OUTBURSTS AT COLLINSVILLE – A CASE STUDY

By

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ABSTRACT

Outbursts have occurred irregularly in the Collinsville Coal field since 1954 when an incident claimed seven lives. Further significant outbursts occurred in 1960, 1961, 1972 and more recently in the No. 2 mine in 1978. From the information collected over 25 years a comprehensive program to minimize the hazard of outbursts has evolved. Further developments of techniques to combat outbursts are being developed.

INTRODUCTION

The Collinsville coal field was first developed in the year 1919. All underground mining has been in the Bowen Seam except for small areas worked in the Garrick and Blake Seams. Workings in the Bowen seam cover a total area of 650ha.

The seam was first worked from the No. 1 and 2 State Mines and Bowen Consolidated No. 1 Mine. All are now closed. Collinsville Coal Company's (C.C.P.) No. 2 and 3 Mines still operate in the seam.

The Collinsville coal measures include 11 coal seams dipping south at about 8 degrees. They are bounded on the east by the Collinsville Fault (throw 230M) and in the west by a thinning of the measures. The measures overlie the Lizzie Creek volcanics. The coal reserves are traversed by a number of major reverse faults and igneous instrusions in the form of felspar porphyry sills.

In 1954, a coal and gas outburst occurred at the No. 1 State mine which resulted in the loss of seven (7) lives. It is unknown as to whether the outburst occurred after the firing of a small shot or other wise. It is estimated that 14 000 m^3 of gas was emitted and 500 tonnes of coal ejected in the outburst (C.C.P. Reports, unpublished).

Following this incident the practice of inducer shot-firing was commenced and a number of outbursts were induced.

Outbursts were also experienced in the deeper sections of C.C.P. No. 3 Mine (formally Dacon No. 3 mine) and three incidents were reported. The mine is now retreating and has less than three years production life.

The No. 2 mine is expected to provide initial access to extensive reserves down dip and is the centre of a research program into mining the deeper reserves.

PART 1

REVIEW OF OUTBURSTS TO 1972

THE 1954 DISASTER

Coal mining had progressed in outburst free conditions for many years. In the evening of October 13, 1954, an outburst occurred in the deepest part of the Collinsville State Mine about one and a half kilometers from the entrance of the No. 1 tunnel. Seven men were killed when 900 tonnes of coal and stone was dislodged and blown 30 metres up the heading.

On the day shift prior to the outburst a disturbance in the floor was encountered (from evidence before the Royal Commission 1956). This consisted of broken soft coal and faulted coal showing "black lead" or slicken-siding. This was recognised by the deputy to be a sign of a fault which was evidenced elsewhere in the mine. Expecting to find an increase in the evolution of carbon dioxide, the deputy was surprised to find no carbon dioxide, not

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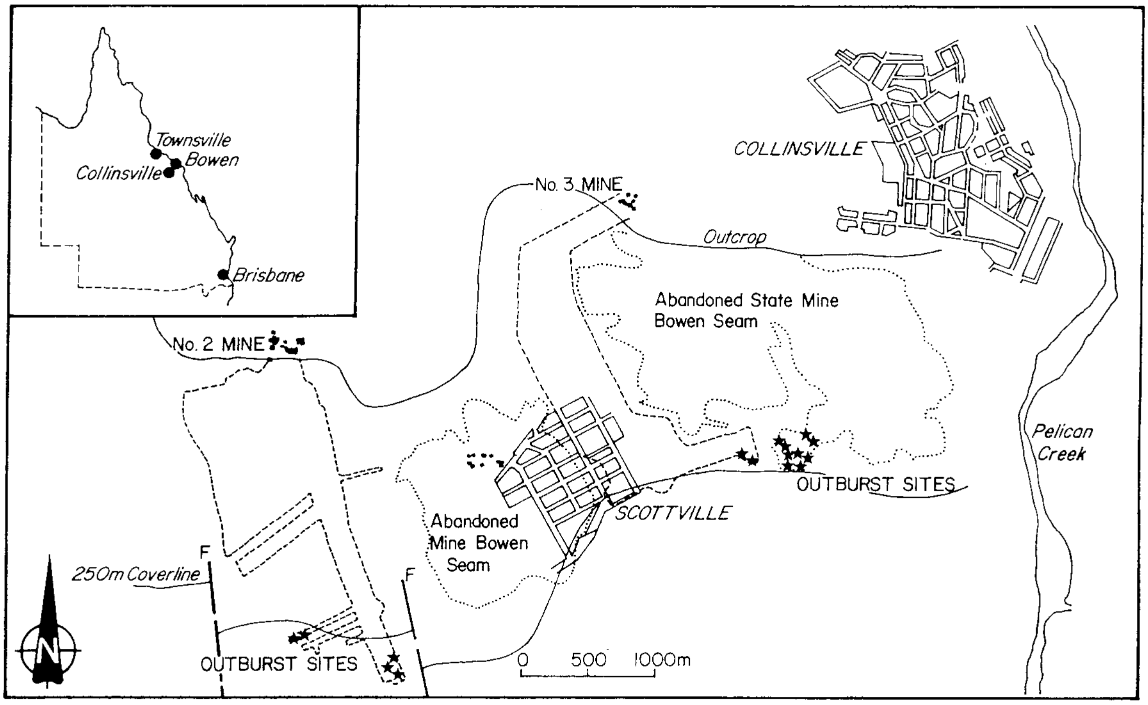

Fig. 1 — Location Map, Outbursts at Collinsville

even at the boreholes which were being drilled prior to blasting the face.

The outburst occurred at 5-50pm. The three men who were presumed to be working at the face must have had some warning as their bodies were found well away from the face and uninjured by the flying coal which was blown 30 metres up the heading.

The death of all seven men was caused by asphyxiation in an atmosphere of carbon dioxide (Q.G.M.J. 1954). The gas released backed up against the ventilation to A9 level and went to level B6 on the return side. See figure 2. A dirty grey haze was reported immediately after the outburst. This haze was thought to be caused by condensation as the warm mine air was cooled by the expanding carbon dioxide gas.

The outburst which expelled carbon dioxide gas was associated with a faulted area (Hargraves, 1963). An interim report by the chief inspector commented, "These same faults had been bared and penetrated for a distance of three quarters of a mile (1.2km) along the fault prior to this unexpected occurrence."

A reverse fault with a throw of nine metres up ahead of the working face intersected a steeply dipping normal fault. A dyke also occurred in this area. See figure 3.

A Royal Commission headed by Mr Justice Sheedy was appointed to examine the practices and circumstances leading to the disaster. It found the occurrence of the outburst was without precedent in Queensland and had occurred only twice in Australia; in 1897 and 1925 at Metropolitan Colliery, N.S.W. They concluded that the condition causing the eruption was unique and that no one could have readily anticipated the outburst.

The investigation revealed that carbon dioxide was not uncommon in the workings and was often seen bubbling out of the floor on the dip face. Evidence was given that "During the fortnight or three weeks prior to October 13, the date of the outburst, the bubbling (of carbon, dioxide from the floor) had increased".

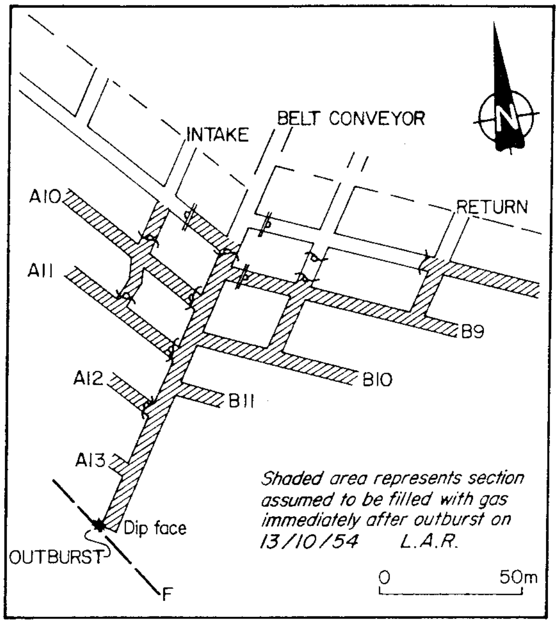

Fig. 2 — No. 1 State Mine Dip Workings

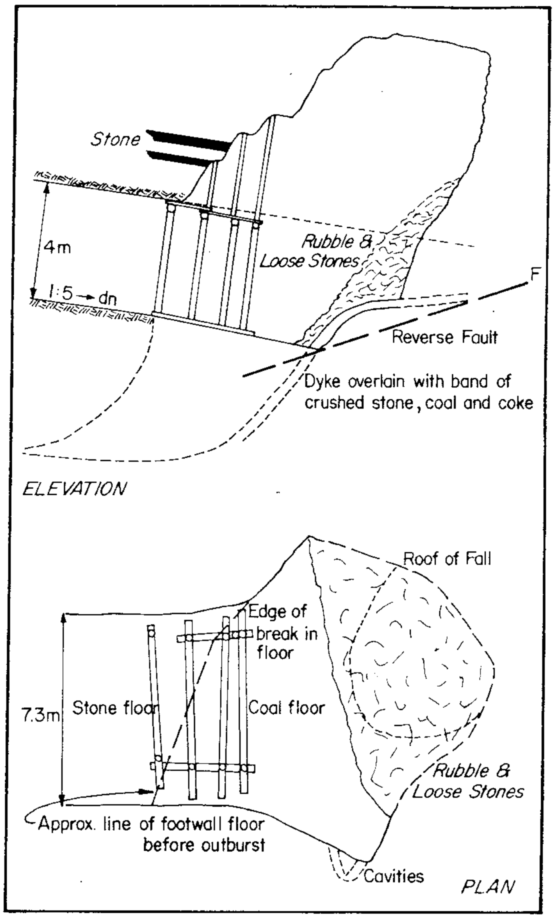

Fig. 3 — Outburst cavity — 13.10.54
(after Rogers, 1955 internal report)

