

SAFETY IN MINES RESEARCH ADVISORY COMMITTEE

SIMRAC

Final Project Report

**Title : REVIEW OF PRACTICES FOR THE PREVENTION,
DETECTION AND CONTROL OF UNDERGROUND
FIRES IN COAL MINES**

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SYNOPSIS

A statistical review is given of the frequency of fires and flammable gas ignitions in South African underground coal mines, both on a simple numerical basis and in relation to underground coal production, for the years 1970 - 1992.

Previously publicised guidelines are referred to and changes in the industry, since the previous guidelines were compiled, are discussed.

This is followed by a more detailed account of fires and ignitions which took place during the years 1986 - 1992, broken down colliery by colliery and including summarised descriptions of the circumstances surrounding many of the incidents.

Detailed recommended guidelines are then presented, based partly on lessons to be learned from incidents recorded and also on individual codes of practice already in existence on collieries and in the mining groups. Finally, in a series of Appendices, theoretical and technical background information relating to underground fires is given for those practitioners who need to have easy access to such information.

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1 FIRE STATISTICS 1970 - 1992

Statistics for the period 1970 - 1985 are taken almost directly from the work of Morris and Badenhorst. The figures for the years 1986 - 1992 are based on information reported to the Mines Inspectorate, as required under Mines and Works Regulations 25.6(c)(iv).

Table 1 UNDERGROUND FIRES AND IGNITIONS 1970 - 1992

Year	Prod. mega-tons	Spontaneous Combustion		Electrical		Flammable Gas		Other		Total	
		Incid.	Incid. per 10 mega-tons	Incid.	Incid. per 10 mega-tons	Incid.	Incid. per 10 mega-tons	Incid.	Incid. per 10 mega-tons	Incid.	Incid. per 10 mega-tons
1970	52,77	7	1,32	3	0,57	-	-	-	-	10	1,89
1971	53,1	2	0,37	-	-	1	0,19	-	-	3	0,56
1972	53,18	3	0,56	1	0,19	2	0,38	2	0,38	8	1,51
1973	54,87	6	1,09	1	0,18	2	0,36	2	0,36	11	1,99
1974	57,47	6	1,04	1	0,17	-	-	1	0,17	8	1,38
1975	60,00	6	1,00	1	0,17	-	-	1	0,17	8	1,34
1976	64,14	3	0,47	2	0,31	-	-	-	-	5	0,78
1977	70,8	7	0,99	2	0,28	1	0,14	3	0,42	13	1,83
1978	74,55	3	0,4	4	0,53	1	0,13	-	-	8	1,06
1979	82,4	9	1,09	5	0,6	-	-	-	-	14	1,69
1980	84,15	2	0,24	2	0,24	2	0,24	-	-	6	0,72
1981	88,03	4	0,45	3	0,34	5	0,57	1	0,11	13	1,47
1982	93,44	2	0,21	10	1,07	-	-	1	0,11	13	1,39
1983	98,54	1	0,1	4	0,4	2	0,2	1	0,1	8	0,8
1984	106,05	5	0,47	6	0,56	5	0,47	1	0,09	17	1,59
1985	115,74	4	0,34	10	0,86	5	0,43	5	0,43	24	2,06
1986	114,87	1	0,09	6	0,52	10	0,87	4	0,35	21	1,83
1987	110	2	0,18	5	0,45	5	0,45	5	0,45	17	1,53
1988	110,13	6	0,54	7	0,63	-	-	4	0,36	17	1,53
1989	108,94	7	0,64	5	0,46	4	0,37	3	0,27	19	1,74
1990	113	-	-	4	0,35	9	0,79	4	0,35	17	1,49
1991	(113)+	1	0,09	12	1,06	5	0,44	6	0,53	24	2,12
1992++	(37,6)+	2	0,53	1	0,26	1	0,26	1	0,26	5	1,31

+ Estimated production

++ To end of April 1992 (provisional)

Figure 1 illustrates graphically the number of incidents per 10 megatons in the various categories as listed in Table 1. The figures for incidents per 10 megatons in Table 1 and Figure 1 show quite wide fluctuations from one year to the next. This is because the number of incidents is relatively small and thus one incident more or one incident less from year to year can cause a wide variation in incidents per 10 megatons. In order to smooth out these variations 3 year averages were calculated (2 year average for 1991/92) and listed in Table 2. Figure 2 displays the same information graphically.

Table 2 INCIDENTS PER 10 MEGATONS : 3 YEAR AVERAGE VALUES

Period	Incidents per 10 megatons				Total
	Spontaneous Combustion	Electrical	Flammable Gas	Other	
1970 - 1972	0,75	0,25	0,19	0,13	1,32
1973 - 1975	1,04	0,17	0,12	0,23	1,56
1976 - 1978	0,62	0,38	0,09	0,14	1,23
1979 - 1981	0,59	0,39	0,27	0,04	1,29
1982 - 1984	0,26	0,67	0,22	0,1	1,25
1985 - 1987	0,2	0,61	0,58	0,41	1,8
1988 - 1990	0,39	0,48	0,38	0,33	1,58
1991 - 1992	0,2	0,86	0,4	0,46	1,92

The frequency of spontaneous combustion was less in the 1980's than it was in the 1970's, and this trend seems to have been maintained so far in the 1990's. It is worth noting, however, that the two year period 1988/89 had the highest number of spontaneous combustion incidents for any two year period in the whole twenty three years. In terms of incidents per unit production the two year period 1973/74 had the highest frequency.

Electrical incidents have escalated somewhat since the late 1970's. This probably reflects the rising trend in mechanisation (See Figure 3). Flammable gas incidents have also escalated, particularly since the mid-1980's. Again this is probably a reflection of increased mechanisation. Fires due to 'other' causes increased rather suddenly in 1985 and the trend has not been reversed.

In making an appraisal of trends over this 23 year period it should be borne in mind that reporting procedures may not have been consistent and this could have distorted trends to some extent. It is not likely, however, that this would lead to any changes in practical conclusions. The overall impression is that the number of incidents of spontaneous combustion has been reduced, particularly since the early 1980's, but it still remains a significant potential hazard, while the number of flammable gas ignitions and fires other than those due to spontaneous combustion has increased over the same period. They also pose a significant potential hazard.

2 PREVIOUS GUIDELINES

An updated document entitled 'Colliery Fires', which is essentially a set of guidelines, was located in the COMRO library. In order to determine whether elements of that document should be incorporated into these guidelines, it was necessary to try and establish the date of the document, and to note changes which have taken place in the industry since the document was compiled. It appears to have originated around 1974 or 1975. Figure 3 shows the production make-up of the industry from 1975 to 1991. The main changes in this time are

