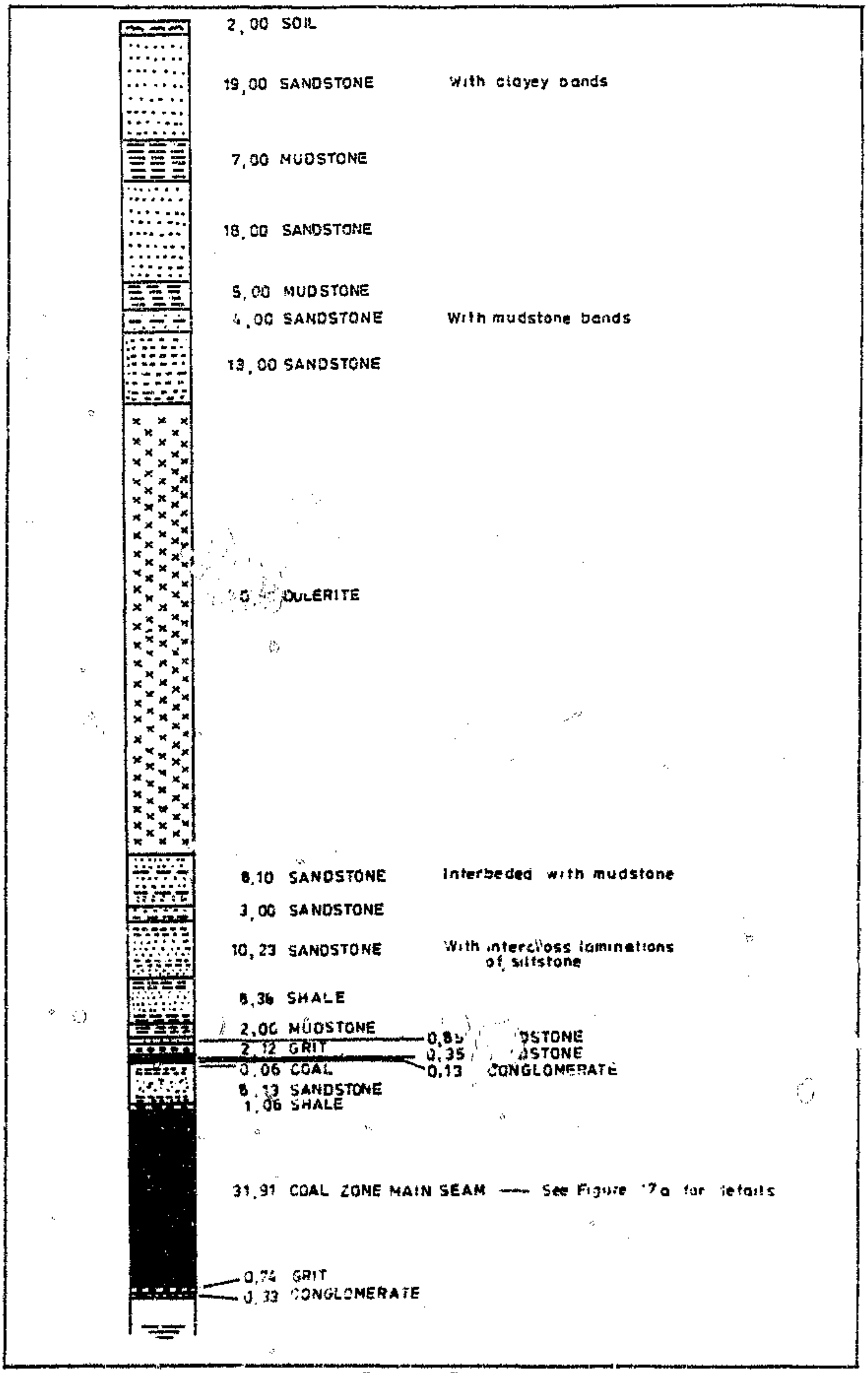


Figure 17

BOREHOLE SECTION 300 METRES FROM THE BOREHOLE WHERE RODS WERE BLOWN OUT AND [illegible] DRILL CHUCK

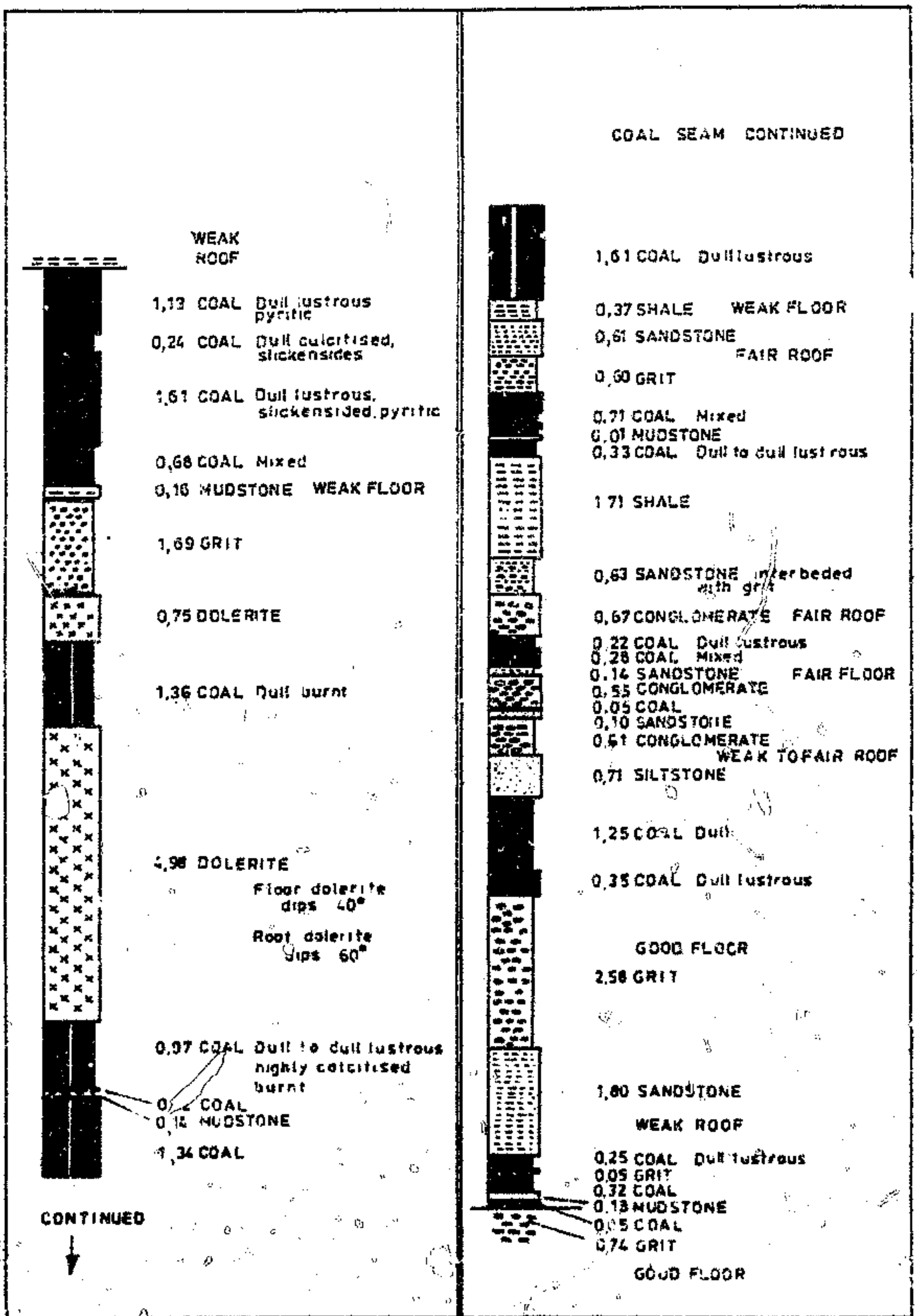


Figure 17a
COAL SEAM SECTION

$$\text{Pressure in Bars} = \frac{\text{Weight of rods in kg}}{\text{area of core barrel in square centimetres}}$$