OUTBURSTS IN THE STATE MINE: 1960-1961

Mining at depth beyond the location of the outburst of 1954 recommenced in 1960. The conclusion of studies into mining areas prone to outbursts was stated by Hardie (1960), "The only widely adopted and generally applicable safety measure against instaneous outbursts is the practice of inducer shotfiring".

Accordingly approval was sought to introduce this type of mining in hazardous areas. A full face was fired, either with delay or instantaneous firing without the use of a cut in the face. The shock of the blast induced outbursts while the men were at a safe distance.

Routine inducer shotfiring practice was approved by the Department of Mines subject to conditions summarized as the following:

1. No individual charge was to exceed 28oz (800 grams).
2. The shotfirer had to hold at least a Deputy's Certificate and to be conversant with the conditions.
3. The firing position had to be in the intake air not less than 305m from the place to be fired.
4. The maximum number of shots was 20 using a Mark VII exploder.
5. The exploder was to be tested daily and a record of tests was kept.
6. Low tension millisecond or instantaneous detonators with copper leads only were used, connected in simple series. No delay was to be over 75 milliseconds between consecutive shots, and the maximum overall delay was 150 milliseconds. Wiring was kept to a minimum and all joints were insulated.
7. The primer was at the back and front respectively of the charge, facing the charge.

8. The shotfiring cable was a twin conductor and specified for low resistance.
9. The shotfirer tested the continuity and resistance of the circuit.
10. The shotfirer recorded details of shotfiring including the total number of shots, the types of detonators used, the number and size of rounds and details of any mis-shots.
11. Provision was made in the locked firing box at the firing point and the locked safety isolation box inbye, for short circuiting of all circuits at all times other than at firing and for protection of cable ends and exploder from unauthorized interference.
12. Weekly maintenance tests of the exploder and shotfiring installations were made.
13. Normal stonedusting was practiced.
14. The face was wetted before charging.

Additional safety measures included monitoring of return gasses and an inspection of the face 15 minutes after firing to ensure the conditions are safe for work. Instantaneous detonators were recommended within 15m of any known or suspected seam disruption.

The first and largest outburst of this series occurred on March 4th, 1960 and approximately 800 tonnes of material (Williams and Rogis, 1980) were dislodged. See figure 4.

Seven other outbursts occurred in a 300m zone during 1960 and 1961. See figures 5 and 6.

These outbursts were triggered using inducer shot firing without injury. The drivage was slow and deteriorating roof conditions caused these workings to be abandoned in 1961.

Hargraves (1963) examined the nature of these outbursts and reported the following, "The roof at State mine has an immediate roof 5feet thick followed by small upper seams. This did not occur at the adjacent Bowen Consolidated mine which was unaffected (by outbursts). Floor heave was a feature in the shale floor — particularly at intersections in first workings. The seam gas seems to escape through the small floor cracks. When cutting the face, the coal sometimes swelled sufficiently to grip the cutter. No relation to cleavage was recognised. A lateral squeeze of top coal was recognisable. Intense folding occurs east of the State mine which was likely to leave a residual stress in the coal."

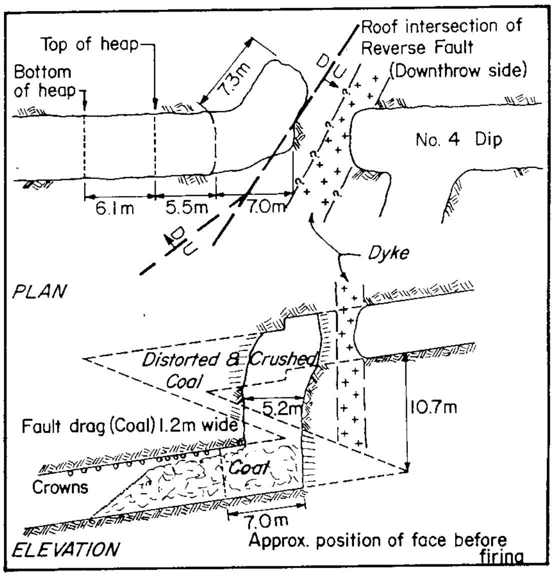

Fig. 4 — Outburst cavity — 4.3.60
(after Ogg, 1960, State Mine internal report)

OUTBURSTS IN THE DACON NO. 3 MINE

Mining at the No. 3 mine was generally in conditions considered as being outburst free. In the early 70's while mining the deeper parts of the Bowen seam near the outburst area in the old State Mine, two small outbursts were encountered. See figure 6.

The gas emission values (method by Hargraves,1963) were generally below $0.5cm^3/g$ and the area considered outburst free. In the deeper cover (240m) in the south-eastern section of the workings the emission values rose to $0.75m^3/g$ and up to 1g/cc. Two small outbursts occurred in this area where the drivage intersected a strike slip fault. See figure 7.

As continuous miners were used, the outbursts occurred with men at the face. This caused some alarm. The section was difficult to work and was subsequently abandoned.