INCIDENTS PER 10 MEGATONS

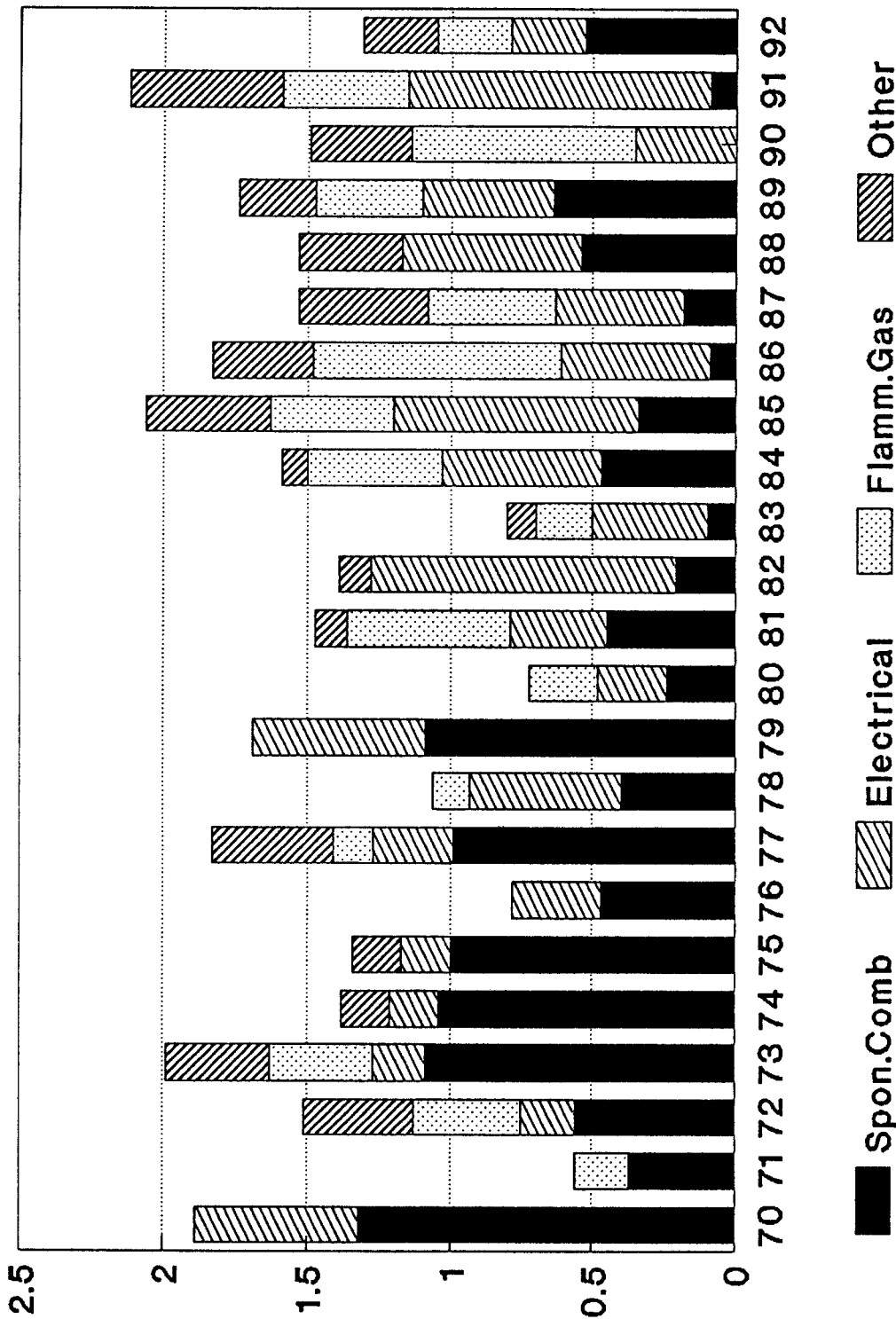


Figure 1

Figure 1 Incidents per 10 megatons

INCIDENTS PER 10 MEGATONS 3 YEAR AVERAGES

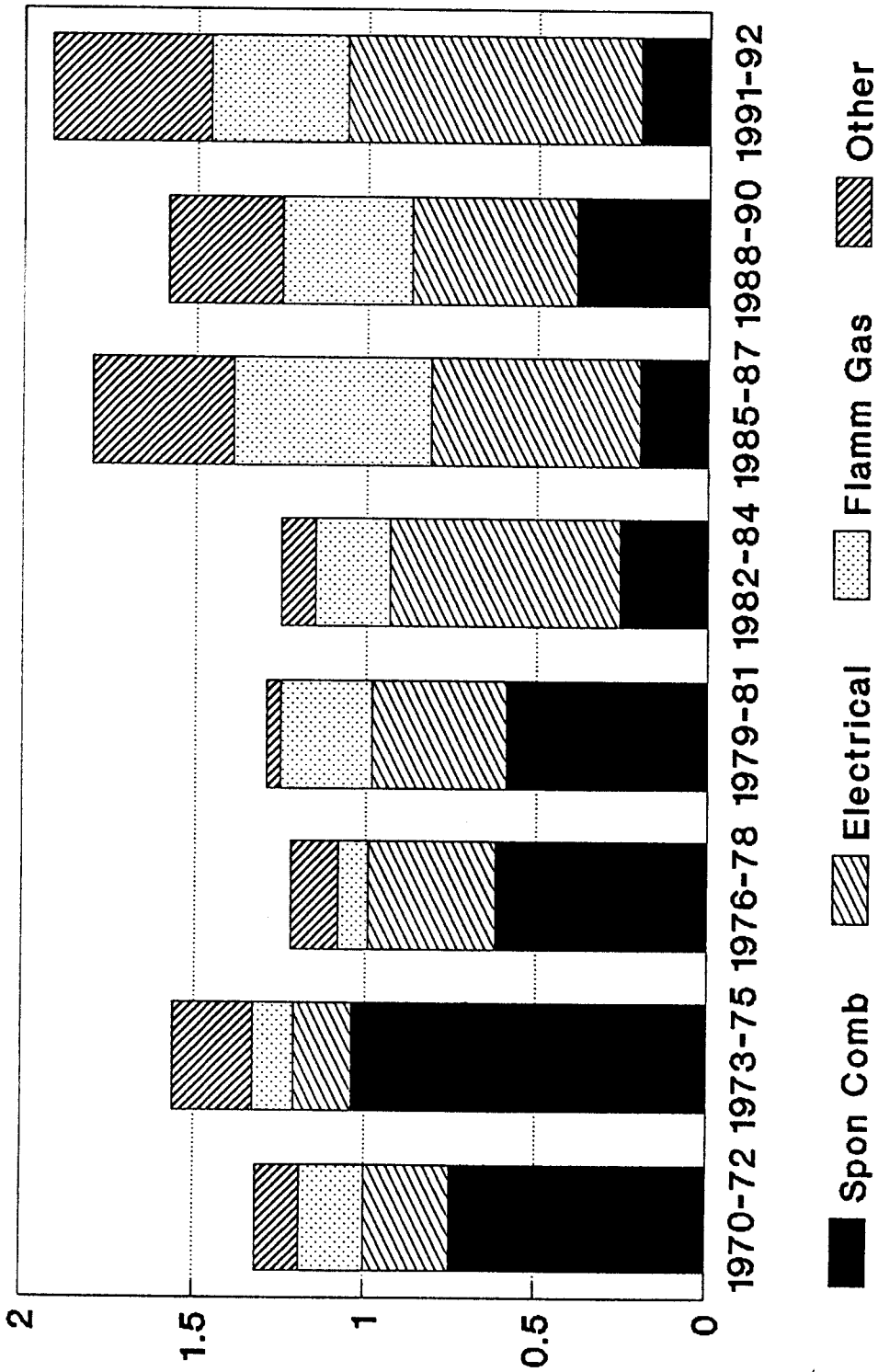


Figure 2

Figure 2 Incidents per 10 megatons (3 years averages)

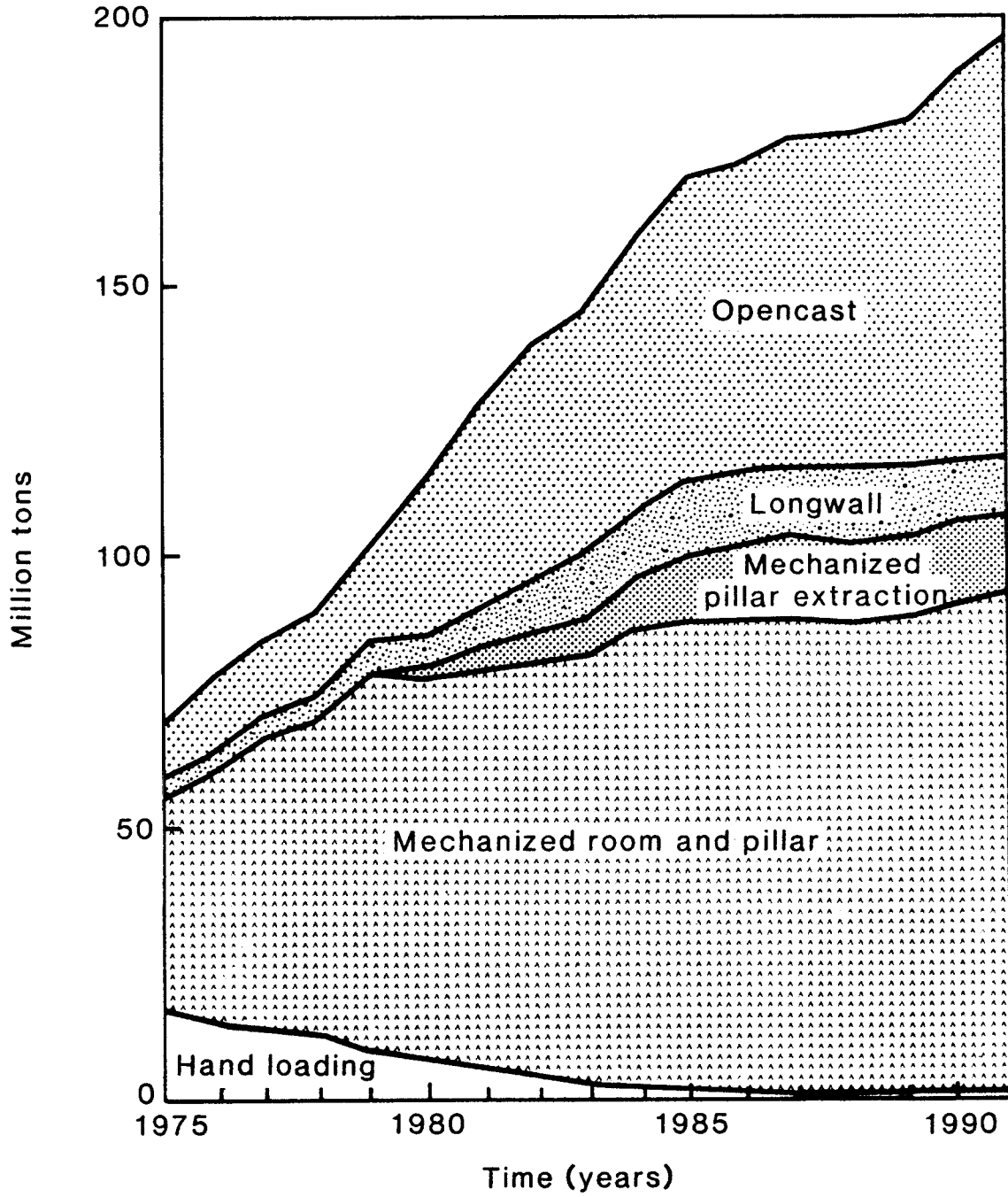


Figure 3 Showing the increase in mechanization which is reflected in the number of electrical and flammable gas incidents

the virtual demise of hand loading and the progression towards higher percentage extraction methods such as longwall and mechanized pillar extraction. One implication of the reduction in hand loading is a reduction in rope haulage systems and their replacement by conveyor systems. This could be expected to increase the risk of fires. On the other hand, mechanized mining lends itself to the establishment of discrete panels, which, in the context of underground fires, is advantageous compared with the less regular pattern of hand loading sections. The most significant effect of mechanized mining, however, is the increased tempo of mining. This means greater expenditure of energy per unit production, which in turn increases dust production and methane release, as well as creating conditions conducive to frictional ignitions. Increased expenditure of energy obviously means increased energy supply, and this makes for a greater risk of electrical fires. Another feature of mechanized mining is the strain placed on the maintenance of services such as ventilation, which makes dust and methane control more onerous. Special problems arise from the use of high percentage extraction methods such as longwall and pillar extraction, because of the possible influx of methane from seams other than the one being mined and the creation of goaf areas.