$$= \frac{\left(\frac{170 \text{ metres} \times 75 \text{ kg}}{6}\right)}{\left((75,7)^2 \times 0,7854 \right)}$$

$$= \frac{2125 \text{ kg}}{45,00 \text{ square centimetres}}$$

= 47,22 kgs per square centimetre

= 48,153 Bar or 4877,9 kPa

8.6 METHODS TO BE ADOPTED IN ALLEVIATING GAS OUTBURST RISKS

Instantaneous gas and coal outbursts pose a problem to the mining of the deeper coal seams in South Africa. Hitherto the occurrences have been confined to coals of high rank, such as the anthracite seams and burnt coal zones which have a volatile content less than 14% and a Kex index of less than 75; the fine dust clouds raised as a result of such outbursts are thus unlikely to develop into a coal dust explosion should an ignition source be simultaneously experienced.

The vast quantities of methane which are emitted during such an outburst can create dangerous situations. Apart from its effect as a dilutant in the normal airflow in the mine where miners could be gassed and overcome by lack of oxygen, such outbursts can seriously reverse the airflow in the affected section and should an ignition source be present where the methane/air mixture is within the explosive ranges a devastating methane explosion could result. An added complication is the possibility of spontaneous combustion occurring within the burnt coal deposited, after the blast, in the affected roadway.

While the coal seam remains unworked the methane gas pressures remain in equilibrium with the virgin coal. However, once an excavation is driven in close proximity to the zones of high gas pressure, areas of high stress are developed around the excavation especially in zones of faults, slips, slickensides, dolerite dykes and sills and burnt coal and mainly these two factors (excessively high methane gas pressure and highly stressed strata) lead to the outbursts under discussion.

Of major importance to Mining Engineers. Geologists and Research and Development Engineers is the question of what can be done to firstly anticipate and secondly deal with the problem with a view to safely negotiating outburst prone zones.

Dr. Kevin Middleton has divided the measures to be adopted as either "active" or "passive". "Passive techniques are adopted to contain the effects of outbursting or predict the onset allowing remedial measures to be introduced. Active techniques are performed to prevent the problem occurring or to induce an outburst under controlled conditions." (Middleton. 1988).

The author would prefer to divide the measures to be taken into:

- Anticipative techniques.
- Mining or coal winning techniques.
- Ancillary techniques.

Anticipative techniques are those measures which ought to be adopted to alert miners that areas prone to outbursts are being approached. Mining or coal winning techniques are adopted in developing through the affected area/areas, and ancillary techniques are those precautions which need highlighting as and when the necessary mining techniques are adopted.

8.6.1 Anticipative Techniques

These may be divided into five major categories as follows:

(a) Prospecting drill holes from surface

These holes, normally drilled to intersect the coal seam measures, provide the Mining Engineer and Geologist with all the relevant information of the coal seam at that point. These include:

- Depth from surface - the deeper the seam the more gas likely to be encountered and the higher the pressure.

- Thickness of seam.

- Nature of strata and coal measures. The presence of dolerite dykes and sills in the overburden and in the seam itself, coupled with slips, fractures, faults, slickensides and burnt coal will indicate a high probability of stressed soft coal which, following the examples given, would indicate the possibility of encountering instantaneous outbursts of methane. Figure 17 shows the nature of coal measures adjacent to a borehole where a methane outburst occurred.

- The proximate and ultimate analysis of coal.

  The proximate analysis will furnish the moisture, ash, volatile content and fixed carbon in the seam in "per cent by weight". The higher the rank of the coal the more likely the seam will be to outbursts especially in South Africa (see Table 1).

Middleton (1988) states that as far as Cynheidre Colliery in Wales is concerned "the complex nature of the geological conditions around the colliery workings renders correlation with outburst incidence very tenuous". From evidence in South Africa it is possible to draw a "hazard" plan of the future underground workings showing not only 'roof' hazards but possible areas prone to outbursts - this plan would be used by the mine officials who become firstly aware of future poor roof conditions but also outburst zones. Dolerite dykes and sills, structurally disturbed coal which has little compressive strength and contains a considerable degree of microfissuring which accounts for its dull appearance and friable consistency should be marked on the plans. Figure 18 is an example of this recommendation.