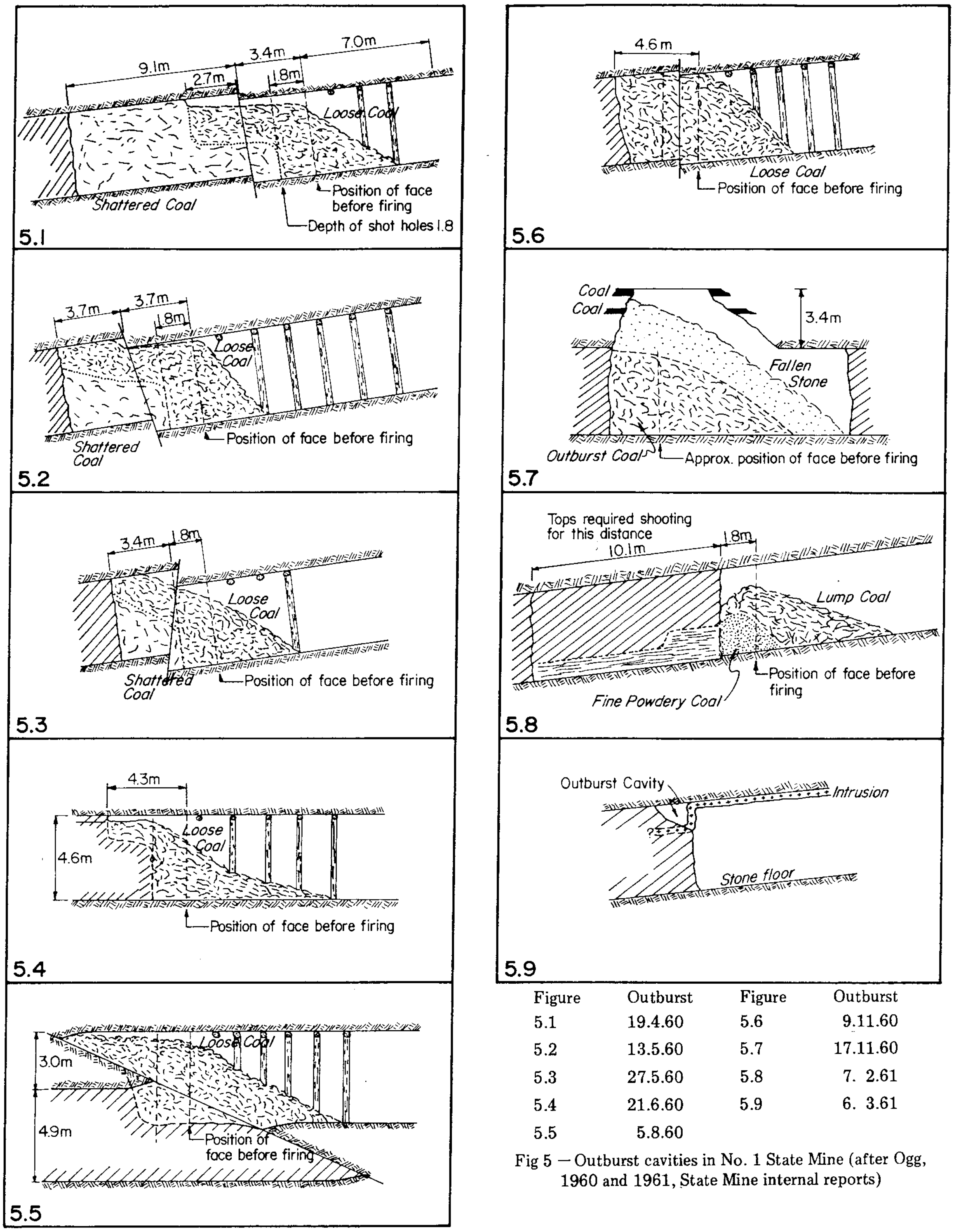

Figure	Outburst	Figure	Outburst
5.1	19.4.60	5.6	9.11.60
5.2	13.5.60	5.7	17.11.60
5.3	27.5.60	5.8	7. 2.61
5.4	21.6.60	5.9	6. 3.61
5.5	5.8.60		

Fig 5 — Outburst cavities in No. 1 State Mine (after Ogg, 1960 and 1961, State Mine internal reports)

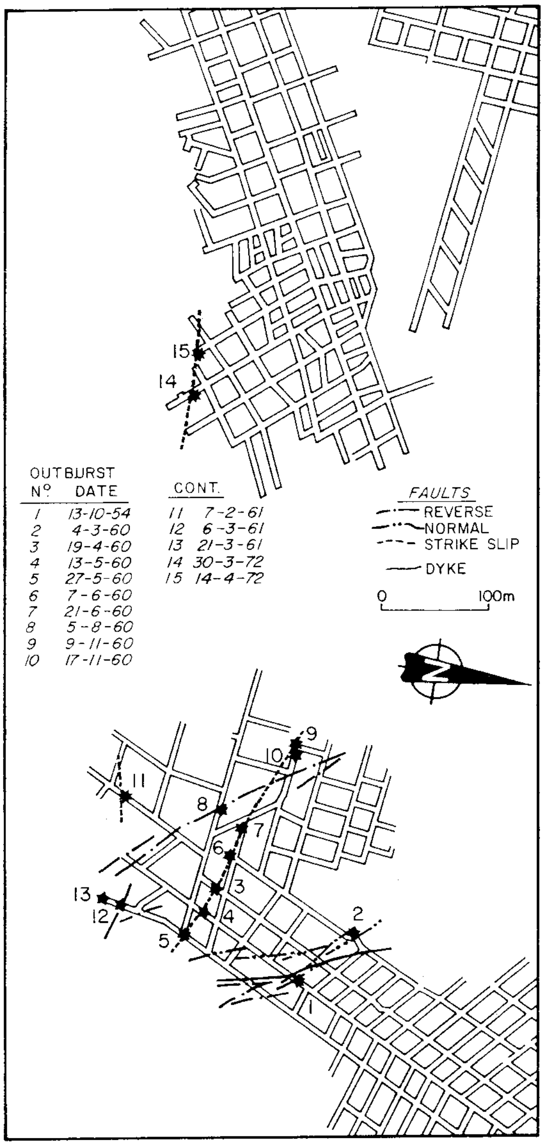

Fig. 6 — Outburst locations, No. 1 State Mine and No. 3 Mine

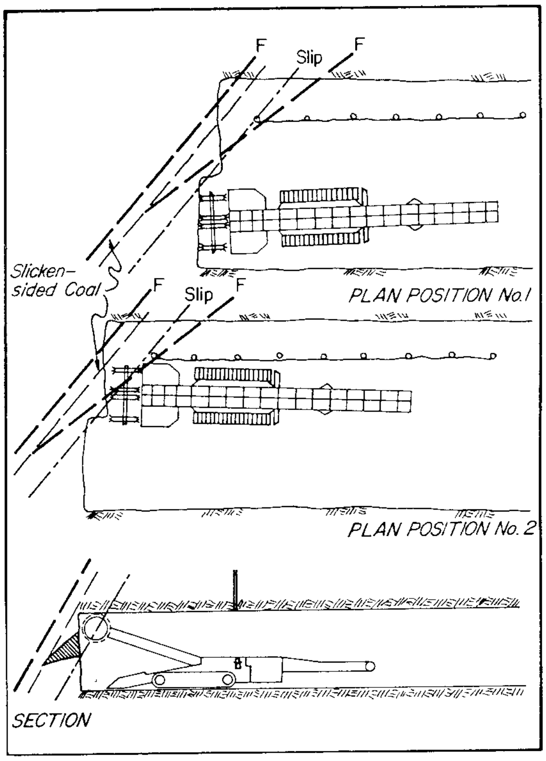

Fig. 7 — Outburst in No. 3 Mine, 30.3.72

PART II

OUTBURSTS AT THE NO. 2 MINE

CONDITIONS

The Permian Bowen seam is the second economic seam of the middle Bowen series and varies in thickness from 2.7m to 8.5m. The coal is fairly high rank,low volatile bituminous coal with low inherent moisture. A typical analysis is as follows:

Inherent Moisture	1.5%
Ash	13.3%
Volatiles	19.7%
Fixed Carbon	66.4%
Sulphur	2.1%
Specific Energy	29.2 MJ/kg
Swelling Index	4

In areas of the mine, the bottom section of the seam is burnt by igneous intrusions. The roof is a massive sandstone over a major area of the mine, it changes in the area prone to outburst to a layered sandstone-siltstone and as working gets deeper to closely laminated sandstone siltstone and shale. The floor strata is a fairly competent shale. A stone lense up to 1.3m thick is located 2.3m below the roof in the area prone to outbursts. The seam dips to the south at 7 degrees.

The workings of the mine are bounded by two major reverse faults 1.5km apart. The Three Mile Creek fault to the west has a total throw of 40 metres and the Reid Creek fault on the east has a throw of 10 metres.

Workings of the mine are approximately 4km west of No. 1 State Mine and are the most westerly of underground workings on the Collinsville field.

Development headings are driven in the top 3 metres of the seam and extend 2.3km down dip from the outcrop to 280 metres below the surface. The mine has been developed on the bord and pillar system. All drivage has been by continuous miner. The size of pillars are 30 metres by 40 metres with 6.4 metre wide roadways.

The seam gas is predominately carbon dioxide but methane is present in the seam below 55 level (220m depth). A typical analysis of the seam gas from the outburst area is,

CO_2 – 90.69 % CH_4 – 6.908 %
CO – 0.0001 % H_2 – .001 %
N_2 – 1.27 % C_2H_4 – 0.13 %

Because of the occurrence of outbursts at the nearby State Mine, seam gas emissions have always been routinely taken at mining faces. The method determines the volume of gas desorbed per gram of sample over 5 minutes.

An upper limit for mining of $1cm^3/g$ was established at Metropolitan Colliery, New South Wales and was introduced at Collinsville. This figure has been maintained by the Queensland Mines Department.

In 1967, the main development headings were stopped at 32 level for a period of 6 years because of increased seam gas and the presence of hydrogen sulphide gas. The faces were then standing on a zone of stress relief faulting with a total throw of approximately 4 metres. Seam gas emission readings were recorded as high as $0.75 cm^3/g$. Whilst the headings were standing an area of coal west to the 3 Mile Creek Fault was developed and extracted using the Wongawilli system.

Before mining recommenced in the headings, ventilation was upgraded by the installation of a second axial-flow fan. Although the headings had been standing for such a length of time and gas had continually been given off, emission values increased at a faster rate than expected and were up to $0.35cm^3/g$ within one pillar length. After mining through this zone emission values fell back to less than $0.10cm^3/g$ and mining advanced, to 59 level (250m Cover) before values rose again. At this point the seam section and roof condition also changed. See figure 8.

Mining below 59 level was slow. Emission Values were generally in excess of $1cm^3/g$ and had to be reduced before mining could advance. The heading development was stopped during 1978, after advancing 360 metres in these conditions.