One would expect, therefore, that a set of guidelines compiled in the mid 1970's would have a different emphasis from one compiled today, and that in some technical respects it would be out of date. This is not to say, however, that these guidelines have no relevance today.

3 BREAKDOWN OF FIRES AND FLAMMABLE GAS INCIDENTS 1986 - 1992

The following statistics, Table 3, are based on fires and flammable gas incidents (mainly ignitions but some flammable gas fires) reported to the Department of Mineral and Energy Affairs in accordance with Regulation 25.6(c)(iv). This regulation requires the manager to report 'any fire or any indication or recrudescence of fire or spontaneous combustion at or in a mine or works or any explosion of gas or dust'. Cross checks have been made against fires and ignitions reported to the Rescue Training Services, and the two sets of figures do not match fully. Both sets of figures appear to underestimate the total number of incidents. Incidents listed in the RTS annual reports are of particular value because the number of rescue teams used is given. This is a reasonable indication of the severity of an incident.

Incidents which occurred in 1986 and 1987 had to be researched with the co-operation of the Chief Inspectors of Mines at Heidelberg, Witbank, Ermelo and Dundee, who organised searches of their archives for the individual incident reports. From 1988 onwards the reporting system was computerised and a print-out listing incidents and giving individual report numbers could be obtained from the statistician at the (then) Government Mining Engineer's Office. The numbers were then transmitted to the Chief Inspectors who produced the individual reports.

Table 3 FIRES AND FLAMMABLE GAS INCIDENTS
1986

Date	Colliery Number	Type of Incident	Reported Circumstances	Teams Used
Thursday 20 February	17	Electrical	Two men were burned while working on a cable which they thought was 'dead'.	NIL
Thursday 27 March	20	Electrical	Cable between circuit breaker and control panel burned due to short circuit.	NIL
Sunday 27 April	2	Spontaneous Combustion	Fire in return airway. Nitrogen used to extinguish it, amongst other measures.	137
Tuesday 29 April	13	Flammable Gas	Gas burned in the roof of the goaf. Found during early examination.	4
Monday 30 June	8	Other	Bearing of a conveyor bottom roller overheated, setting fire to coal on the floor.	NIL
Tuesday 24 June	2	Electrical	Water pump cable burned.	NIL
Friday 11 July	3	Other	Fire discovered in opencast. Thought to have been caused by blasting with diesel and ammonium nitrate.	1
Friday 19 September	21	Electrical	Transformer failure, causing smoke and fumes to be emitted.	NIL
Friday 14 November	14	Flammable Gas	Gas burning against the roof. Cause described as spontaneous combustion.	NIL
Not known	15	Flammable Gas	Ignition in the goaf. 1 man injured.	NIL
Not known	12	Other	Coal dust burned on shuttle car. Caused by overheated drum.	NIL
Not known	31	Other	Coalex explosive burned after blasting.	NIL
Not known	13	Flammable Gas	Gas burned in goaf. Work place evacuated. 9 injured.	NIL
Not known	16	Electrical	Control panel of cutter burned. Short circuit.	NIL
Not known	1	Flammable Gas	Gas ignited in the goaf behind a longwall face. Gang withdrawal.	NIL
Not known	1	Flammable Gas	Gas ignited in the goaf behind a longwall face. Gang withdrawal..	NIL
Not known	1	Flammable Gas	Gas ignited in the goaf behind a longwall face.	NIL
Not known	12	Flammable Gas	Gas ignited in the face of a continuous miner heading. The machine was not cutting.	NIL
Not known	13	Flammable Gas	Gas ignited in the face of a continuous miner heading while the machine was cutting.	NIL
Not known	1	Flammable Gas	Gas ignited in the goaf after a fall of waste.	NIL
Not known	12	Electrical	Fire in a transformer. Safety circuit bridged out.	NIL

Totals	Spontaneous combustion	1
	Electrical	6
	Flammable Gas	10
	Other	<u>4</u>
		<u>21</u>

Table 3 (Continued)
1987

Date	Colliery Number	Type of Incident	Reported Circumstances	Teams Used
Thursday 8 January	22	Electrical	Cable blew.	NIL
Friday 16 January	32	Other	Explosives magazine set alight.	2
Wednesday 8 April	9	Flammable Gas	Cause not known. + 2 Dead. 9 Injured.	2
Thursday 9 April	15	Flammable Gas	Cause not known. + 35 Dead. 5 Injured.	22
Thursday 14 May	9	Electrical	Short circuit.	1
Tuesday 9 June	8	Other	Overheated bearing on conveyor caused coal to glow.	NIL
Wednesday 29 July	23	Electrical	Short circuit on 6600 V feeder cable. 3 Injured.	NIL
Thursday 27 August	24	Other	Veld fire set fire to outcrop coal. Smoke spread into mine.	NIL
Saturday 12 September	3	Spontaneous Combustion	Fire started at intersection of old bord and pillar workings and opencast.	NIL
Wednesday 21 October	23	Electrical	Failure of 6,6 kV cable joint.	NIL
Wednesday 11 November	2	Other	Friction fire - conveyor.	NIL
Thursday 10 December	13	Electrical	Cable insulation burned after cable overheated.	1
Not known	1	Spontaneous Combustion	Broken coal smouldered.	NIL
Not known	16	Flammable Gas	Gas ignited during cutting at the face in a continuous miner section. 1 Man injured.	NIL
Not known	16	Flammable Gas	Gas ignited during cutting at the face in a continuous miner section.	NIL
Not known	1	Flammable Gas	Gas ignited during cutting at the face in a continuous miner section	NIL
Not known	13	Other	Gas burned in a coupling, causing a friction fire.	NIL

Totals	Spontaneous combustion	2
	Electrical	5
	Flammable Gas	5
	Other	<u>5</u>
		<u>17</u>

+ Cause not known when incident report was examined.

Table 3 (Continued)
1988

Date	Colliery Number	Type of Incident	Reported Circumstances	Teams Used
Friday 8 January	4	Spontaneous Combustion	Report not located.	-
Monday 8 February	5	Spontaneous Combustion	Report not located.	-
Saturday 5 March	17	Electrical	Feeder breaker being trammed connected to wrong (inadequate) cable - overheated.	NIL
Thursday 24 March	1	Spontaneous Combustion	Heaped up coal burned. Scaling on sidewall near dyke.	NIL
Monday 28 March	1	Other	Suspected gas ignition took place in goaf when roof fall occurred.	NIL
Thursday 31 March	1	Other	Fire in goaf.	NIL
Thursday 31 March	12	Electrical	Traction motor cable replaced. Earth leakage fault at cable gland entry ignited rubber (flammable) sheathing.	NIL
Friday 8 April	12	Other	Bottom return roller stuck in compacted duff. Friction fire developed. Ineffective inspection.	NIL
Wednesday 13 April	10	Electrical	Repairs made to power distribution box in sub-station. When main breaker was switched on a flash occurred. 1 Man injured.	NIL
Monday 9 May	1	Other	Bottom roller failed. Friction fire occurred - ineffective inspection.	NIL
Thursday 9 June	12	Electrical	Fault to earth occurred when isolator switch was re-energised. Faulty equipment.	NIL
Monday 13 June	6	Spontaneous Combustion	Heap of coal burned. Loaded out and dumped in water.	NIL
Tuesday 28 June	2	Spontaneous Combustion	Major fire. Two seams mined 14 m apart. Sealing programme plus nitrogen injection used.	12
Thursday 12 July	3	Electrical	Insulation failed on a trailing cable. Outer sleeve of cable joint burned.	NIL
Wednesday 10 August	24	Electrical	Power went off at sub-station. When it was restored a flash occurred at the flameproof section switches. Inadequate earthing.	NIL
Wednesday 26 October	34	Spontaneous Combustion	Accident report not located.	-
Tuesday 22 November	1	Electrical	Transformer caught fire. Suspected loose connection or phase to phase fault. Self rescuers used.	6