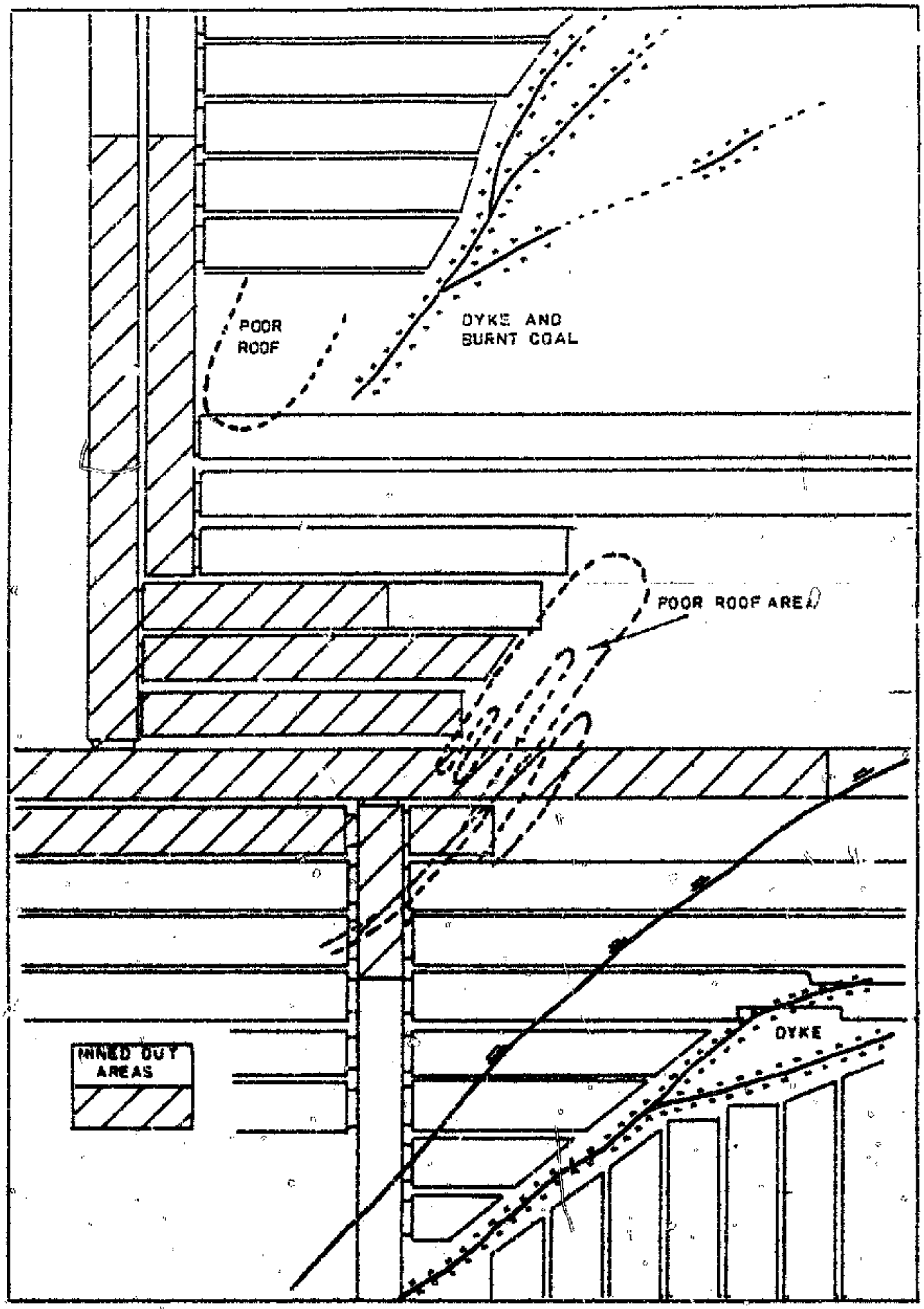


Figure 18
TYPICAL EXAMPLE OF HAZARD PLAN
SHOWING POOR ROOF AREAS AND AREAS
CONSIDERED TO BE OUTBURST PRONE

(b) Horizontal in-seam longhole drilling

Longhole in-seam horizontal drilling has reached a high state of perfection in South Africa. Holes up to 1000 metres in length are common. Until two years ago in the geologically disturbed South Rand coalfield longhole drilling had failed in several respects, the main failure being the inability of the machines to drill into and through massive burnt coal zones. Immediately the core barrel struck burnt coal the friable nature of these measures jammed the drill bit and the rods had to be withdrawn and the hole abandoned. The information obtained was therefore very scanty.

Recent developments have ensured that holes of 400 metres in length can be drilled and these holes serve two purposes:

- Gas pressures and rate of gas emissions can be measured providing valuable information on desorption rates in the coal seam.

- Geological disturbances are intersected and these are plotted on the future mining plans thereby providing valuable information on disturbed areas which may be prone to methane outbursts. They also provide data for the preparation of the hazard plan shown in Figure 18.

  Highly friable material and voids in the burnt coal (indicated by a drop of drilling pressure) would indicate high risk areas.

(c) Methane drainage

Two aspects are of note in this section. The drilling of 6 metre pilot holes in the face of roadways advancing in highly stressed areas and burnt coal will be dealt with in the section "Coal Winning Techniques". The second aspect concerns horizontal in-seam longhole drilling in advance of the headings or longwall face. The author deals with this technique in a previous chapter which includes methane drainage in gassy seams, such as those mined at Indumeni Colliery.

(d) Coal permeability

One of the main problems with regard to permeability is the relatively low permeability of outburst prone regions when compared with normal coal zones. This is probably due to high stress concentrations. The depth of the coal seam, tectonic and mining induced stresses are considered to be responsible for the reduced permeability. Thus gas which would normally migrate through micro-fissures as mining takes place is retained and bursts out when pressure differences are increased as a result of mining activity. Boreholes are therefore not likely to drain the methane to a degree which would minimise the risk of an outburst. Pilot holes in the advancing faces are covered in the next section.

(e) **Seismic Testing**

The exact definition of the protected zone is obtained from gas dynamic testing in the seam to be protected and the existing mine road system. Seismic transmission measurements are made and used to assess the danger of outbursts in certain parts of the coal seam. The essence of seismic testing is that isolines of the seismic propagation velocity can be obtained for the plane bounded by roadways. The seismic wave velocity is much faster in solid, hard rock than in broken and loose structures. The boundary of the relaxed zone can be obtained by comparing the change in values obtained from measurements taken prior to and following the mining of the protective seam.

Gas-dynamic measurements can be used to determine the

rearrangement of stresses in the seam to be protected. The degree of loosening (fracturing) can be established from the volume of gas drained, the gas pressure within the seam and the change in permeability. Alternatively, density measurements based on gamma-gamma radiation may be used to determine the degree of loosening.

The protective effect is extended to a much larger area than is normally indicated in published literature. The time factor must be taken into account because the protection is time dependent. Although compression of the loosened zone will recur, the outburst protection of the seam is unchanged. Experience has indicated that in low-stability coal seams that do not have igneous intrusions the danger of outbursts can be eliminated by destressing the seam by pre-working the neighbouring seams, particularly if it is accompanied by seam injection water. In the coking coal seams volley firing may be necessary to ensure pre-fracturing of the seams. Seismic methods have not been used in South Africa to date for this particular application.

8.6.2 Mining and Coal Winning Techniques

As coal mining approaches suspected areas of potential outburst activity, the mining method may be modified as follows:

(a) Advance drilling of the faces

Figure 19 shows the method used in massive burnt coal drivages in the South Rand and Natal coalfields whereby a series of 38 mm holes each 6 metres in length are drilled before cutting drilling and blasting the face. This precaution was taken after the gas outburst incident in 1969 in Section 7 in the South Rand coalfield and also at Indumeni Colliery.

The intention is to ascertain whether methane is contained under high pressure in pockets at some point ahead of the face; when blowers of methane are detected:

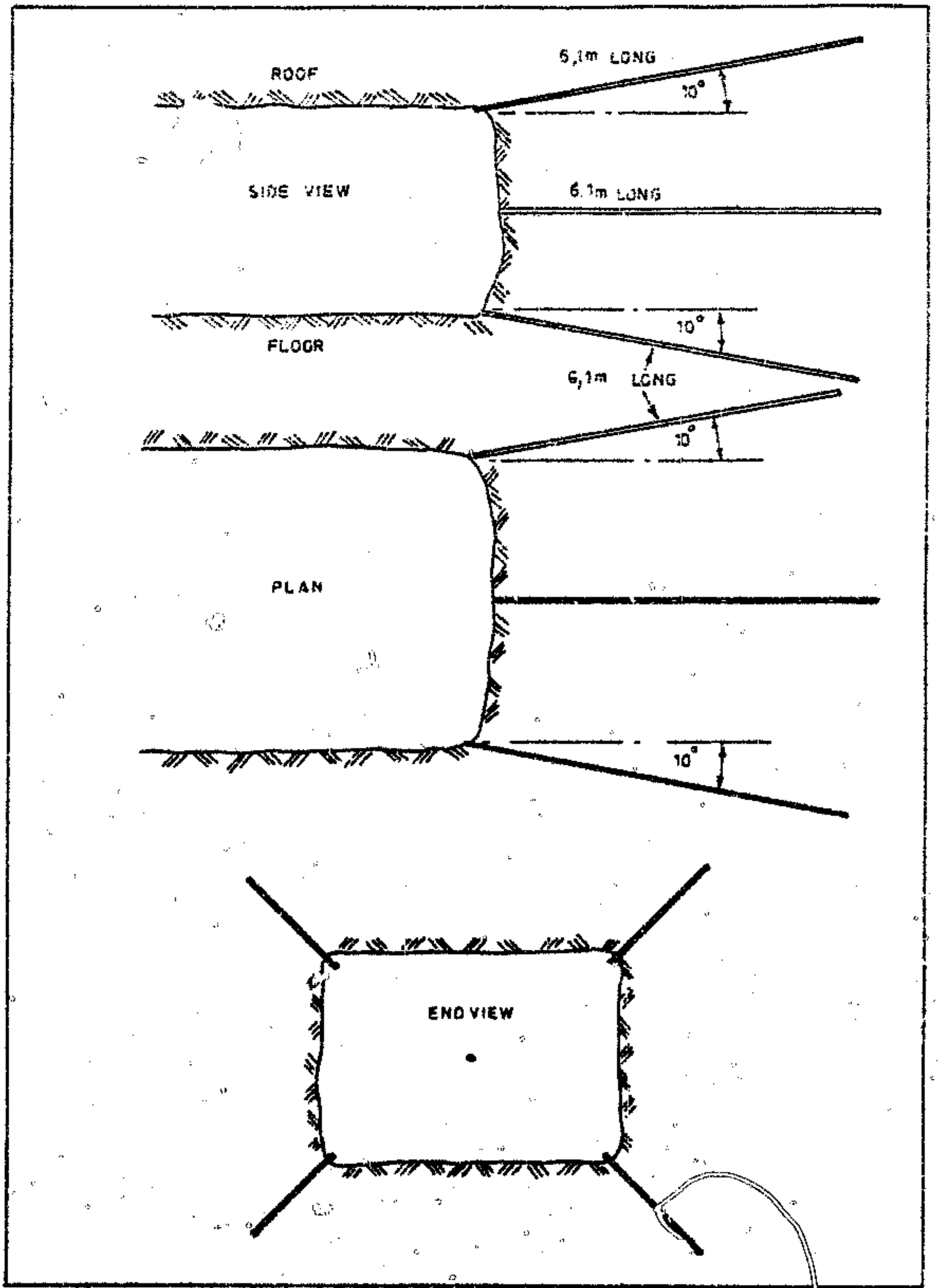


Figure 19

GAS DRAINAGE PILOT HOLES
(BURNT COAL ZONES)
DRILLED PRIOR TO EACH CUT

- all work is to cease in the place
- power in the section is to be switched off
- the Section Managers are to be notified and they are to develop a further plan of action.

It is recorded that several years ago in the South Rand coalfield, two men drilling such a hole intersected a pore/pores containing methane under pressure and were pushed back several metres when the gas blew the drill rods out of the hole. The necessary precautions were taken and the face was advanced by volley blasting. No outburst occurred.

(b) **Volley blasting or inducer blasting**

This is the most prevalent method used in dealing with the problem of methane outbursts and is practiced both in the South Rand and Indumeni coalfields. Williams (1960) records that in the outburst prone seams of Ponthenry in South Wales "volley blasting is the only method then of dealing with the problem of outbursts. The area of likely outburst is recognised by the lively nature of the coal and the occurrence of small outbursts of a few kilogrammes only. Holes are drilled in this zone, and provided this is done carefully (preferably by hand rotary drilling, certainly not by percussive drills) no outburst will occur. The holes are charged with explosive, the mine cleared of all personnel, and the shots fired simultaneously from the surface. At Ponthenry about ten holes, each charged with 0,5 kg of Polar Dynobel, were employed. The shock of the detonation induces the outburst, and after the ventilation has cleared firedamp from the mine, work can be resumed in safety. Up to 30 tons of coal have been blown out with one shot by this method at Ponthenry."

In South Africa either the affected zone is cut using a coalcutter, the height of the cut off the floor being dependent on stone intrusions; then drilled and blasted. Alternatively, the face is drilled using a "burn" cut and blasted. Due to poor roof conditions which are normally found in burnt coal zones holes are seldom drilled to a depth of more than 1,5 metres and explosive charges per hole are limited to approximately 600 grams per hole.

Explosively induced outbursts are common practice throughout the world. Ancillary precautions are necessary when this method is used and these are dealt with in the next section.

(c) **Speed of face advance**

The slower the rate of advance in outburst prone zones the more safe the operation will be since stress changes will occur more slowly and personnel will have sufficient time to react to changing circumstances. Since almost all incidents in South Africa occur in geologically disturbed ground conditions high rates of advance are not possible.

A reduction of rate of advance gives greater time for stress relaxation and gas release. He postulates a varying thickness of destressed strata depending on rate of advance of mining, on strength of both coal and rock and on the width of the heading and thickness of the coal seam. Hargraves (1983), in showing greater incidence of outbursts on the latter days of a five day working week demonstrates the influence of rate of advance. But, for instance, the recorded occurrence of outbu[illegible] on Mondays suggests that reduction of rate of advance is not an effective sole preventive in the Metropolitan Colliery Bulli seam application.

Hargraves (1983) further studied coal mined both with inducer shotfiring and with continuous mining. In general, the average size of explosively won coal (won daily from the one place) reduced as the days of the week advanced, indicating that the period of gas emission and no production over weekends diminished the contribution of gas in breaking the coal. Likewise, with single shifted continuous mining the coal was of smaller size in the afternoon than in the morning.

The limitation of daily rates of advance in longwalls and headings in outbursting mines in Europe is aimed at prevention, along with other controls imposed.

(d) Face Preparation

In most instances of outbursts in South Africa evidence is that during and after the event support in the form of girders are dislodged and serious falls of roof occur - often these falls lead to spontaneous combustion of the coal lying in the drive and this in turn leads to fires with the resultant danger of a methane/coal dust explosion. The sealing off of the area or roadway is also no guarantee that the air/methane mixture behind the seal will not enter the explosive range which may also lead to an explosion. The author has had personal experience of this at the Balgray Colliery explosion in 1973.

Further work on gas composition behind seals has shown that some of the traditional methods of evaluating the problem are unreliable.

Morris (1984) has stated: "The author discusses the work of some early researchers, namely Rhead and Wheeler (1910); Winmill and Graham (1913-16); Porter and Ralston (1914); Bone (1918); Partington (1919); Graham (1921); and Storrow and Graham (1924). It is felt that many of these ratios have been lost to the present mining engineer through time and the discussions of these ratios will give additional tools to those involved in the crisis of a mine fire.

Further, it will be seen in the test that Partington's suppositions are suspect. The author has expanded the theory to derive a fire ratio which may be used to determine whether the fire behind the seals has become extinct, and further when an explosion behind the seals is imminent."

It follows from a safety view point therefore that attempts should be made to support highly stressed roof in burnt coal drives with 2 metre long steel wire cables in conjunction with full column resin. Should it still be necessary to use steel arches or square sets then these may be installed. Figure 20 depicts the type of support envisaged in burnt coal to prevent falls of roof coal during an outburst. This practice is supported by Hargraves.

In order to prevent roof collapse ahead of steel sets in very friable burnt coal zones in the South Rand coalfield spiling using 19 mm diameter 3,3 metre long round bar is used ahead of steel square sets as shown in section and plan in Figure 21. This method successfully ensures no roof falls ahead of the last steel support. Prior to this method of support being introduced, roof falls 7 metres high have occurred with the fall extending beyond the face line thus making it almost impossible, apart from installing spiling rods, to "catch" the roof for the next set of steel support.

Six metre high cavities above the normal roof line add further problems the most serious of which is the accumulation of dangerous methane concentrations which:

- make it dangerous to blast the face

- difficult to remove by ventilation

- can cause anoxia to timber crews packing up the void above the normal roof line.