The change in roof conditions resulted in a number of falls until mining widths were reduced to 5 metres and

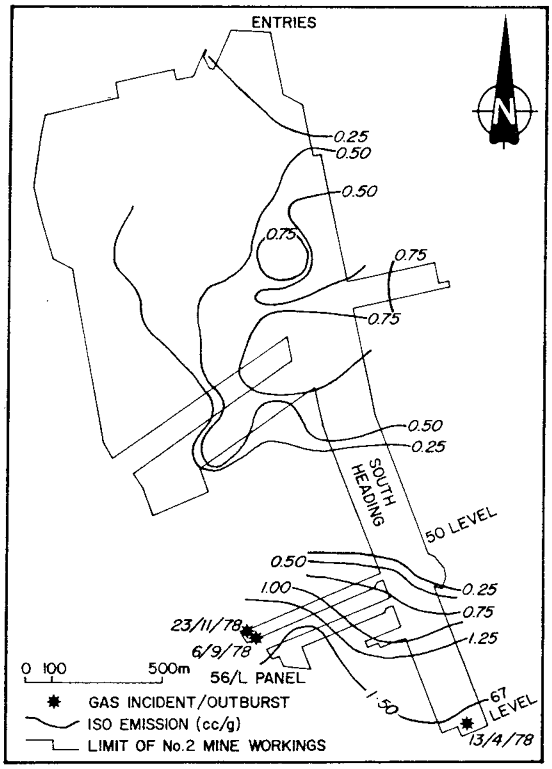

Fig. 8 — Outburst locations and gas emission in No. 2 Mine

resin anchor bolts introduced. For a time the development headings were stopped for a period of 10 months with bleeder holes bored 30 metres in advance. Mining then advanced two pillar lengths (80 metres) with little problem. This maybe attributed to a de-stressing of the area whilst the headings were standing.

Two of the six outbursts occurring in this area contained coal. The combined gas and coal outbursts occurred in a panel heading being driven west from the main development and at the site of a 6 metre reverse fault. The gas outburst occurred in the main development headings, with gas being emitted from the floor of the working section accompanied by floor heave. Each outburst was adjacent to a geological disturbance. The occurrence of one of the gas outbursts and the two gas and coal outbursts are described in the next section of the paper. See Figure 8.

GAS EMISSION FROM SEAM FLOOR
13TH, APRIL, 1978

This gas outburst occurred in No. 5 heading below 68 level in the Main Development Headings. Seam gas emission values in excess of 1 cm^3/g had been experienced for a distance of 300 metres prior to the incident. To allow mining to advance,the value was reduced by drilling three 100mm holes in the top section. Drainage was slow but relatively effective with holes bored normal to the main cleat, however very little gas drainage resulted from holes bored on the "teeth". Whilst this system was effective in the top section of the seam a number of "gas releases" were experienced from the floor of the drive. To overcome this problem four holes were drilled across the bord. They were inclined into the floor ahead of the face and penetrated the stone lense. This proved effective as a large volume of gas was emitted when the face advanced over the holes.

The outburst occurred at 11.15am on the day shift as the continuous miner was cutting coal from the west side of the face. See figure 9. Fifteen (15) cars of coal had been mined and emission readings taken in the top section (only) gave readings of 0.42, 0.47 and 0.34 cm^3/g.

The miner operator was the only person at the face and reported he heard a dull bump and the continuous

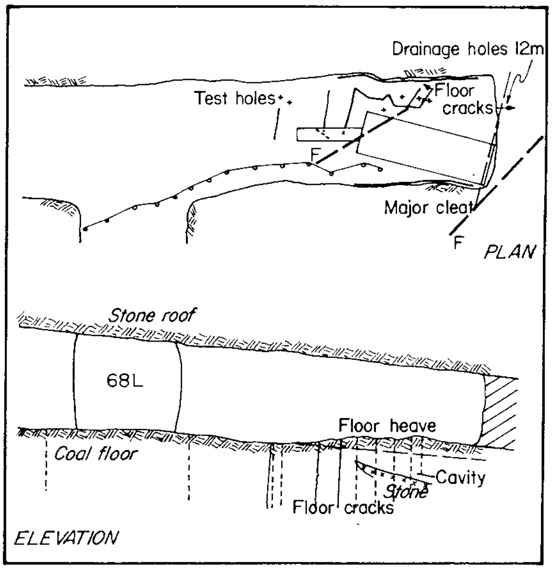

Fig. 9 — Gas emission site 13.4.78

miner was pushed sideways. He observed the props beside him bend and felt gas (CO_2) building up. He left the face immediately. Other crew members situated at an intersection just outbye were not aware of the problem. The deputy reported gas to a height of 1.5m at the tail of the miner 10 minutes after the incident.

Ventilation at the face was 5.1m^3/sec. On inspection it was evident the gas had been emitted from floor cracks associated with a floor heave of approximately 0.6 metres. It was not possible to determine the volume of gas given off. A gas emission taken in the face after the outburst gave 0.51cm^3/g.

Mining bottom coal at the site exposed a reverse fault with an upthrow of 0.95 metres. This fault had been intersected 40 metres back from the face in the top section of the seam. It was proven that floor holes had not been bored in this area.

GAS AND COAL OUTBURST
6TH, SEPTEMBER, 1978

This outburst occurred at approximately 1.30pm on the day shift at the face of 53½ L in a panel being driven to the west of the main development. The headings had advanced 100 metres in an area where seam gas emission values had exceeded 1 cm^3/g.

Development had been completed under the following precautions.

1. A 100mm pilot-hole was maintained 10 metres in advance of the face to provide information on geology.
2. Four floor holes were drilled across the face and were inclined ahead to guard against gas outburst from the floor.
3. Pulsed infusion shot firing was used to reduce the emission level below 1 cm^3/g.
4. Face ventilation was well-maintained (no limit set).
5. AUER SR 16B self rescuers were worn at all times.
6. Shift inspection sheets were used to collect information related to outburst forecasting. See appendix A.

Twenty cars of coal had been mined on the shift, and the face had advanced 12.3 metres. Seam gas emission readings had been 0.34, 0.72, and 1.10 cm^3/g. The reading of 1.10 cm^3/g was obtained from the north side of the face after noticing an increase in CO_2 level at the miner operator. Mining had continued on the south side to square the face to enable roof support to be erected.

The continuous miner was cutting coal with the head against the roof when the outburst occurred and no doubt this restricted the blast. See Figs. 10 and 11.

After an increase in the CO_2 level was observed by the operator and a reading of 1.1cm^3/g was recorded from the