Totals	Spontaneous combustion	6
	Electrical	7
	Flammable Gas	0
	Other	<u>4</u>
		<u>17</u>

Table 3
1989 (Continued)

Date	Colliery Number	Type of Incident	Reported Circumstances	Teams Used
Saturday 7 January (RTS Saturday 7 January)	19	Flammable Gas	Roof bolt hole drilled near to a gas blower. When the rods were withdrawn and the bit reached the mouth of the hole, gas was ignited. Duff was ignited in turn.	2
Monday 16 January	1	Electrical	4 mm ² x 4 core lighting cable short circuited and PVC burned. The circuit was not equipped with earth leakage protection.	NIL
Saturday 21 January	7	Flammable Gas	Gas burned in cut during cutting operations. No explosion. No methane detected in the general atmosphere.	NIL
Wednesday 25 January	2	Spontaneous Combustion	Heating in return airway caused by fall of ground.	2
Saturday 29 January	8	Electrical	Underground supply isolated for work on surface sub-station. When power was restored earth leakage protection tripped twice. Cable joint blew, burning outer casing.	NIL
Tuesday 14 February	8	Spontaneous Combustion	Heating in fractured sidewall at air crossing site.	2
Friday 24 February	16	Electrical	Main contractor flashed, causing cable insulation to burn.	NIL
Wednesday 8 March	6	Spontaneous Combustion	Burnt coal discarded in back area. Coal heaps ignited when section moved on.	2
Tuesday 28 March	33	Other	Explosives burned in hole due to failed detonation. Blasting correct.	NIL
Tuesday 25 April	7	Spontaneous Combustion	Fire in sealed section. Fire seals built and water pumped in. Ambiguous statements about one man being injured.	4
Friday 26 May	1	Other	Cutter set on fire under right hand guide. Possible short circuit between phases.	NIL
Thursday 6 July	15	Flammable Gas	Continuous miner picks cut against sandstone roof, igniting gas. Evidence of inadequate ventilation. Methane recorder out of order.	NIL
Saturday 29 July	10	Spontaneous Combustion	Roof fall occurred then fire broke out in right hand barrier road. Fan attendant noticed smoke.	9
Tuesday 19 September	25	Electrical	Overloading of 500 V cable. Faulty joint overheated. 70 mm ² core cables to be replaced by 95 mm ² core cables. Self rescuers donned.	NIL
Thursday 5 October	14	Other	Friction between idler and belt set fire to fine coal, waste paper and timber cribbing. Potentially very dangerous.	NIL
Tuesday 31 October	17	Flammable Gas	Cutting in burnt coal with dyke stringers. Gas ignited in the cut.	NIL
Tuesday 28 November	1	Spontaneous Combustion	Heaped up coal ignited. Material dumped in water.	1
Wednesday 29 November	9	Spontaneous Combustion	One cubic metre of fine coal smouldered.	NIL
Friday 29 December	13	Electrical	Roof fell onto a feeder cable near a joint, causing a fire. Circuit breaker tripped.	1

Totals	Spontaneous combustion	7
	Electrical	5
	Flammable Gas	4
	Other	3
		<u>19</u>

Table 3 (Continued)
1990

Date	Colliery Number	Type of Incident	Reported Circumstances	Teams Used
Wednesday 24 January	13	Flammable Gas	Cutter had cut 8 m and was retracting. Gas was ignited. Suspected that a methane hole had been intersected. 3 men injured.	NIL
Saturday 17 February	12	Flammable Gas	Continuous miner intersected stone dyke. Burnt coal fell out and gas was ignited.	NIL
Tuesday 27 February	15	Flammable Gas	Gas burned after blasting. Suspected short circuit on blasting cable or burning of explosives after incomplete detonation.	11
Friday 1 June	6	Other	Compressor overheated and lubricant ignited when fusible plug blew. Thermostat and temperature cut out failed to operate. Better inspections instituted.	NIL
Monday 25 June	15	Flammable Gas	Coal and stone being removed from a continuous miner on which roof had fallen. 10 men injured. Cause not known.	NIL
Monday 9 July	1	Flammable Gas	Continuous miner was pulling out of face when roof fell on cutting boom and ignited methane.	NIL
Friday 20 July	26	Electrical	Cable feeding a fan blew setting fire to insulation. Regulation 10.3.2 complied with.	NIL
Sunday 29 July	19	Flammable Gas	Sealed, abandoned workings. Gas ignited by lightning.	1
Thursday 9 August	1	Flammable Gas	Continuous miner cutting near roof in burnt coal. Roof fell on cutting boom and ignited gas.	1
Sunday 26 August	27	Other	Bearing collapsed at conveyor tail end. Hot metal fell onto loose coal which started to smoulder. Extinguished with hose.	NIL
Tuesday 28 August	16	Other	Cement block stuck against conveyor belt. Frictional heat caused coal to smoulder - ineffective inspection.	NIL
Thursday 13 September	18	Flammable Gas	Continuous miner sumped in, moved back to re-align then moved forward to cut cubby. Cut into sandstone band and ignited gas.	NIL
Thursday 20 September	18	Flammable Gas	Continuous miner moved forward, sweeping the floor. Picks sparked against sandstone, igniting gas.	NIL
Thursday 25 October	32	Other	Fire caused by blasting in opencast section spread underground. Major operation required to seal and flood the area.	1
Friday 23 November	27	Electrical	LHD battery cable caught in prop shaft. Short circuit occurred and cable sleeve burned. Cable not attached correctly.	NIL
Tuesday 27 November	28	Electrical	Earth fault in flameproof lamp caused lamp to burn. No earth leakage protection. Possible unsound design. Good internal enquiry.	NIL
Saturday 1 December	16	Electrical	Switchgear being tested 'victorkop' blew. One man injured. Charge laid for not following test procedure.	NIL

Totals	Spontaneous combustion	0
	Electrical	4
	Flammable Gas	9
	Other	<u>4</u>
		<u>17</u>

Table 3
1991 (Continued)

Date	Colliery Number	Type of Incident	Reported Circumstances	Teams Used
Friday 4 January	1	Electrical	Cable blew next to switchgear while repairs were being carried out to shuttle car supply.	NIL
Tuesday 8 January	12	Other	Compressor lost oil and started to overheat. Oil on the floor burned. Did not trip out owing to low oil or overheating.	NIL
Friday 11 January	9	Electrical	Short circuit in control circuit of switchgear. Fire in control transformer.	NIL
Monday 25 February	24	Electrical	Bank of switches being moved forward. Cable overheated due to short circuit. Burnt cable became detached setting fire to conveyor and trailing cable.	NIL
Wednesday 10 April	2	Spontaneous Combustion	Fire detected by smell. Coal pillar scaling. Fire put out by drilling a hole and injecting 70 tons of nitrogen, then sealing off.	2
Thursday 25 April	15	Electrical	Fire on coal cutter discovered on pre-shift examination. Electrician disconnected supply. Protection circuits in order. 31 m trailing cable (coiled on cutter) burned.	NIL
Thursday 9 May	7	Flammable Gas	Eickoff shearer cutting 10 m from tail gate. Drum cut into sandstone roof. Sparks ignited pocket of methane.	NIL
Wednesday 15 May	29	Electrical	Supply cable to traction motor trapped by coal between motor and main frame. The damaged cable caught fire when the machine was put into fast mode. Dry powder extinguishers used.	NIL
Monday 20 May	12	Other	Bearing cup catching on fluid coupling. Sparks ignited coal dust and oil surrounding input shaft bearing. Put out by fire extinguishers.	NIL
Friday 20 May	29	Other	Build up of duff under conveyor. Duff started to smoulder. Extinguished with water.	NIL
Friday 12 July	27	Other	Conveyor belt, timber and coal dust smouldering. Smoke entered main intake supplying section.	NIL
Monday 15 July	29	Electrical	Supply cable damaged by sharp edge of box. 2 Phases shorted to earth. Earth leakage faulty. Fire put out by fire extinguishers.	NIL
Thursday 25 July	12	Electrical	950 V cable lying against ribside sustained damage. Blew at weak spot. Tripped out at main switches. Put out by fire extinguishers. Comprehensive internal report.	NIL
Tuesday 13 August	2	Electrical	Transformer failed and started smoking. It was isolated and replaced.	NIL
Wednesday 21 August	15	Flammable Gas	Gas ignited in cut while a back holing was being cut. A dyke had been negotiated.	NIL
Monday 26 August	1	Flammable Gas	Methane ignited in burnt coal. Fire ensued. Jet fan not working due to electrical fault. All seals in position. Put out by rescue teams. One person injured.	2
Tuesday 8 October	27	Other	Compressor oil pipe burst. (Compressor seized.) Spilt oil ignited. Workers used M20 self rescuers to get to long duration sets. Put out by water and fire extinguishers.	1
Saturday 12 November	16	Flammable Gas	Gas ignition. Fire burned back to intersection. Almost finished cutting last road on right hand side. Not known whether cutting head was turning. Piece of side wall against cutting head CH ₄ had been found at 08:30 and fan moved up. No CH ₄ found at 08:40. Ignition at 08:50.	NIL
Wednesday 13 November	27	Other	Fine coal smouldering at tail end of section. Extinguished by water. Probably caused by friction.	NIL
Thursday 28 November	17	Electrical	Conveyor starter control transformer overloaded, causing a fire. Overload setting re-examined.	NIL
Monday 2 December	7	Electrical	Pump attendant reported loss of power. A fault had occurred on a transformer. Oil from an oil filled fused isolator spilled and set alight.	NIL
Tuesday 10 December	1	Flammable Gas	Gas ignition in a continuous miner section. One man killed. Nine treated for burns.	4
Wednesday 11 December	30	Electrical	Transformer caught fire. Oil and cables burned. All personnel evacuated. Five donned self-rescuers, 24 did not. Cables adjacent to sub-station were not fireproofed. Rescue team called out but did not go underground.	1
Thursday 19 December	7	Electrical	Workers replacing pipes saw a flash and then fumes. All 6 donned self-rescuers from a cache and went to refuge bay. Fire caused by a ruptured plug on a 6,6 kW power supply cable entering 1 000 kVA transformer.	1
Totals		Spontaneous combustion	1	
		Electrical	12	
		Flammable Gas	5	
		Other	6	
			<u>24</u>	