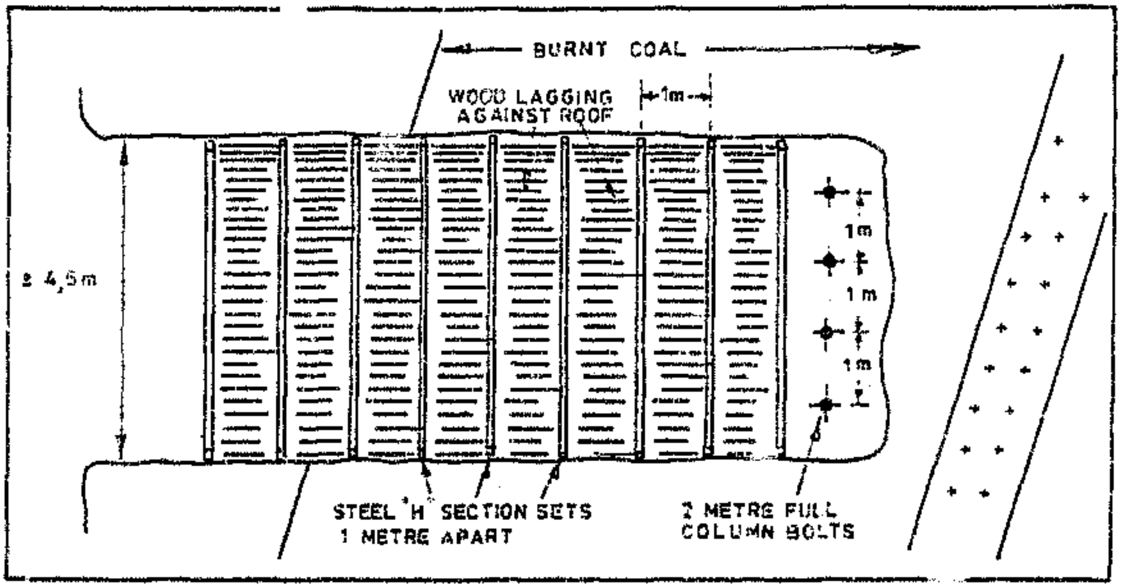


Figure 20

PLAN SHOWING SUPPORT OF ROADWAY IN MASSIVE BURNT COAL

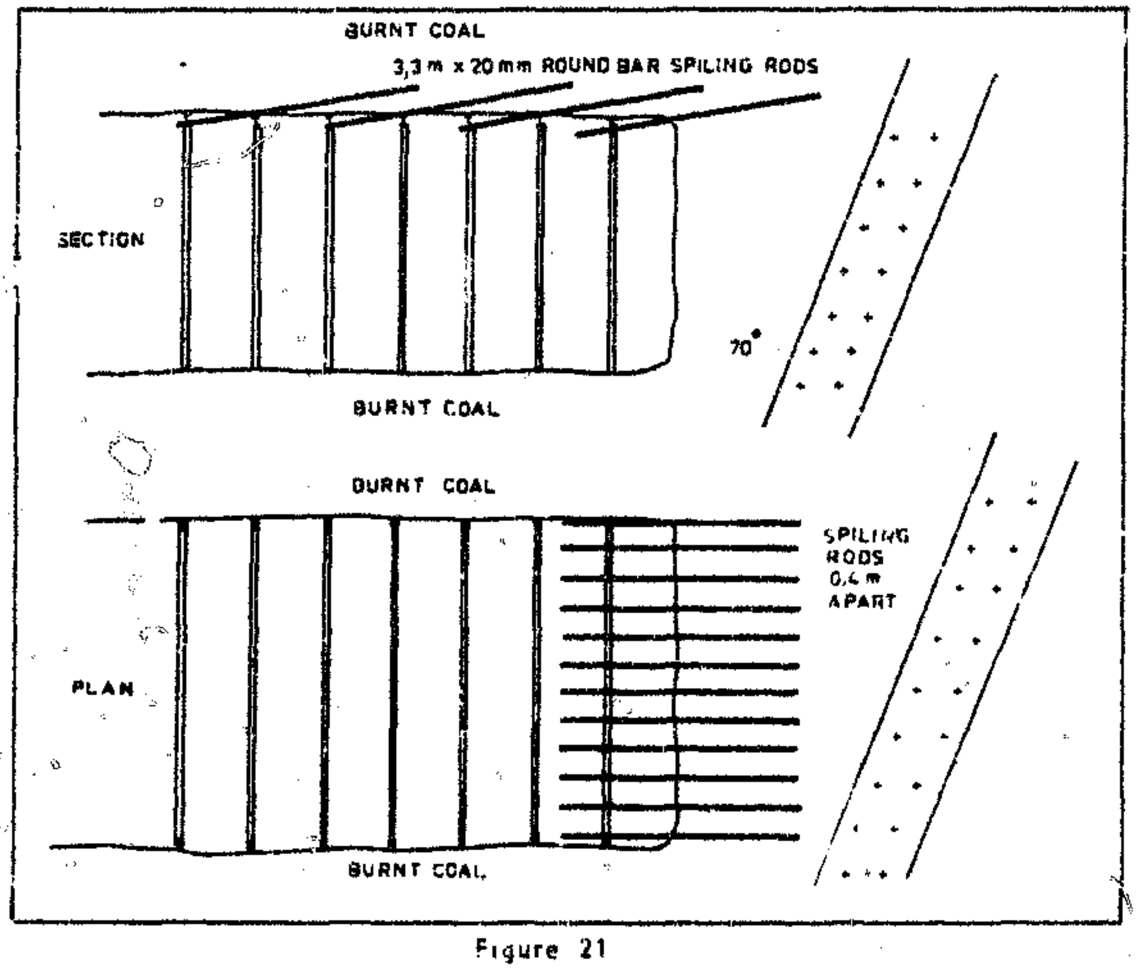


Figure 21

(e) Gas testing and methane alarms

Automatic firedamp detectors with audio-visual alarms should be used in all roadways being worked in massive burnt coal - this practice will ensure that operators will become aware of large methane emissions as and when they occur.

(f) Methane drainage tunnels

In areas known to have large quantities of methane or where disturbed and highly stressed ground conditions exist (see borehole section Figure 17) it may be advisable to follow the French pattern of driving two parallel methane drainage 'galleries'. There are several advantages in employing this method:

- A large quantity of ventilation can be directed into only two roadways.
- Supervision is generally better than that found in a multi-road section.
- All standard safety precautions can be applied, for example, automatic firedamp detectors and improved roof support.
- The section is treated as a development end and production is not of prime importance.
- A large block of coal can be drained of methane.
- Scientific tests are easier to apply and control since the unit is considered a "methane drainage gallery".

- Disciplined working procedures can be chosen to minimise danger.

(g) Remote control of coal mining machinery

Remote control of face loading and transporting machinery (continuous miners and shuttlecars) has been recommended primarily to ensure that in the event of an outburst occurring operators are not exposed to the blast of coal and gas. This practice has not as yet found favour in South African collieries mainly since the methods used (umbilical cord or radio controlled) are not well known and secondly because of the relatively small number of incidents recorded. Since inducer shot firing is practiced it has also been concluded (incorrectly) that after shotfiring, if an outburst has not occurred it will not occur until after the next volley blast. (Two of the recorded instances at Springfield Colliery have occurred several hours after the face was blasted).