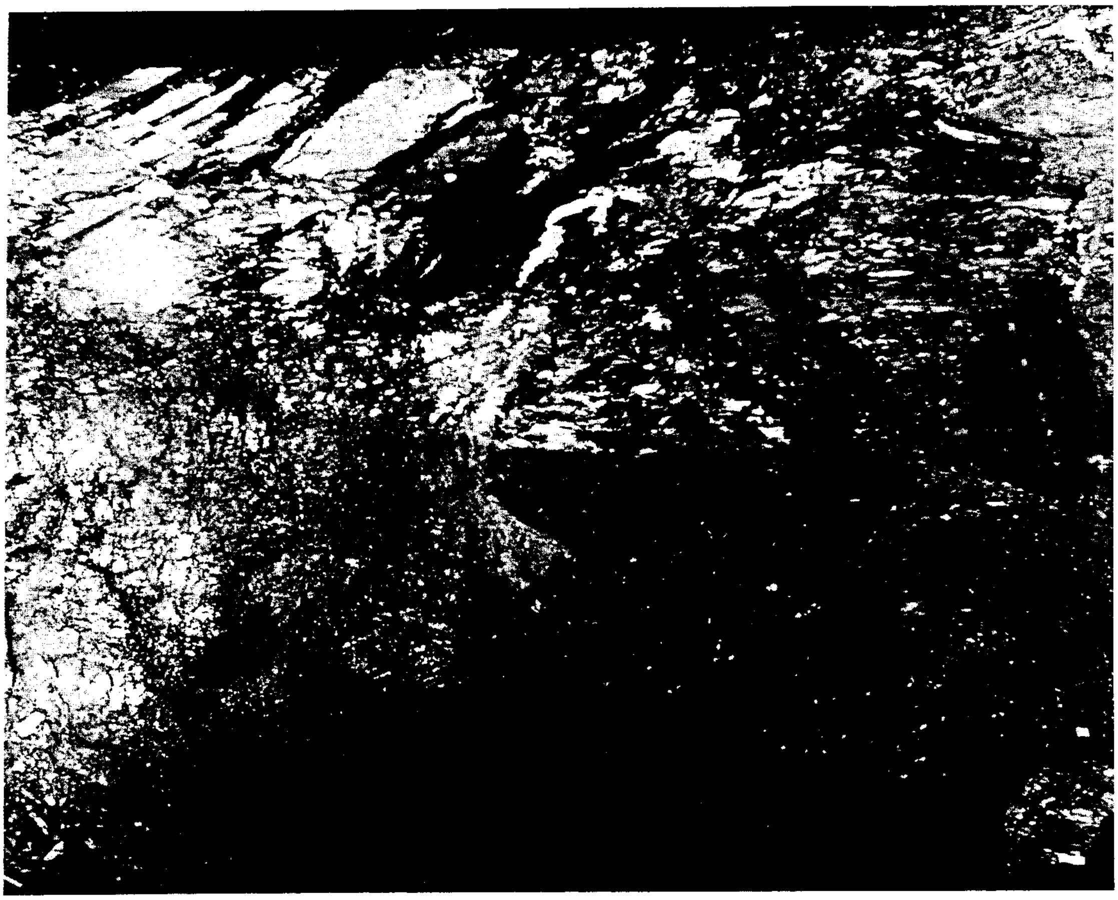

Fig. 10 — Outburst cavity, 6.9.78

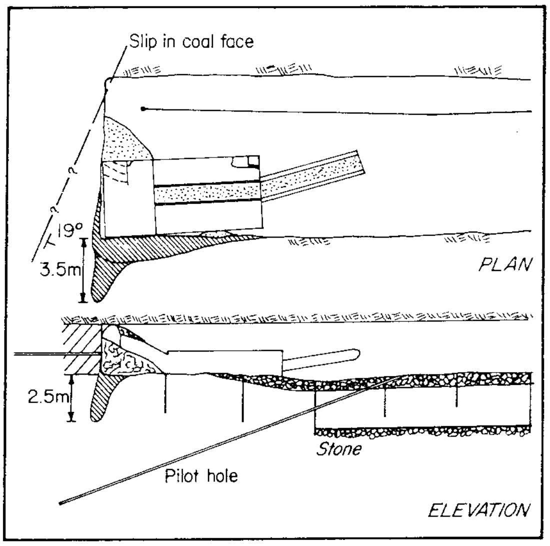

Fig. 11 — Outburst site 6.9.78

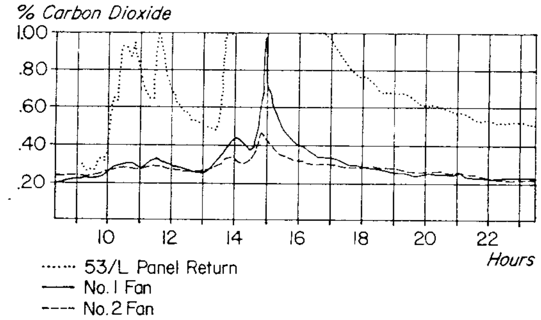

Fig. 12 — Gas chart, 6.9.78

north side of the face.

No warning noise was heard by the crew. They left the face when coal stalled the conveyor on the miner and spilled out over the head. $2420m^3$ of gas was given off in 3.5 hours following the outburst.

Recordings on the M.S.A. gas monitor indicated a peak of 1% CO_2 at the main ventilating fans and a level in excess of 1% for three hours in the panel return. See gas chart Figure 12. Face ventilation at the time was $4.6m^3$/sec.

Approximately 25 tonne of coal was loaded from the outburst site. The cavity took the shape of a 3 metre diameter cone extending downwards at an angle of 45 degrees for 4 metres into the left hand rib.

Exploration drilling proved a 6 metre downthrow reverse fault ahead of the face and this was later mined.

After this outburst, the additional measures were adopted for mining in areas where the seam gas emission exceeded 1 cm^3/g.

1. Three pilot holes were maintained 15 metres in advance of the face. The rib holes were flanked slightly outward.
2. A minimum air ventilation of $9.5m^3$/sec was maintained.
3. The face was kept as square as practicable with a maximum advance per cut of 1.35m.
4. Housekeeping was maintained at a high standard in the face area.
5. Siebe Gorman compressed air units were purchased to replace AUER 16B. These were kept on the machines and in a box at the last cut-through. Subsequently, Medic Air bottles were installed on the continuous miners.

The investigation sheets are included in Appendix A.

GAS AND COAL OUTBURST
23RD, NOVEMBER, 1978

This outburst occurred on 53 level during the afternoon shift. The level had penetrated into a six metre reverse down throw fault and had advanced approximately nine metres under the lip. This is the same fault from which the outburst occurred in 53½ level on the 6th September, 1978. See Figures 8 and 13. The previous shift had reported the coal as being very disturbed in the interval 0.8 to 1.3 metres below the roof. The entire face was very "drummy". Difficulty was experienced in drilling the gas emission holes as the drill kept bogging in the soft coal. There was an increase in gas activity at the end of the day shift and the gas emission had been 0.93 cm^3/g.

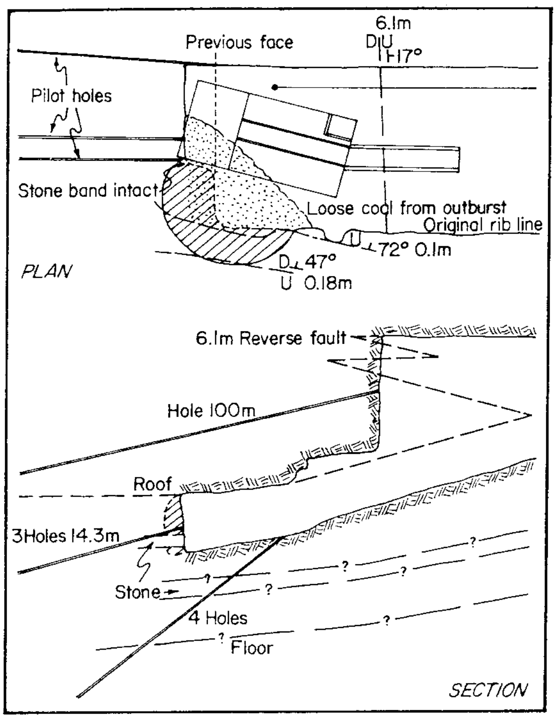

Fig. 13 — Outburst, 23.11.78

Emission values the previous day had been 0.30 cm^3/g and a small cone shaped burst occurred in the left hand rib emitting 230m^3 of gas.

The level was being advanced according to established safety measures. Three pilot holes extended 14.3m in advance of the face and four floor holes had been drilled 6.1m into the floor.

The afternoon shift was given instructions to square the face and bore bleeder holes ahead and into the floor and insert the venturi extractor.

The second car of coal was being loaded from the north side of the face as the outburst occurred from the south side of the bord. See figures 13 and 14.

It was reported that a noise was heard above that of the miner and was described as a "whoosh". The deputy stated the coal appeared to "roll out" and push over the head of the miner.

Gas built up to the rear of the shuttle car (15 metres from the face). The crew had ample time to retreat from the face.

It was reported that a puff of red dust was seen before the coal moved.

Ventilation at the face was 13.8m^3/sec. Recording from the M.S.A. gas monitor showed CO_2 in excess of 1% for a period of 18 minutes at a sample point in the panel return. See the gas chart figure 15. 260m^3 of gas was emitted in twenty-five minutes and 940m^3 over six hours.

Approximately 19 tonnes of outburst coal was recovered.

The outburst was confined to the top section of coal above the stone lense and extended back approximately 3 metres from the original rib line. A small strike slip fault with a throw of 100mm was evident at the site after the outburst.

Fig. 14 — Outburst cavity, 23.11.78

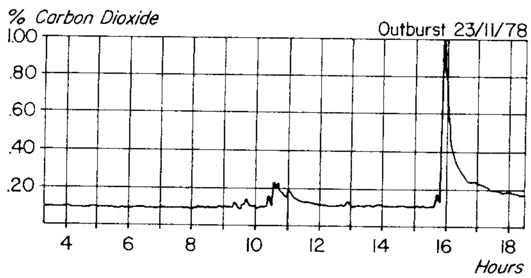

Fig. 15 — Gas chart from No. 1 Fan 23.11.78

Emission reading taken on the 27th September following the outburst still gave readings of $0.94cm^3/g$, two metres in advance of the face.

The investigation sheets are included in Appendix B.

PRECAUTIONS AGAINST OUTBURST OCCURRENCE

Using the current techniques, the reserves of the Bowen seam have been extensively mined to a depth of 250 metres. In order to mine coal down dip of the existing workings, the application of science to mining is required to overcome the problem of outbursts. The problem has not been solved, but the methodology is apparent. The precautions taken against outbursts may be divided into three groups.

1. Areas most liable to outbursts are defined.
2. Means are devised to reduce the seam gas and stress.
3. Mining practices are used which give adequate protection to the men underground.

Historically most emphasis has been put on reducing seam gassiness which was measured by the emission values. Methods including water infusion, bleeder holes, pulsed infusion shotfiring and suction gas drainage. The mining practices have evolved as shown in the previous section.