Table 3 (Continued)
1992

Date	Colliery Number	Type of Incident	Reported Circumstances	Teams Used
Thursday 2 January	7	Spontaneous Combustion	Suspected heating in worked out longwall section. Section sealed off.	NIL
Friday 24 January	12	Flammable Gas	Methane burned at the goaf edge behind a longwall face. Went out due to oxygen depletion.	NIL
Wednesday 4 March	11	Spontaneous Combustion	Small heating in burnt coal. Dug out.	NIL
Wednesday 15 April	24	Electrical	Short circuit occurred on a scoop battery causing a fire. Driver used 2 fire extinguishers before the fire went out. Thought to have burned itself out. A loose connection on the take off terminal was thought to have caused the short circuit. The fire lasted 10 minutes.	NIL
Saturday 25 April	29	Other	LHD caught alight. Could not be put out with fire extinguishers. Set fire to coal on conveyor and spread. Whole section sealed off by 32 seals. 45 people donned self-rescuers. 16 went to refuge bay. No fatalities or serious injuries.	

Totals	Spontaneous combustion	2
	Electrical	1
	Flammable Gas	1
	Other	<u>1</u>
		<u>5</u>

The willing co-operation of all concerned is gratefully acknowledged.

Individual mines are listed by code numbers. The names of mines can be made known to the companies concerned on request.

3.1 Summary of Information in Table 3

Spontaneous combustion

19 Incidents in total

1 incident required 137 teams and nitrogen inertisation

1 incident required 12 teams plus sealing and nitrogen inertisation

1 incident required 9 teams

1 incident required 4 teams plus sealing

4 incidents required 2 teams (one of these incidents used nitrogen)

1 incident required Nil teams

2 reports were not located

It seems that there were perhaps eight incidents of fairly major significance while the remainder were relatively innocuous.

Electrical

40 incidents in total

3 incidents caused injury (3 men, 2 men and 1 man respectively)

4 incidents led to the use of self-rescuers, and teams were used in 3 of these incidents

6 incidents required teams (6, 1, 1, 1, 1, and 1 respectively)

Flammable gas

34 incidents in total

3 incidents caused fatalities (2, 35 and 1 respectively)

9 incidents caused injuries (ranging from 1 to 10 injuries per incident)

8 incidents were in the goaf - the rest were in the face area

4 incidents involved burnt coal

1 incident was caused by blasting

1 incident was caused by lightning, no friction involved

7 incidents required the use of rescue teams (22, 11, 4, 4, 2, 2 and 1 respectively)

16 incidents involved coal face machinery

2 incidents were substantial explosions (sub judice)

Friction did play a part or could have played a part in all the incidents except the one due to blasting and the one caused by lightning.

Other

27 incidents in total

11 were associated with conveyors

1 was in an explosives magazine

1 was a veld fire

2 were due to blasting

2 were suspected ignitions in the goaf

2 started in opencast workings

3 were due to compressors overheating

4 were due to cutting machinery overheating

1 was due to an LHD overheating (section sealed off)

4 incidents required the use of rescue teams (1, 2, 1 and 1 respectively)

Table 4 REPORTED INCIDENTS MINE BY MINE 1986 - APRIL 1992

Colliery Number	Spontaneous Combustion	Electrical	Flammable Gas	Other	Total
1	3	3	9	4	19
2	4	2	0	1	7
3	1	1	0	1	3
4	1	0	0	0	1
5	1	0	0	0	1
6	2	0	0	1	3
7	2	2	2	0	6
8	1	1	0	2	4
9	1	2	1	0	4
10	1	1	0	0	2
11	1	0	0	0	1
12	0	4	3	4	11
13	0	2	4	1	7
14	0	0	1	1	2
15	0	1	6	0	7
16	0	3	3	1	7
17	0	3	1	0	4
18	0	0	2	0	2
19	0	0	2	0	2
20	0	1	0	0	1
21	0	1	0	0	1
22	0	1	0	0	1
23	0	2	0	0	2
24	0	3	0	1	4
25	0	1	0	0	1
26	0	1	0	0	1
27	0	1	0	4	5
28	0	1	0	0	1
29	0	2	0	2	4
30	0	1	0	0	1
31	0	0	0	1	1
32	0	0	0	2	2
33	0	0	0	1	1
34	1	0	0	0	1
Total	19	40	34	27	120

3.2 Discussion of Information in Table 4

Spontaneous combustion

Mine No. 2 accounted for 21 % of the incidents

Mines No. 1 and No. 2 accounted for 36,8 % of the incidents

Mines No. 1, 2, 6 and 7 accounted for 57,8 % of the incidents

12 mines accounted for all the incidents, and only 4 mines reported more than one incident

Electrical

Mine No. 12 accounted for 10 % of the incidents

Mines No. 1, 12, 16, 17 and 24 accounted for 40 % of the incidents

Mines No. 1, 12, 16, 17, 24, 2, 7, 9, 13, 23 and 29 accounted for 70 % of the incidents

23 mines accounted for all the incidents and only 11 mines reported more than one incident

Flammable gas

Mine No. 1 accounted for 26,4 % of the incidents

Mines No. 1 and 15 accounted for 44 % of the incidents

Mines No. 1, 15 and 13 accounted for 56 % of the incidents

Mines No. 1, 15, 13, 12 and 16 accounted for 73,5 % of the incidents

11 mines accounted for all of the incidents and only 8 mines reported more than one incident

Other

Mines No. 1, 12, and 27 accounted for 44 % of the incidents

15 mines accounted for all of the incidents and only 6 mines reported more than one incident

Total

Seven mines accounted for 53 % of the incidents

34 mines accounted for all of the incidents and 21 mines reported more than one incident

Electrical fires are generally more evenly spread between collieries than the other categories of fire. Spontaneous combustion is confined to a few specific mines and the reduction in spontaneous combustion incidents is partly due to some of these mines closing. Flammable gas incidents are not very widespread either. Incidents in the remaining ("other") categories appear not to be widespread, which is surprising. One would expect them to be more random because of the wide variety of possible causes. The low incidence, overall, of conveyor fires is in contrast to other countries. Literature from most mining countries indicates a marked predominance of conveyor fires. In any event, conveyor fires have such a high life threatening potential that they must always be taken seriously.

A further aspect worthy of comment is that, of the 55 underground coal mines, members of the RTS, only 21 mines have reported more than one fire or ignition in the past 6 - 7 years, and many, if not most, of the incidents were fairly easily dealt with. A major fire or explosion requiring a full and sustained emergency organisation is a comparatively rare occurrence in South African coal mining and many of the officials will have had little or no first hand experience of such an occurrence. This would seem to point to a need for periodic refresher training and practise in the setting up and operation of fire emergency organisational measures, control centres etc.

4 RECOMMENDED GUIDELINES

4.1 Introduction

The mining regulations relating to underground fires are quite wide ranging and it is true to say that very few fires, if any, would occur at a mine where all these regulations were complied with fully. Any fires which did occur would be dealt with promptly before they could escalate. All mines have codes of practice and/or standard procedures which specify what measures shall be taken, and by whom, to ensure compliance with the regulations. Fires occur, then, because people do not fulfil their obligations in respect of the codes of practice and the regulations, either by choice or neglect, or because they lack the knowledge.

Training and motivation are of the utmost importance in this regard and while training routines form an integral part of mining operations, and can readily be applied to fire prevention and fire control, motivation may not be so easy to impart, especially in an industry where most of the labour force will spend a working lifetime without experiencing an underground fire. This is a major challenge to management.

4.2 Prevention

4.2.1 Spontaneous combustion

In order to plan measures for the prevention of spontaneous combustion it is first necessary to understand how the process starts and propagates.