(h) Life support systems

The author has discussed the one incident in the South Rand coalfield when, after an outburst, the miner and one of his workers suffered from anoxia and had to be rescued by the shiftboss. The Mines and Works Regulations now make it mandatory for all underground coal mine workers to wear a 30 minute self rescuer; furthermore, mines practice stringent training sessions teaching workers how to use the self rescuer and simulated incidents have now been introduced on most collieries to test the effectiveness of the training programmes in the use of self rescuers.

At West Cliff Colliery in New South Wales which in its early stages of development suffered from regular outbursts, all continuous miners and shuttlecars were equipped with compressed air cylinders and face masks in the cabs. The air supply was sufficient for 4 hours and on several occasions, was almost exhausted before the machine and its driver were recovered.

(i) Avoiding highly stressed areas

Highly stressed areas which are pulverised by tectonic movements are prone to methane outbursts in South Africa. It is impossible to totally avoid working these areas but the policy should be to develop, through the disturbance, only those roadways which are necessary to establish sound ventilation systems and access for men, machines and conveyors.

(j) Coal structural conditions

A problem arose in an opencast operation where excessive water and soft impervious clay overburden was collapsing in the highwall and endangering the dragline. Consultants specialising in soil sciences were called in and a 12 point hazard plan was developed. These individual hazard indices each ranging from 1-10 in degree of hazard were built into a composite index which alerted dragline operators of impending danger. For example, edge distances of the tub off the highwall had to be increased for safety reasons.

Brown, Rigby and Barker-Read (1984) have put forward a similar hazard plan to warn miners. The author quotes from a paper entitled 'Gas emission and outburst prediction.'

"A simple index has been devised to facilitate a rapid quantitative assessment of the structural condition of coal samples from the outburst-prone seams of West Wales. The 'Average Structural Index' (A.S.I.) is based on a points allocation system to macroscopic structural features. Initially, a determination of the overall character of the sample is made on the basis of whether the sample is normal, abnormal or outburst-type coal. Further examination detects the presence or absence of the commonly encountered slickensided planes or feather-type fracture markings. Flexibility in points rating is possible in cases of intense slickensiding or widespread occurrence of feather-type fractures. In such cases an additional point is allotted to mark the predominance of these features. Typical values of the A.S.I. are noted below.

Normal coal	1
Normal coal + slickensides	2
Normal coal + feather fractures	3
Normal coal + slickensides + feather fractures	4
Abnormal coal	5
Abnormal coal + slickensides	6
Abnormal coal + feather fractures	7
Abnormal coal + slickensides + feather fractures	8
Outburst-type coal	9

The points given to each sample from a particular site are totalled and the A.S.I. for a suite of samples is calculated by averaging the individual totals. The A.S.I. has the advantage that it may be performed underground at the face or on the surface but has the disadvantage that hand-size samples are required."

It is recommended that a similar Combined Hazard Index be developed for use in South Africa but including alongside abnormal coal the following - abnormal/burnt coal. Many collieries already emp' y roof hazard plans which indicate areas where poor roof conditions could be encountered.

(k) Mining of neighbouring seams prone to outbursts

Hitherto this particular circumstance has only arisen once in South Africa but the system is termed protective seam mining. Figure 22 illustrates the method. Neighbouring seams above or below the seam prone to outbursts are worked first and boreholes are drilled from the top seam, for example, to the outburst-prone seam to drain off methane. This system worked well in the Necsak coalfield in Hungary (Banhegyi and Rado, 1984).

In fairly shallow coalfields this method could be used by drilling holes from surface to the outburst prone coal seam.

Figure 23 depicts the method of drilling adopted at a colliery in Natal which was prone to methane outbursts. Development was carried out in the lower seam and 65 mm diameter boreholes were drilled into the top seam. Methane was bled from these holes and pressures of 698 kPa (100 lbs/square inch) were measured at the collar of the hole in the roof. This high pressure is indicative of the permeability of the normal coal measures at this colliery (Watson and Ogilvie, 1970 and 1971).

8.6.3 Ancilliary Techniques

As well as predicting outburst zones and adapting the mining method, additional precautions must be adopted. These are:

(a) Stonedusting

The dangerous conditions which arise as a result of a methane outburst and resulting fine coal dust make it necessary to stonedust the roof, floor and sides of drives which are advancing in burnt coal right into the face before blasting. The application of stone dust to the section itself should be of the highest standard.

An example of the procedures that are laid down at a Transvaal Colliery with regard to stonedusting in an outburst prone zone is set out below:

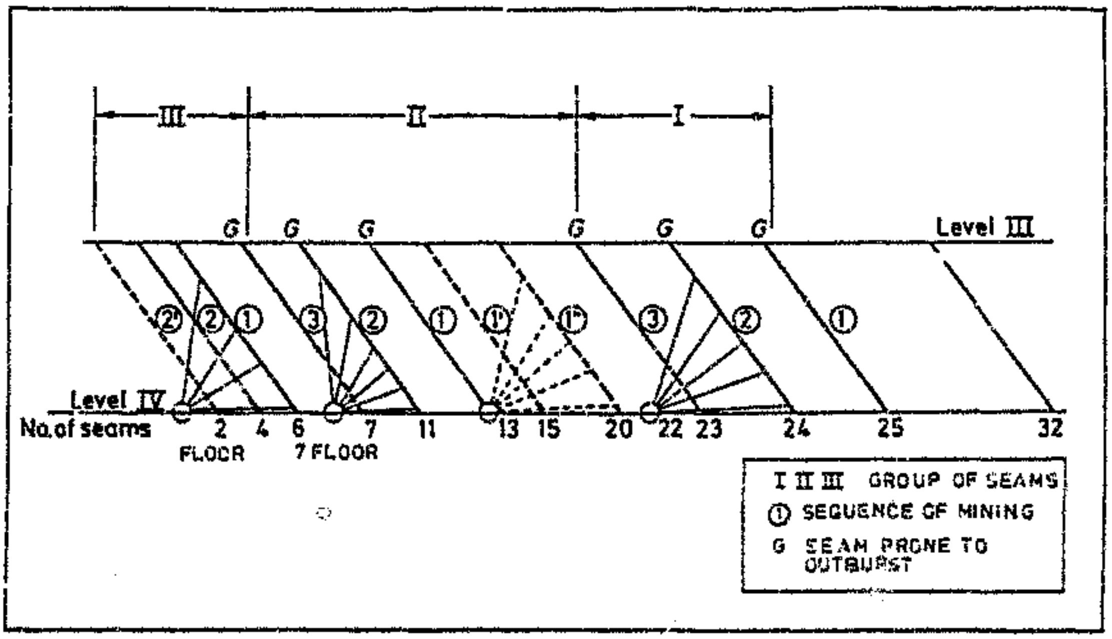


Figure 22

SEQUENCE OF MINING SEAMS AT ISTVÁN SHAFT

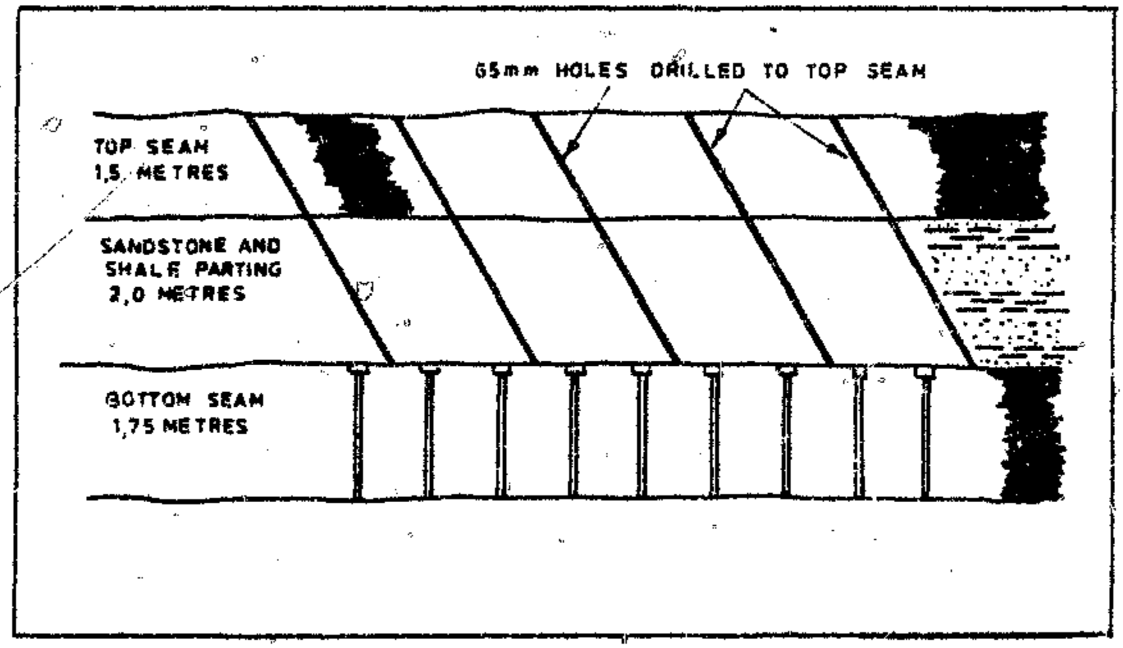


Figure 23

METHANE DRAINAGE HOLES DRILLED FROM BOTTOM SEAM WORKINGS TO TOP SEAM - INDUMENI

- The face and immediate roof and sides must be thoroughly stonedusted immediately before steel supports are set. Stonedusting must then be done on the supports and packing timbers.

- Before cutting commences the face must again be thoroughly stonedusted.

- At the end of each shift the miner must arrange to have each face re-stonedusted.

(b) **Flameproofing of equipment**

It is advisable to have the flameproofing of the electric equipment in sections which are advancing in outburst prone zones checked more frequently than would be done in normal sections and the results of such examinations recorded in a logbook kept at the shaft offices. Such entries should be countersigned by a responsible Engineer once a week. An example of such an examination report is given below:

"An inspection of the general flameproofing standards for electrical equipment was undertaken in Section G480 on Friday, 10 March 1989, in association with Messrs. Hodge and Ellis.

I advise that the inspection included the section electrical distribution, electrically driven mobile equipment as well as a flameproof fan and coal drill. With the exception of two stripped bolts in a coalcutter contactor enclosure, which was still found to be flameproof, no sub standard conditions were found.

Signed

Johannesburg"

(c) Power supply

The first task to be carried out by persons after an outburst occurs is to switch off the electric power supply to the section. This action will minimise the danger of faulty electric gear causing a methane explosion should the methane/air content pass into the explosive range.

(d) Face ventilation

It is advisable to have an air quantity of not less than 50 m^3/second entering the section, in order to dilute the methane produced from an outburst to safe limits as soon as possible. This should ensure that persons are not affected by anoxia and will reduce the dangers of methane ignition.

- The force fans used should be either 37 kW or 22 kW and should not ventilate more than 2 faces. If an 11 kW fan is available then only one face may be ventilated with this fan.

- The end of the force ventilation ducting should not be more than 5 metres from the face at all times.

- The quantity of air delivered at the face should not be less than 0.14 m^3/sec per square metre of face area. The quantity and leakage shall be checked at least weekly by the Environmental Officer and the results recorded in a book called the Burnt Coal Record Book which shall be kept in the environmental department.

- Ducting shall be kept in good condition. The miner should regularly check all joints and method of suspending the ducting. Any damage shall be repaired immediately or the ducting replaced if not readily reparable.

- Spare ducting should be kept in the section store. Methane alarm detectors shall be made available to machine operators such that one will always be in operation in each face being worked.

- The miner should make hourly checks with his methanometer and all officials should check the face whenever they visit the area.

(e) Special precautions review: monthly

It is advisable to develop a report sheet which will incorporate, inter alia, a remarks column for special precautions which the persons who work in the section should take.

An example of such a report is given below:

- The first defence against a dangerous build up of methane is adequate ventilation and a continual awareness by all supervisors and workers in the area of possible problems in this regard.

- Mining standards in problem areas must always be excellent.

- Visits to such problem areas by senior officials should serve to enforce the high standards required at all times for safe operations.

- An entry should be made in the shiftbosses logbook to state that the section is mining in burnt coal and that all precautions as laid down in Mine Standards are being taken.

Monthly review

This form will be signed by the responsible persons monthly and submitted together with the Burnt Coal Record Book to the Mine Manager for his signature.

Section: ____________ Shaft: ______________ Date: ___________

Special Precautions:

____

____

____

Signatures:

Miner: _____________________ Fitter: ______________________

Shiftboss: _________________ Electrician: _________________

Mine Overseer: _____________ General Engineering

Section Manager: ___________ Supervisor: __________________

Production Manager: ________ Section Engineer: ____________

Chief Engineer: _____________

Environmental Control Officer: ____________________

Mine Manager: ____________________________________

Date: ____

This type of report can be adapted to each particular application.

(f) **Hydraulic or foam stimulation**

In the relatively low permeable burnt coal zones in the South Rand and Natal coalfields, methane drainage may be substantially increased by using hydraulic or foam stimulation in horizontal drill holes drilled into the burnt coal zone. Deul and Kim (1978) state that hydraulic or foam stimulation has been used to increase the gas production from 23 of the 39 wells. Pre-stimulation gas production rates have ranged from 500 to 10 000 cfd. After stimulation, production rates of over 100 Mcfd have been obtained from boreholes in the Pittsburgh and Mary Lee coalbeds. In boreholes stimulated under Bureau of Mines contracts and subsequently mined through, no damage to roof or floor was observed. In all cases, stimulation served to widen and extend the fractures naturally occurring within the coal, and stimulation fractures were contained largely within the coalbed.

Hydraulic stimulation has been practiced in a deep Transvaal Colliery (New Denmark; 200 metres) in competent sandstones overlying the seam in order to encourage early goafing of the roof once pillar extraction had commenced. The operation was successful and roof caving commenced after 60 metres of pillars had been removed. Without the use of hydraulic stimulation of the immediate roof, the physical model (Australian Coal Industry Research Laboratories (A.C.I.R.L.) predicted a retreat length of 104 metres before goaf initiation. Mathematical modelling predicted lengths of 95 to 140 metres.

The burnt coal zones of the South Rand and Indumeni coalfields tend to become friable when mined by single advancing roadways. These high stresses caused by mining necessitate the installation of steel arch support at intervals of [illegible] metre so as to prevent the collapse of the exposed roof - sometimes to a height of 6 metres. In 1984 an attempt was made to cementate the burnt coal zone using a 50 mm diameter borehole drilled to a length of 10 metres into the face of a roadway. A standpipe was grouted into the hole and a cement grout mixture was pumped into the hole at a pressure of 20 Bar. Immediately grout broke out of broken fracture planes in the face. In fact pressure dropped off to almost nil indicating that the grout was not causing fracturing deeper inside the hole but simply escaping into the face of the roadway. Further advance of the roadway indicated this to be the case. Hence hydraulic stimulation would have to take place while roadways were still being driven in coal and before they reached the burnt coal zone.