A research programme co-ordinated by C.C.P. is assessing the geological influences on outbursts and developing options available for mining the deeper coal reserves.

SEAM GAS EMISSION

The upper limit for mining of $1cm^3/g$ was introduced at Collinsville in the hope of preventing outbursts. Readings were taken by the section deputy at the following frequencies.

Value less than
$0.7cm^3/g$ each pillar length (30m)
$0.7 - 0.8cm^3/g$ each shift
$0.8 - 0.9cm^3/g$ each half shift
$0.9 - 1.0cm^3/g$ each 3 metre advance

Results were periodically checked by the survey department. Even though readings were taken regularly, on outburst did occur whilst mining at emission value less than $1cm^3/g$.

The method used for determining emission values is quite simple and reasonably accurate, however, anomolies can occur in the one face depending on the sample position and type of coal. One cause of this may be attributed to the influence of stress.

An experiment was undertaken in a face in 56L West Panel to determine variations at different horizons and positions across the face. In all 47 readings on 0.6m grid were taken.

The face had been standing 32 hours with four 80mm bleeder holes drilled in advance. It was considered these holes had little effect on results as the heading was being driven parallel to the major cleat and gas drainage was minimal.

Two emission meters were used and the exercise took 3 hours 10 minutes. The meters were cross checked on a number of test holes.

Values varied from $0.43 - 1.15cm^3/g$. The average of all readings being $0.79cm^3/g$. See table 1.

43% of the readings occurred within $\pm$ 0.1 cm^3/g of the averages. 72% of the readings occurred within $\pm$ 0.2 cm^3/g of the averages. Highest readings were obtained towards the virgin side rib and closest the floor of the drive. Also it was noted on a number of occasions that abnormally high readings were obtained from a face prior to holing into another drive.

The mining cycle in the development heading was such that adjacent headings were driven beyond the cut through alignment thus allowing bleeder holes to be drilled along the direction of the cut through. After standing for a time it was possible to drive the cut-through from either direction. However, little problem would be experienced until mining approached within 4 metres of holing and emission values would rise sometimes beyond $1cm^3/g$.

Because of its simplicity, the method has to be recognised until a more reliable one is found.

equalled the infusion rate. On ceasing pumping, gas flow was negligible, with water continuing to flow from the entire face area. The water was charged with gas, being very effervescent. Gas flow increased as the water was displaced. The face was usually mined immediately after infusion. On one occasion, it was not possible to mine a site, and 40 hours after infusion, g as flow had stabilized at 0.8 times the flow before treatment.

BLEEDER HOLES

Three 80mm diameter holes were drilled in advance of the face for a distance of up to 40 metres. One hole was on the bord centre line, with one flank hole either side. The flank holes were angled out at 5^{O}, with all holes being drilled parallel to the roof. Within three days after drilling, it was possible to mine to the extent of the holes before high gas was encountered. See figure 17.

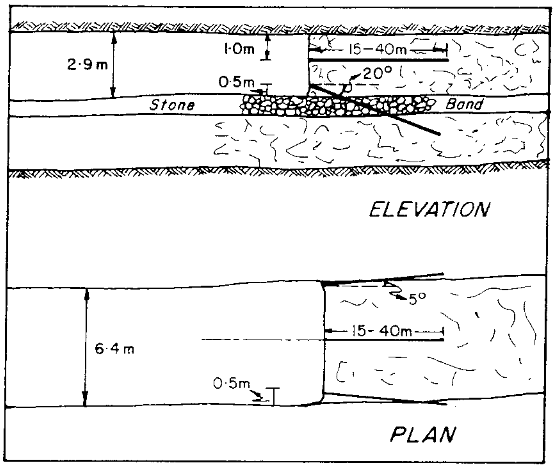

Fig. 17 — Layout of bleeder holes

The holes were drilled for two reasons. They allowed gas to drain from the area to be mined, and they gave a warning of any major faulting.

The 80mm diameter holes were drilled using a Gardner Denver A.T.D. 2000 rotary machine. An Atlas Copco ROC606 machine was later purchased and 100mm diameter holes were then drilled, with a higher penetration rate. Although tests were limited, there was no apparent increase in gas flow rate from the larger diameter holes. Previous tests indicated that 80mm diameter holes were much more effective than 43mm diameter ones.

The following records from a face illustrate this. Emission Value before drilling three 20m x 43mm holes — $0.96cm^3/g$.

E.V. 250 hours after drilling — 0.91 cm^3/gm.

E.V. 40 hours after drilling three 30m x 80mm holes — 0.53 cm^3/gm.

E.V. after mining 27 metres — $0.85cm^3/gm$.

Tests with bleeder holes were implemented when water infusion was found to be impractical. Comparisons were done between the two methods, and it was found that while infusion allowed mining for an average of only 15 metres, it was possible to mine to the extent of bleeder holes. See figure 18.

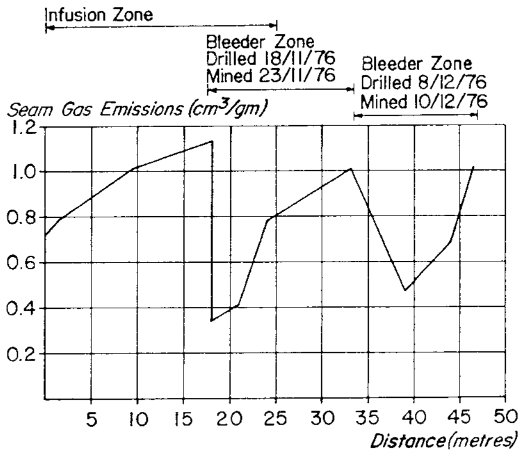

Fig. 18 — Comparison of the effects of infusion and bleeder holes

The dip roadways were normal to the cleat and were driven first. Because of natural bleed-off along cleat into the dip roads, no holes were required before mining the levels. When holes were drilled in the levels, that is, parallel to the cleat, gas flow was negligible. Holes drilled on the dip roadways showed considerable variation. See figure 19.

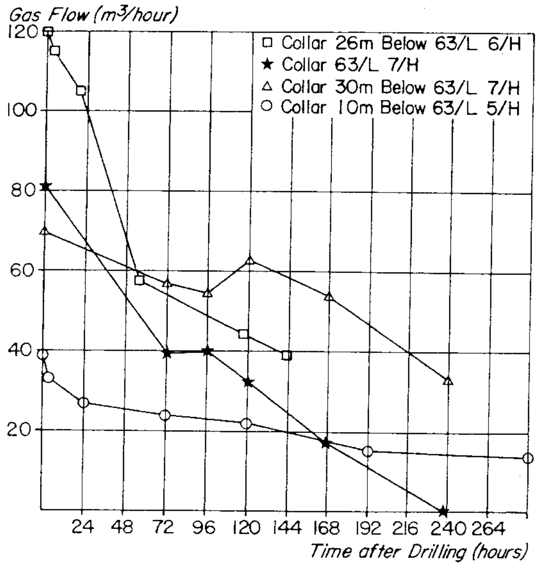

Fig. 19 — Flow from bleeder holes (80mm dia)

As well as the variations in flows on different faces, there was other differences as shown by figure 20 which indicates variations in flows recorded on three holes drilled parallel and in the same horizon on the same face.

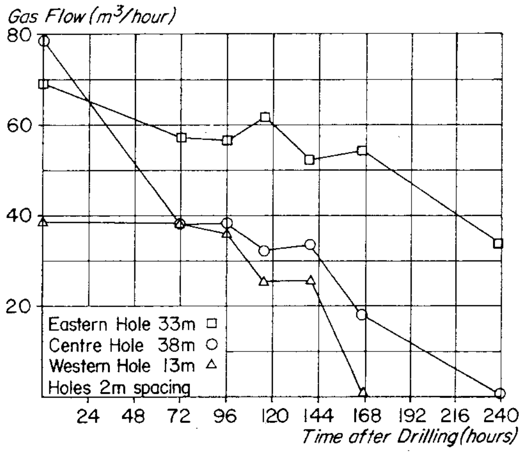

Fig. 20 — Flow from bleeder holes in No. 7 heading below 63 level

Despite the flow variations, it was possible to predict with considerable certainty when a site could be mined. This is illustrated by the curves in Figure 19. Flow rates decreased steeply for the first 72 hours, after which the curves flattened out. At the point, where the curves levelled, water would begin to flow from the face holes. It was found that once this water flow commenced, it was possible to mine to the extent of the holes before obtaining high gas emission values.

Using bleeder holes, it was possible to achieve acceptable output.

PULSED INFUSION SHOTFIRING

Three 43mm diameter holes are drilled across the face at about mid-working height. The holes are then charged with submarine explosives and sealed at the front using dilatable seals. Connections are made to a pressure pump and the holes pressurized. After reaching the predetermined pressure the explosive is detonated.