Spontaneous combustion of coal can be defined as the process whereby coal and oxygen react spontaneously under ambient conditions. The result of this exothermic reaction is the generation of gaseous products such as carbon monoxide and carbon dioxide together with hydrogen, ethylene and propylene, and the heating of the coal itself. If the heat produced by this reaction cannot be dissipated faster than it is produced then the temperature of the coal rises. The temperature rise has the effect of accelerating the rate of reaction since this rate exhibits an exponential dependence on temperature. Thus, once the oxidation process begins, it propagates at an ever increasing rate (assuming that conditions in both the solid and gaseous phase remain conducive to the reaction) until first a hotspot then a fire develops.

Research is still proceeding to establish the exact mechanism of the oxidation process, but it is generally thought to occur in four overlapping steps (Falcon, 1985). These steps are adsorption/chemisorption, adsorption-complex decomposition, oxycoal formation, and combustion.

- (i) **Adsorption/chemisorption.** During this stage oxygen is adsorbed onto the surface of the coal and forms relatively unstable adsorption complexes which may then decompose or react further. The formation of these adsorption complexes requires a very low activation energy (3 - 4 kcal/mole) and occurs predominately at temperatures up to 70°C, although it may also continue occurring for the duration of the oxidation process. No carbon monoxide is evolved during this stage.
- (ii) **Adsorption-complex decomposition.** During this stage decomposition of the adsorption complexes occurs and the weight of the coal decreases. Again a relatively low activation energy (about 6,5 kcal/mole) is required. The decomposition occurs at temperatures between 80°C and 150°C, although some sources (SIMTARS, 1989) suggest that it may occur at temperatures as low as 35 - 45°C. The decomposition of these complexes is oxygen independent and yields mainly carbon monoxide.
- (iii) **Oxycoal formation.** During this stage chemical reactions occur which lead from the unstable adsorption/chemisorption complexes to the formation of stable oxygen-carbon complexes called oxycoal. A high activation energy in the region of 16 kcal/mole is required for these reactions to proceed. Typical reaction temperatures vary between 150°C and 230°C. These high temperatures are maintained by the high exothermicity of the oxycoal reactions. Carbon dioxide (from the decarboxylation of the carboxyl groups in the oxycoal) is the dominant reaction product during this stage.
- (iv) **Combustion.** During this stage the formation of oxycoal ceases and coal combustion commences. Temperatures are typically in excess of 230°C in order to satisfy the high activation energy requirement of 20 kcal/mole. When combustion of the coal occurs there is soot formation and a rapid decrease in weight.

Spontaneous combustion and/or heating of coal can, in theory, occur wherever coal and oxygen come into contact. It can develop in oxygen concentrations around 10 % and active heatings may be sustained at levels as low as 6 % O₂. Smouldering combustion can be maintained at an oxygen level of 2 %. Situations where sufficient oxygen is present for the onset of heating or spontaneous combustion would thus be expected to occur frequently during the various operations involved from the initial mining of the coal to its final consumption by the end user. Spontaneous combustion does, indeed, take place in underground workings, disused mines, ships' cargo holds and mine dumps.

The factors affecting susceptibility of a coal seam to spontaneous combustion can be classified into seam factors, geological factors and mining factors. These factors overlap to some extent.

(i) Seam factors

These are intrinsic to the particular seam and are natural phenomena over which management has little control.

(a) Coal rank

Broadly speaking rank is a function of the ratio of carbon to oxygen and volatiles in the coal. This is dependent on the character of the original plant debris from which the coal was formed and the amount of change it has undergone during the period of formation. The higher the rank (anthracite) the slower the rate of oxidation and the lower the risk of spontaneous combustion. The lower the rank (lignite) the higher the rate of oxidation and the greater the risk of spontaneous combustion. These are, however, generalisations.

(b) Petrographic composition

There is some controversy concerning the relative importance of the various petrographic constituents in promoting the spontaneous combustion of coal. The following order, from highest to lowest susceptibility, seems to find a lot of support: vitrain, clarain, durain and fusain. Quantification of the constituents is useful in determining risk liability.

(c) Temperature

The rate of oxidation has an exponential dependence on temperature. This is perhaps not so significant in determining the onset of spontaneous combustion, but once oxidation has commenced the rate of reaction doubles for every 10°C rise in temperature.

(d) Particle size

The particle size of a given quantity of material is important as it relates directly to the surface area available for reaction. As far as coal is concerned the issue is complicated by the existence of a complex network of pores.

These pores have the effect of greatly increasing the surface area. The external surface is usually regarded as a macro-structure, while the internal surface is regarded as a micro-structure. Oxidation begins at active sites on the macro-structure of the coal, which obviously increase in number as the particle size decreases. As the temperature rises so the sites increase in size. Temperature increase also increases the oxygen permeability of the micro-structure, encouraging active site formation. Small particle size also leads to relatively rapid desorption of methane, which then allows for the infusion of oxygen.

(e) **Metal sulphides**

Sulphides such as pyrites are oxidised and play a minor role in initiating spontaneous combustion. The heat released leads to increased oxidation, but the sulphides also swell when oxidised which causes disintegration of the coal, creating larger surface areas and more air pathways. It has been known since the beginning of mining that high sulphide coals were more prone to spontaneous combustion than low sulphide coals. The debate as to the exact mechanisms involved has never been concluded.

(f) **Physical properties**

Porosity, permeability, hardness, thermal conductivity and thermal capacity all affect the rate of oxidation of coal, and these properties have been included in mathematical models attempting to describe the spontaneous combustion phenomenon (Young, 1986 and Bryson, 1985). High porosity increases surface area, thus creating potential for the development of heating sites within the coal. High porosity is associated also with high inherent moisture content. Permeability is an indicator of the ease with which air can flow through the coal seam. High permeability therefore means a high likelihood of spontaneous combustion. The hardness of coal determines the ease with which the coal breaks into small pieces. Soft coal has a greater propensity to spontaneous combustion. A high thermal conductivity implies fast movement of heat away from hot spots and therefore less likelihood of spontaneous combustion developing. High thermal capacity implies a small temperature increase per unit of heat generated, hence a lower liability to spontaneous combustion.

(g) Strata movement

The crushing of coal under strata pressure has been considered as a contributor to spontaneous combustion. The heat produced is relatively unimportant. The increase in surface area is more important.

(h) Microbial activity

Microbes are present in coal mine water. One group of microbes feeds on sulphides in the coal to produce acid water, but this is not of great significance. Another group feeds on the mine timber and the microbial degradation can be accompanied by a rise in temperature, which could initiate spontaneous combustion in coal (or in the timber itself).

(ii) Geological factors

These are factors arising from the way in which the coal seams and associated strata are structured.

(a) Seam thickness

Coal from thick seams is not inherently more liable to spontaneous combustion than coal from thin seams. However, thick seams are more likely to contain bands of inferior coal or shale, which are prone to spontaneous combustion. In addition, mining methods in thick seams are often such that coal is left behind, usually in a fractured state. This situation carries a high risk of spontaneous combustion.

(b) Seam gradient

The mining of steep seams often results in coal bands being left behind, so that, other things being equal, steep seam mining could be expected to be more prone to spontaneous combustion than level seam mining. Steep seam mining may also lead to ventilation problems due to buoyancy effects. On the other hand, mining to the rise in steep seams leads to water accumulating in the goaf, and to high methane concentrations following closely behind the face. Both these factors tend to inhibit spontaneous combustion. Methane immediately behind the face is a mixed blessing, however, and mining to the dip is preferred from the point of view of methane control.

(c) Faulting

Faulting increases the risk of spontaneous combustion because of the presence of finely ground coal on the fault plane. Coal usually has to be left behind when mining through faults, which makes matters worse, while a fault can provide a ready path for air leakage. The latter point is exacerbated further if concentrated timber support has to be provided against the faults.

(d) Natural burnt coal and coal which has been oxidised

Previous oxidation or heating of coal has a variable effect on its propensity to spontaneous combustion. If the coal temperature rises to 30°C - 35°C and then cools, it is unlikely to reheat as its initial affinity for oxygen has been satisfied. If the coal heats to about 220°C - 400°C out of contact with air and then cools, it appears to be more oxidisable at low temperatures. If it heats to 500°C and cools its adsorption capacity at low temperatures is reduced. The bulk of the evidence suggests that coal which has been subjected to heating, whether in geological time as in the case of natural burnt coal, or in previous incidents brought about by mining, has an increased susceptibility to further heating.

(e) Outbursts

Outbursts of gas from a coal seam often result in the liberation of large quantities of powdered coal, which is highly prone to spontaneous combustion if not removed promptly. Usually it is removed promptly.

(f) Depth of cover

The depth of a coal seam has an indirect effect on the likelihood of spontaneous combustion in that deep seams usually have high methane concentrations and the presence of this methane inhibits the adsorption effects associated with the onset of spontaneous combustion. There is also a rough correlation between coal rank and depth (the deeper the seam the higher the rank).

Shallow seams, on the other hand, are susceptible to spontaneous combustion because of the presence of natural openings to surface, along which air can pass, which are opened up further when mining starts.

(g) Overlying and underlying seams

Air circulation can take place between the seam being mined and overlying or underlying seams, through cracks which develop because of mining. When seams are quite close together, the intervening strata are often more prone to spontaneous combustion than the seams themselves.