(g) Seam gas content to volatile matter

Lunarzewski and Battino (1984) in a paper entitled "Application of gas predrainage for prevention of CO_2 gas outburst" state:

"Recognised methods of gas-emission prediction for collieries that experience or envisage gas problems rely, among other factors, on curves that relate seam gas content to the volatile matter of the coal. Unfortunately, such relationships have not, to date, been developed for Australian coal seams, although the need for these curves is unquestioned. To acquire a basic understanding of the variation of seam gas content with volatile matter, and to assess the gas hazard level for Australian coal mining conditions, a preliminary curve was prepared by Lunarzewsk and Battino based on gas contents calculated by the direct method of measurement as developed by the U.S.B.M. and with the use of both fresh face coal samples and geological bore cores. It should be pointed out that for Metropolitan colliery conditions, the gas content of the working Bulli Seam is approximately 10 m^3/ton with a volatile matter of 18 to 20% (ash-free dry state).

The work carried out to date at Metropolitan Colliery certainly gives encouragement that CO_2 gas outbursts can be controlled or prevented by the application of gas pre-drainage in the working seam by the use of suction.

The investigations conducted to date were concentrated in one panel of only one colliery in Australia. To confirm the results that have been attained further experimentation should be pursued in similarly affected coal mines.

One of the problems facing the Mining Engineer in South Africa is the low permeability of the high rank low volatile burnt coal zones where to date all recorded outbursts have occurred; unless hydraulic or foam stimulation can be successfully applied to increase fracture planes and so improve the rate of methane desorption it is difficult to pre-drain methane from these outburst prone zones.

8.7 CONCLUSIONS

The studies in South Africa have shown that for the outburst phenomenon to occur one or more of the following conditions must be fulfilled.

1. The quantity of gas present in the coal seams must exceed a certain limit; this limit has not been defined in South African collieries yet but Lama (1980) mentions a figure of 10 m^3/ton of coal in the case of methane for Australian mines.

2. Comparatively low strength of coal compared to the stresses acting in the seams.

3. Depths are greater than 150 metres from surface.

4. Thick (20 metre - 70 metre) dolerite sills overlie the seams.

5. The coal has been burnt by dolerite intrusions in the forms of dykes and sills. It is of high rank and has been tectonicaly disturbed. Volatile content does not exceed 14% and can be as low as 8%.

   It usually has shear zones, slips and slickensides and horizontal disturbance of the visible layers is evident. Unless promptly supported, roof collapses will in all cases occur. The coal is dry.

6. It usually has low permeability and high adsorption capacity where outbursts occur.

7. Spontaneous combustion is a threat in all cases after an outburst.

8. There must be some initiating factors present such as vibration from machine, blast initiation, withdrawal or setting of supports, drilling of holes, etc.

The period of the occurrence of an outburst can vary from a fraction of a second to several minutes and outbursts can occur almost simultaneously with the initiating factor or up to 18 hours thereafter.

The quantity of material thrown may very from a few tons to hundreds of tons with high quantities of methane released into the mine ventilation system.

This creates extremely dangerous mining conditions and has elsewhere in the world led to loss of life and equipment and to gas explosions followed by dust explosions.

Two basic theories have been put forward to explain the phenomenon of outbursts. The most commonly accepted is the dynamic theory (Skocinski, 1954, Khodot, 1961 and 1964). According to this theory both rock pressure and gas play an equal role. Rock pressure causes fracturing and release of a large volume of gas which is instrumental in the displacement of coal. A number of research workers in Russia, France, Belgium and Germany support this concept.

The second theory was put forward by Ruff (1930) and is known as the "nest" theory. According to this theory, nests or pockets of low strength (already crushed) coal are saturated with gas. If one of the localised pockets is suddenly punctured with the advance of the face, a large volume of almost adhesionless coal particles floating in gas is displaced, with a dynamic effect.

A further theory put forward is that when the volatiles and water in the coal are driven off during the process of the igneous intrusion into the coal measures, methane is adsorbed onto the surface of the coal thus increasing the methane content in the burnt coal zone.

Methods which can predict and alleviate outburst risks are divided into those which anticipate the outburst, measures used during the mining of affected areas and ancillary techniques.

Of the anticipative methods, vertical and horizontal prospecting holes and geological assessment techniques hold most promise for anticipating outburst prone zones.

In coal winning techniques advance drilling of the face prior to blasting, and finally volley blasting, are the most prevalent measures adopted to safely induce outbursts.

General good mining standards will greatly assist in reducing the dangers of outbursts when they do occur.

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CHAPTER 9

CONCLUSION

Simplistically, the build up of methane to explosive limits in underground workings as a result of one or more factors coupled simultaneously with an igniting source of the correct temperature and duration, will lead to a methane explosion. The presence of fine coal dust in mine roadways poses the added danger of a coal dust explosion. The synergy derived from human errors, resulting from carelessness, lack of training and experience, taking chances, short circuiting good mining standards coupled with difficult mining conditions, and economic circumstances, for example, relatively low selling prices received for coal thereby leading to severe cost reductions, has resulted in devastating explosions.

Ventilation plays a major role in preventing the formation of explosive methane/air mixtures and the use of automatic firedamp detectors and velometers to monitor environmental standards in the mine is important. Supervision and training of staff is often neglected but means of improving this aspect of the operations by the introduction of formal courses are well documented. Unless supervisors set an example and test for methane regularly, explosions will continue to occur. More planning and thought needs to be given to the sealing off of old worked-out areas.

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

Particular attention needs to be focused on the draining of methane from the goaf cavity, and preventing recirculation of the ventilation at auxiliary fans.

Mining Engineers are often lulled into a false sense of safety by having their operation described as "well ventilated mines" and "clean and well managed operations". Unless they attend to the details of good and efficient face ventilation, methane and coal dust explosions are likely to occur. The stoppage of one or more main ventilation fans for any length of time has, without exception, led to dangerous accumulations of methane.

Correct supervision and training of Electricians in the proper methods of installing and maintaining electric systems are a pre-requisite to preventing electric sparking. The choice of equipment plays a major role in ensuring that a safe environment is established underground.

It is incumbent on Mining Engineers to increase supervision when major work is necessary in loading out coal from old workings (falls of roof and spalling of ribsides) and repairing 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. Adequate stone dusting does not receive enough attention, particularly in old areas.

Poor training, insufficient experience, and a lack of attention to detail and discipline, leads Miners to drill, charge and blast working faces incorrectly with little attention to safety. These omissions provide the recipe for a disaster and, when carried out in the present of methane, lead to the unsafe practices.

Improved training of Mining Engineers in the use of explosives underground, will increase their awareness of the dangers attached to blasting.

In many instances, the prompt and efficient attention to fires has resulted in the minimum disruption to operations and the speedy restoration of the environment to a safe condition.

If carelessness and folly are some of the hallmarks identified in this investigation, they are more than matched by the great courage displayed in every incident by the Miners who comprised the rescue teams following explosions.

Finally, it is worth remembering: "Boast not thyself of tomorrow; for thou knowest not what a day may bring forth".

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