The explosive wave is transmitted along the full depth of the hole via the water to open up cleavages and allow the release of the gas. Water is continually pumped into the face after the firing to displace gas.

A Petrometal Type SP400-13 air operated pump, and one metre long Stauff dilatable seals were used with ICI Hydrobel explosives and Hydrostan detonators. Up to 1.36kg of Hydrobel per hole was used. Water pressure was varied up to 7Kpa and hole depths ranged from 4 to 9 metres.

Thirty-nine faces were treated with pulsed infusion, between 16th May and 26th October, 1978. On seventeen occasions, seals were ejected from the holes on firing. One misfire occurred, but, the detonator initiated when reconnected. On numerous occasions, explosive found lying on the floor after firing, and a new hazard was introduced.

The operation of drilling, loading, sealing, pressurizing and firing took three men an average of two hours. This made the infusion time consuming, in a section that was already uneconomical. Frequently, extreme difficulty was experienced in loading, due to the shattered nature of the coal in potential outburst zones. Infusion was far from complete. It would take 16 to 20 hours to infuse the face fully.

In most instances, there was a slight decrease in the emission value. At 53 level, 28 metres inbye 10A Heading, the face was fired on the 18th, 19th and 20th September, with no reduction in emission values. By the 21st September, there were four floor holes, three pilot holes, eleven pulsed infusion and four emission holes in the face. The emission values were still 1.0 cm^3/g. Three infusion holes were sealed and connected to a Venturi extractor. Emission values were reduced to an acceptable level within two hours.

One small outburst occurred in a zone that had been pulsed infused. On this occasion (27/10/79), the face (53 Level, 24.2 metres inbye 11A Heading) had been fired some fifteen hours before the outburst. The face was advanced 1.2 metres when the outburst occurred.

As the zone over which the reduction in E.V.'s could not be increased, and the infusion operation could not be fitted into the mining sequence without consistently reducing output, it was concluded that water infusion was not a practical solution to the problem.

GAS DRAINAGE

Small scale tests were conducted on gas drainage during 1978 in 53L West Panel where previously bleeder holes and pulsed infusion shotfiring had failed to reduce the emission values. See figure 21. Driveage in this panel is parallel to the main cleat of the seam and consequently presented the most difficult conditions for trials.

A Stauff Venturi stone duster capable of creating a vacuum equivalent to 250 mm. of mercury was used in the trials.

Three 43mm holes were drilled across the face to a depth of 6 metres at mid-working height and approximately 2 metres spacing. Each hole was sealed using Stauff dilatable seals and coupled via a manifold to the Stauff Venturi duster. See figure 22.

In one case following the failure of pulsed infusion shotfiring, gas drainage methods lowered the emission values from 1.08cm^3/g to 0.77cm^3/g inside two hours and mining was able to advance 6 metres before values again rose in excess of 1cm^3/g.

Flow rates from gas drainage holes were increased by up to 30 times with this method and the success of these

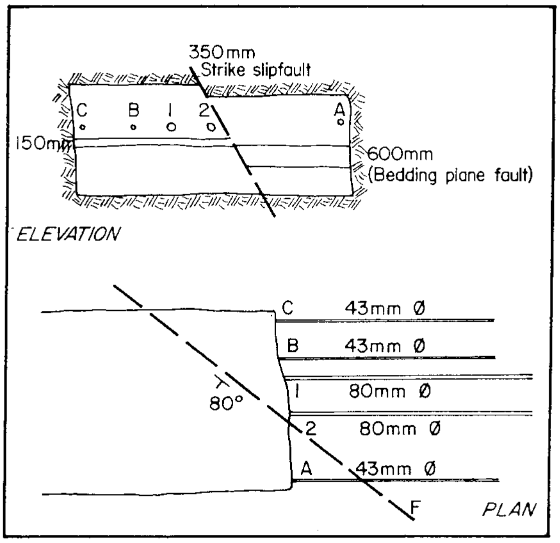

Fig. 21 — Gas drainage site 53 level west

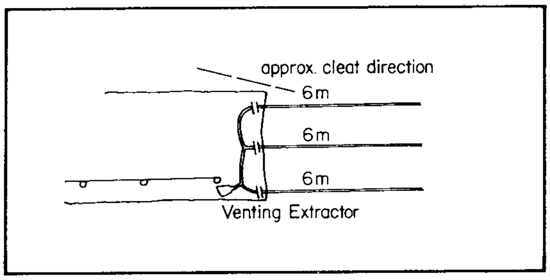

Fig. 22 — Method of gas drainage

small scale trials prompted the purchase of a Nash Model AL673 drainage pump.

RESEARCH INTO THE OCCURRENCE OF OUTBURSTS

The potential for outbursts to occur in the Bowen seam will probably increase as deeper coal reserves are mined. Investigations are being undertaken to define the outburst potential and to reduce their occurrence and severity.

The Company supports a coal research section of its own as well as encouraging research programs being carried out by the Australian Coal Industry Research Laboratories (A.C.I.R.L.) and the Commonwealth Scientific and Industrial Research Organization (C.S.I.R.O.).

The current research consists of:

a. defining and using geological parameters which relate to outbursts as discussed by Williams and Rogis (1980)
b. developing instantaneous gas monitoring systems and
c. improving suction drainage practice

CONCLUSIONS

Outbursts at Collinsville have three things in common.

1. The coal has a depth of cover greater than 200 metres.
2. The coal contains a large amount of gas.
3. The outburst occurs at the site of some prior geological disturbance.

In a geologically complex mining area such as Collinsville, a study of the geology in existing workings, exploration boreholes and developing headings must be actively pursued if fault delineation and coal property variations are to be useful in predicting outbursts.

The measures taken to prevent outbursts have met with mixed success. Of the various techniques available, vacuum gas drainage appears to be the most effective method of extracting seam gas in high gas faces. In the small scale tests conducted to date emission values were consistently and rapidly reduced so as to permit the continuation of mining. Research into the more detailed aspects of gas drainage at Collinsville is currently being undertaken jointly by C.C.P. and A.C.I.R.L.

Outburst problems have been resolved overseas by working a lower seam in close proximity first, however, at Collinsville the lower Blake seam is expected to present similar problems. In view of this it is suggested that the mine design must keep the number of development headings in virgin areas to a minimum. Headings should be advanced using a remote controlled continuous miner operated from a distance of 50 metres with a very high standard of face ventilation.

The major volume of production could then be obtained from a system such as longwall where the rates of advance are slower. A destressed zone ahead of the longwall face may successfully reduce in the hazard in outburst prone areas. Because of other problems at Collinsville including seam thickness, restricted areas due to faulting and the varying thickness of coal above the stone lense, other methods will have to be investigated.

Communications play a major part in the working of difficult areas. The work force at Collinsville demonstrated a high resistance to mining the areas prone to outburst because of their concern for safety. Weekly meetings of approximately 1 hour duration were held with the production crews from the affected area. These covered a resumé of information and happenings during the past week and developments intended in the future and also answered any items of concern.

This also has the advantage of keeping employees aware of the problem and protects against the development of a lax attitude.

High costs are incurred when mining deep coal in poor conditions. Because of the low production achieved when mining areas prone to outbursts and the need to spend considerable amounts of money on research, mining companies require financial assistance if these resources are to be recovered. The present selling price of coal on the local markets cannot carry an operation with this problem. The subsidy could be in the form of a higher price for coal mined from these areas or annual lump sum payment for development and research.

ACKNOWLEDGEMENTS

The permission of the Collinsville Coal Co. and M.I.M. Holdings Ltd. allowing the preparation and presentation of this paper is acknowledged. The views expressed are not necessarily those of the companies or the Queensland Mines Department.

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APPENDIX A

INVESTIGATION REPORT

1. Date: 6th September, 1978
2. Time: 1.30 p.m.
3. Location:

   53½ Level 12A Heading 23468N 83061E 255 metre cover.
4. Previous E.V.: Emission (cm^3/gm)

1.1	0.72	0.54	1.20
Date			
6/9	6/9	6/9	5/9
Distance from site (m)			
0	8.8	12.3	12.3

5. Direction of Drivage: West
6. Gas Reduction Method:

   Bleeder — 1 x 39m x 100mm diam. pilot holes, collar 28.5m back — Drilled A/S 31.8.78. 4 x 15m c 100mm diam. floor holes, collar 12.3m back. Dip 26°. Drilled A/S 4.9.78 (refer outburst plan).

   Comments:

   It was noted that there was considerable crushing of the pilot hole, for 20 metres before the outburst site. This is not normal. The only previous occurrence of this was at 63 Level 6-7 Heading. At this site, the floor heaved and gas was released.