(h) Seam gases and water

The presence of significant quantities of methane will have the effect of retarding the early development of spontaneous combustion since the pores, cracks etc. will be filled with methane, and oxygen will not be able to penetrate. Significant quantities of water will have a similar effect. Once mining starts, the mining factor (see later) will nullify these effects.

(iii) Mining factors

These are factors which arise as a consequence of the methods of extraction. They are the only factors over which mine management has complete control. Thus, it is in this area where sound planning and disciplined mining can have the most marked effect on the risk and frequency of spontaneous combustion. Control measures are based on the prevention of contact between reactive coal or other strata and oxygen or moisture, or the provision of sufficient ventilation to limit the temperature rise of such reactive material.

(a) Oxygen

The rate of oxidation of coal is governed by the oxygen content of the air immediately surrounding the coal particle, while the total amount of oxidation which may occur is determined by the total available oxygen. At low levels of oxygen supply the rate of reaction increases dramatically as a result of a small increase in oxygen supply. However, an already active heating with a high availability of oxygen will not be slowed appreciably until a substantial reduction in oxygen availability is achieved. This has been confirmed in many actual cases. A very minor increase in ventilation can start a heating. A very major decrease in ventilation is required to stop one.

(b) Moisture and humidity

Water may come into contact with coal by two mechanisms i.e. surface wetting or surface adsorption. From a spontaneous combustion point of view wetting is unimportant since the heat of wetting is relatively small, hence the effect is more likely to be one of cooling, as a result of water temperature, than heating. Adsorption, which occurs when the quantity of water initially present is less than the equilibrium quantity for the vapour pressure of the air, presents a more serious problem. The heat of adsorption is very high and could raise the temperature of the coal to a level where the rate of oxidation is greatly increased. Thus, air with a high vapour pressure (humid air) flowing through broken coal may increase the risk of spontaneous combustion. Air with a low vapour pressure flowing through fractured coal with adsorbed water will lead to vaporisation, which will cool the coal. There is some evidence that, on the Highveld, the incidence of spontaneous combustion can be related to the onset of the summer rainfall season.

(c) Pillar design

Pillar design is based on the principle that an unconfined pillar of coal will accept the load of overlying strata both by yielding elastically and developing zones of failed material around its perimeter. Yield will continue until load distribution is complete. When yield is complete there should be a substantial core of confined, high strength material remaining. If the yield zones of the pillar are too large, apart from making pillar failure more likely, this will increase the likelihood of spontaneous combustion by providing paths for the slow seepage of air through fractured coal.

(d) Ventilation pressures and flows

No matter what method of mining is used, the ventilation flow and pressure distribution impose a major influence on the development and control of heatings. A pressure difference across a fractured zone can induce a flow of air likely to be in the optimum range for the development of a heating. An even more critical situation arises where a section containing a heating has been sealed off to try and extinguish the heating. Smouldering can be sustained at an oxygen level of 2 %, so it is essential to eliminate virtually all ingress of air into the section. This means reducing the pressure difference across the section to zero while maintaining pressure differences around the mine for normal ventilation. This implies a detailed quantitative and up to

date knowledge of the mine's ventilation pressure and flow distribution, which of course is essential in combating all types of underground fires.

(e) **Practical preventive measures**

There are some preventive measures which are common to all mining methods and others which are specific to one particular mining method.

- **Bord and pillar**

Heatings in operational bord and pillar sections (as opposed to mined out sections) invariably start in places which are accessible, and should therefore never become major fires.

The heatings often start in piles of broken coal either from spillage, falls of roof containing coal or sloughing of pillars. They can also start where top coal is left unmined.

Wherever possible, coal should not be left, and where this is unavoidable, vigilance is required to spot the first sign of heating. If such a heating should start it should be dug out so as to leave only completely solid coal. Piles of broken coal should be removed from the mine and where such piles are prevalent the removal must be done systematically, possibly by mechanical means. A typical pile of broken coal underground is particularly vulnerable because it contains pieces of mixed sizes. Small coal (5 mm diameter and less) possesses a large surface area for oxidation, but a pile consisting only of small coal would have few voids and would therefore not be penetrated easily by air. Conversely, a pile of large coal pieces provides easy access to air, so that temperature rise is inhibited. A pile of mixed sizes contains sufficient small coal to present a large surface area, and sufficient large pieces to create voids for the passage of air.

Where a pile of coal cannot be removed straight away, it should be spread out to an even depth to avoid creating hot air convection "chimneys" at high points.

- **Mined out sections**

Once a section is mined out, deterioration sets in and this usually means that falls take place. The scaling of pillars increases and there is generally more broken coal lying around than in a working section. The situation is aggravated further if roof coal has been left unmined. In those circumstances the section must either remain fully ventilated or it must be sealed off. If the section is half ventilated, this could lead to air currents in the critical range most likely to promote spontaneous combustion, not to mention accumulations of explosive gas mixtures.

The best practice is to seal off mined out sections. The development of discrete panels, with as few entries as practicable, lends itself to quick and efficient sealing, more so if the seals are partially prepared from the time that mining commences. It is also essential that the sites of the seals are supported as well as possible so as to minimise the development of cracks and fissures.

- **Longwall mining**

Advancing

Advancing longwall faces have a special problem in that there is a ventilation pressure difference across the goaf, which will promote leakage. This has the potential for causing spontaneous combustion and special sealing measures have to be taken to minimise the leakage. Since advancing longwall mining is not practised in South Africa, no further comment will be made on this method.

Retreating

The goaf of a retreating longwall face may or may not have a pressure difference across it. If there is a gas bleeder road through the goaf to the face start line then there is a potential for leakage just as in advancing longwall mining, and here the need to control the methane fringe at the goaf edge may be in conflict with the need to prevent spontaneous combustion. Even where there is no gas bleeder road and the goaf is completely 'dead', there is evidence that minor ventilation currents can penetrate the goaf right back to the

face start line. This effect can be reduced by building seals at intervals in the abandoned gate roads behind the face. A localised gas bleeder system can be used to control the goaf edge methane, with only a limited increase in risk of spontaneous combustion, by adopting the system shown in Figure 4. Since the air path through the goaf is short and 'renewable', a particular 'block' of strata is not exposed to the air path long enough for spontaneous combustion to develop. Another practice, when heatings occur in the goaf, is to increase the mining tempo to promote the rapid formation of extinctive atmospheres in the goaf.

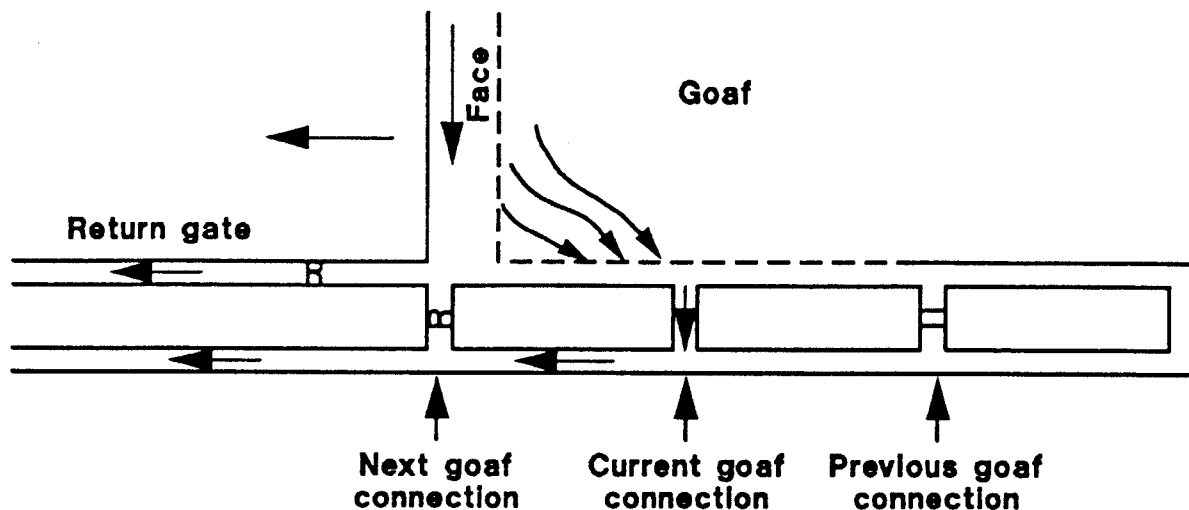


Figure 4 Controlling the airflow through the goaf for methane and spontaneous combustion purposes

The cardinal rules to follow are:

- Leave the minimum quantity of coal in the goaf.
- Install and maintain an effective continuous monitoring system to follow trends in carbon monoxide levels, flammable gas levels and airflows.
- Prepare stopping sites in advance, preferably with partially built stoppings at these sites and sufficient materials to complete them rapidly. It is also important to provide extra support at the stopping sites to minimise the development of ground fractures.
- Mine rapidly and uniformly (do not allow stop/start mining).

In general terms, longwall mining, particularly retreat mining, should be more favourable than bord and pillar mining in that the ventilation circuits are simpler and easier to control, so stray currents are less likely to occur. There are also fewer points to monitor for the early detection of spontaneous combustion. Techniques such as pressure balancing (see Appendix III) are also easier to apply.