   Pulsed Infusion:

   3 x 43mm diameter x 6 metre fired N/S 5.9.78. 400 psi water pressure — 4 sticks Hydrobel per hole.

   Comments:

   When the face was being infused, water was seeping from the entire face (unusual): The deputy; C. McKenzie, stated that he did not hear the shot go off, but said he did bump his elbow against a five gallon drum as he turned the exploder. He said the shot appeared good, with the seals remaining in the holes. The soft slickensided band slumped out approximately 0.2 of a metre, leaving a 50mm gap on the parting with the harder band above.
7. Distance Mined since standing: 12.3 metres.
8. Outburst Location: Bottom rib at face.
9. Miner Location: Bottom Rib.
10. Operation being Performed:

    Cutting out bottom side to square up face.
11. Activity Before Occurrence:

    Audible warning — two bumps 30 minutes previously (similar to car running into miner), while mining top side. 20% CO_2 at miner driver at same time. Two areas of fractured coal (similar to bulled-out shot); one at corner of face on top rib, just under soft band; other at corner of face on bottom rib on previous cut (4m from outburst site). Both were conical, 0.3m diam. 0.5m deep.
12. Changes in Coal Properties

    Before Outburst:

    Hardness — none apparent.

    Roof — no apparent change in strata.

    Moisture — none apparent in close vicinity — seam appeared to contain large quantity of free moisture under stone band for 50m back from site.
13. Changes in Desorbed Gas

    Quantity: Gas emission values no higher than usual for this area of the mine.

    From Floor: Floor holes appeared to have higher flow than normal.

    From Face & Ribs: Appeared to be more gas coming from face than normal.
14. Changes in Geological features before:

    Roof jointing — none apparent.

    Cleavage direction — none (62°).

    Cleavage spacing — none.

    Faults — normal fault dipping 83°N 0.5m displacement estimated to be 9m above top rib (north).

    Slips: Level 60° Faults 351° Dip 19°E.

    Description of Plane: gouge along plane in roof, heavily slickensided in coal.

15. Seam Gas Analysis:
    CO_2 - 92.09%
    CO 0.0001
    CH_4 - 6.54%
    H_2 - 0.01%
    N_2 - 1.2%
    C_2H_6 - 0.067
16. Quantity of Gas Evolved:
    At Outburst: = $242m^3$
    1½ Hours before Outburst = $670m^3$
    3½ Hours before Outburst = $450m^3$
    5 Hours before Outburst = $436m^3$
17. Air Quantity at Face: $4.6m^3$/sec.
18. Tonnage of Coal: 25 tonne (approx).
19. Position in Bord of Release: Floor and bottom rib at face (refer to Plan, figure 10).
20. Intensity of Release: Miner head at roof, coal filled shovel, stalling flights, coal was coming back over the top of the head, along with a cloud of dust. No noise or movement apparent at time of outburst.
21. Duration of release: Not known — men departed. It was reported the gas was at the back of the car (15 metres from face + 1% for 3 hours) as men retreated.
22. Time for Gas to Clear: Not known, only some CO_2 on floor around miner 45 minutes after.
23. Nature of Coal Where Outburst Occurred:

    Extremely soft and slickedsided (analysis on plan) — hard bright band near roof appeared distorted as area was evacuated.

    Total Moisture — 4.8%, Inherent Moisture — 2.4%, Volatiles — 20.7%,

    Ash — 13.0%, C.S.1. - 5.5.

    Channel samples to be collected for maceral analysis, at intervals back from site.
24. Activity Immediately After:

    Temperature Change: None apparent Smell: None

    Haze: Yes — similar to heat haze, at face.
25. Change in Seam Thickness: Stone band not in floor — (refer Plan).

    Change in Seam Gradient: None (major fault).
26. Other Relevant Comments:

    It was stated that the ventilation was fluctuating all day. There were no changes being made to the ventilation system. The men present thought the fans had stopped when the outburst occurred. The brattice line was 78m. long. The air quantity at the panel regulator was $13.4m^3$/S. The fluctuation would have been caused by the gas quantities being evolved during mining.

APPENDIX B
INVESTIGATION SHEET

1. Date: Thursday 23rd November 1978.
2. Time: 3.25 p.m.
3. Location: 53 Level 48.7 metres inbye 11A Heading
4. Previous E.V.: See Table 2
5. Direction of Drivage: West.
6. Gas Reduction Method:

    Because of the low gas emission values, no gas reduction was considered necessary. Three 100mm. diameter pilot holes extended 14.3 m in advance. Four 100 mm diameter floor holes extended 6.1 m in advance. The instructions for the shift were to mine a further 1.2 metres, then to drill pilot and floor holes, and install the Venturi extractor.
7. Distance Mined Since Standing: 1.2 metres.
8. Outburst Location:

    Bottom side of face and bottom rib. Stone band remained intact, the coal above outbursting. (Refer Plan in figure 13).
9. Miner Location: Top Rib.
10. Operation Being Performed:

    Advancing top side. Cut was 1.2 metres, the head cutting the stone band in the bottom section of the face when the outburst occurred.

11. Activity Before Occurrence:

There was an increase in gas activity at the end of the dayshift. The emission value being $0.93m^3/gm$. Because of this increase, the afternoon shift were the instructed as stated in six (6).

12. Changes in Coal Properties Before Outburst:

0.8 to 1.3m soft, slickensided, very disturbed,

1.3-1.8 m stone — very hard.

1.8-2.2 coal — bright.

Roof: Conditions had deteriorated due to the close proximately of the fault plane. No significant change.

Moisture: Water was present along the major fault plane.

TABLE 2

Date	27.10	30.10	10.30	30.10	31.10	31.10	1.11	1.11	3.11	21/11	22/11
Time	9.45a	8.00a	11.00a	1.30p	8.15a	10.30a	8.00a	11am	9am	1.30p	
Emission cm^3/gm	0.76	0.26	0.94	0.98	0.64	1.02	0.70	0.84	0.26	0.30	0.93
Distance Advance Since last Reading(m)	2.7	nil	2.2	3.8	nil	2.2	4.8	3.0	2	2	2
Distance from Outburst(m)	23.7	23.7	21.5	17.7	17.7	15.5	10.7	7.7	5.7	5.7	
Distance From Fault	16	16	13.8	10	10	7.8	3	0	2	4	7.7

Coal Composition: None visibly apparent — the coal was very disturbed over a distance of 40 metres from the fault.

13. Changes in Desorbed Gas Quantity:

As mentioned in (11), increased activity apparent on 21.11.78, although the gas emission value was low.

14. Changes in Geological Features Before:

Roof Jointing: None apparent.

Cleavage Direction: None apparent.

Faults: 7.7 metres west of 6.2 metre throw reverse fault, dip 17 deg. E. On loading out the out-burst coal coal, two .1 metre throw normal faults were exposed.

15. Return Gas Analysis: CO — Nil CH_4 — 0.25%

CO_2 +2% O_2 — 19.75%

16. Quantity of Gas Evolved: 940 m^3 over 5.6 hours. Estimated maxiumum CO_2 — 2.4%; $260m^3$ gas over 25 minutes immediately after outburst.

17. Air Quantity at Face: $13.8m^3/sec$.

18. Tonnage of Coal: 19.2 tonnes

19. Position in Bord of Release: Bottom side of face and bottom rib.

20. Intensity of Release: The deputy stated that the coal appeared to "roll out". The edge of the outburst coal was 2.7 metres from the face. Gas built up to the back of the shuttle car. (15m from face). The build up was not immediate, ample time available for the retreat of the crew.

21. Duration of Release. The bottom side of the face was seen to roll out, followed by a build up of gas +1% for 18 minutes at the panel regulator (CO_2).

22. Time for Gas to Clear: Unknown — less than 15 minutes.

23: Nature of Coal Where Outburst Occurred:

The coal was very disturbed in the interval 0.8 to 1.3 metres from the roof. The entire face coal was very"drummy". Difficulty was experienced in drilling the gas emission hole, as the drill kept bogging in the soft coal (unusual).

24. Activity Immediately After: Temperature change; the warmth of the CO_2 gas was felt.

Smell: Non apparent

Haze: A red dust haze was seen for an instant before the coal "rolled out".

Gas Activity: Normal.

25. Change in Seam Thickness: Top rib 0 − 0.8 coal − medium bright, hard; 0.8 − 1.6 coal − soft; 1.6 − 2.1 stone. Bottom rib − 0.0 − 0.8 coal, med. bright; 0.8 − 1.3 coal soft; 1.3 − 1.8 stone.

26. Other Relevant Comments: All outbursts to date have occurred on the bottom side of the face. This is the first one when the miner has not been mining the bottom side. A puff of red dust was seen before the coal "rolled out". There was a noise before, described as a "whoosh". This was audible above the noise of the miner.