Statistically the increase in longwall mining relative to bord and pillar, starting around 1980, has not coincided with an industry-wide increase in the frequency of spontaneous combustion, rather the opposite.

- **Pillar extraction**

Pillar extraction ventilation systems require frequent changes and can lead to slow moving stray currents. It is also difficult to ensure that no coal is left in the goaf both from pillar remnants and roof coal which falls when caving takes place. It is essential that mining proceeds as rapidly as possible, leaving virtually no pillar remnants, and that roof control is of the highest order, minimising the occurrence of unscheduled roof falls and the crushing of pillars. All loose coal which is accessible must be removed from the mine promptly.

Gas bleeder roads are often necessary to control flammable gas in the goaf, and this can provide conditions conducive to spontaneous combustion. However, monitoring the carbon monoxide content of the gas bleeder system should provide very early indications of heatings. Should the carbon monoxide content increase, indicating a possible heating, the gas bleeder may have to be discontinued. Other means of controlling flammable gas would then have to be found. On the other hand, increasing the flow in the gas bleeder road might introduce a greater cooling effect and stop the development of the heating.

As a general comment, a particular block of strata, in its unmined state is not subject to any influences which might cause it to oxidise and heat up. As mining proceeds these influences commence and reach a peak. When the face has bypassed the block of strata, it is then part of the goaf, exposed to an extinctive atmosphere, and the cycle is complete. The quicker the rate of mining, the shorter the cycle will be and

The less chance there will be of that particular block of strata, or any other, heating spontaneously.

4.2.2 Electrical

Of the electrical fires reported in the period 1986 - 1992, a few were caused by disregard for operating procedures such as failing to ensure that circuits were isolated properly when they should have been, or bridging out protection devices. At least one was due to human error, when the wrong cable, of inadequate capacity, was connected to an item of machinery.

Most of the fires were caused by the failure of equipment or systems, and could be ascribed to faults in system design, installation, inspection or maintenance, or simply to mechanical damage.

Regulation 21.1.1.1 is all embracing, and requires that 'All electrical apparatus should be selected, installed, worked and maintained in such a manner as not to constitute a hazard and shall be placed and protected in such a manner that no person can be injured by inadvertent contact with any live portion thereof'. Subsequent regulations elaborate in considerable and specific detail as to how compliance with Regulation 21.1.1.1 is to be achieved.

Consideration of the incidents which occurred during the years 1986 - 1992, and a perusal of fire incident surveys which have been carried out in the past, seem to indicate that 'history repeats itself'. The comments which follow are, in some measure, repetitions of comments which have appeared in previous documents.

Electrical apparatus should be adequate to deal with any load likely to be met. It is important that the loading factors be reviewed periodically. Increases in loading should be catered for so that overloading does not take place, and if there is a decrease in loading, the setting of overload protection should be adjusted accordingly.

Maintenance schedules must be designed to replace contaminated transformer and circuit breaker oils, and coal dust must be systematically removed from motors and other apparatus.

Cables should be installed so as to minimise the risk of damage by moving machinery or roof falls. Protection devices do not always function effectively when such damage causes inter-phase short circuits.

When a shift ends and an idle period follows, power must be cut off from machines and trailing cables by switching off circuit breakers and disconnecting trailing cables.

Responsibility for this must be allocated to specific persons.

Collieries are vulnerable at times of general power cuts or power failure. Under-voltage relays do not always trip circuit breakers and unattended circuits become live when power is restored.

- **Typical procedures in use at collieries for the prevention of electrical fires**

(i) **General**

Electrical equipment must be well ventilated and kept free of coal dust and oil.

(ii) **Shafts and adits**

The area around shafts and adits must be kept clean and free of accumulations of flammable materials.

Oil-filled transformers and switchgear sited in the vicinity of shafts, adits and cable tunnels must be mounted over a bunded compartment filled with 50 mm crushed stone. The volume of the bund must be four times the volume of oil. Should it be considered possible that in the event of a transformer explosion, burning oil could be blown into the shaft/adit, a 220 mm blast wall of a height at least 200 mm above the mounted height of the transformer must be provided between the transformer and the shaft/adit.

Only dry transformers and switchgear should be purchased in future for use in sub-stations at shaft tops. Flammable liquids must not be stored in the immediate vicinity of shafts and reasonable precautions must be taken to prevent flammable liquids from running into and down a shaft/adit.

(iii) **Surface cable or pipe tunnels into shafts**

Cable/pipe tunnels must be sealed from the shaft by means of a fireproof wall or door. A breather pipe and drain pipe can be installed through the seal. In cable tunnels where a fire would have serious consequences the installation of a fire detector and automatic sprinkler system can be considered. There must be no possibility of oil or other flammable liquids draining to the tunnel.

(iv) Underground sub-stations

Only dry type transformers and switchgear should be purchased for future use underground. All sub-stations shall be kept clean and free of accumulations of flammable materials. Sub-stations must be adequately ventilated. Where fans and filters are used to force ventilate a sub-station, the dust filter bags must be cleaned regularly. All sub-stations must be provided with two class C fire extinguishers mounted at the sub-station on the intake side.

The above general precautions apply to flameproof mobile sub-stations, small oil-free sub-stations (not more than 4 HT switches or 800 kVA transformer capacity) and mobile sectional sub-stations with oil-filled switches and/or transformers.

(a) Large oil-free sub-station (> 800 kVA transformers or > 4 HT switches)

Sub-stations shall have concrete floors and brick end walls. The interior shall be painted or white washed. Where possible sub-stations shall have main and emergency steel plate doors. Where there is a ceiling the space above the ceiling must be ventilated. Cables in trenches shall be covered in sand. Cable entries into the sub-station shall be sealed with fire proof materials. The exhaust side door or wall shall be provided with an adequate opening to allow ventilation. The inlet door shall be provided with an opening at head height to allow inspection and inlet air except where the sub-station is provided with a fan and filter. The opening should be just big enough to allow ventilation. Where possible the sub-stations should exhaust directly or via a duct into a return airway. Where the sub-station cannot exhaust into a return airway the ventilation openings at both the inlet and outlet ends should be provided with flaps which can be closed in the event of a fire. Automatic fire extinguisher systems of the gas type may be considered for large installations or installations in main intake airways.

(b) Sub-stations with oil-filled equipment (excluding mobile section sub-stations)

The provisions relating to large oil-free sub-stations (above) apply.

In addition:

Oil-filled transformers must be mounted over a bunded area of four times the transformer oil volume, filled with 50 mm crushed stone. A sill must be built across the door openings to prevent oil running out of the sub-station. Exposed coal surfaces should be protected preferably by walling, guniting or plastering. Two class C fire extinguishers and two sand buckets must be placed at the sub-station on the intake side. Where practicable a water point and a hose of at least 25 mm diameter should be placed at a site convenient to the sub-station. Oil-filled transformer installations situated in main intake airways and which are not ventilated to return must be equipped with automatic high velocity water spray fire fighting systems. It is recommended that oil-filled switchgear and transformers used underground be checked on a quarterly basis for hot spots using infra-red scanning equipment.

- (c) Small single oil-filled transformers (25 kVA or less) or single oil circuit breakers

The transformer/switch shall be sited away from the coal pillar and be mounted over a bunded area filled with 50 mm crushed stone, the capacity of the bund to be four times the volume of oil contained in the transformer/switch. One class C fire extinguisher must be placed on the intake side of the site.

- (v) Battery charging stations

Battery bays shall be so constructed that the intake air entering the battery bay is vented to a return airway and not permitted to continue as intake air to a working place. The floor of the bay shall be impervious and constructed to allow regular wash down. A 25 mm diameter water pipe with hose must be provided within 50 m of any battery in the bay. Any oil-filled equipment must be mounted over a bunded area of adequate capacity, filled with 50 mm stone. Two class C fire extinguishers must be provided on the intake side of the bay.

(vi) Cables

Only flame retardant cables and joints constructed to an approved specification shall be purchased for future use underground. Cables shall, as far as possible, be suspended and protected from damage. Cables shall be neatly run and not bunched. Where large numbers of cables are run they should be run on racks. Where practical, multiple layers of cables in racks should be avoided. Where large numbers of cables are run in intake airways for any appreciable distance, fire barriers should be provided at intervals. Fire barriers in cable racks in intake airways should be at least 2 m long and at intervals of not more than 100 m. Jute served cables should have a fire break of 2 m every 100 m. The jute serving should be stripped or fire barriers installed. Where possible cables should not be installed within 300 mm of timber. A fire barrier can take the form of a sand-filled box, a coating of vermiculite or other fire resistant plaster or intumescent paint.

4.2.3 Flammable gas

Before setting out preventive measures it is worth reflecting on the types of incidents which have taken place since 1986, and which it is hoped can be prevented in the future.

Of 34 incidents some 16 were directly involved with coal face machinery, while 8 took place in the goaf (7 in 1986, no more until 1992). This makes 24 in which the ignition source was almost certain to have been friction. Of the remaining 10, two were sub judice at the time of compiling the statistics, one was caused by blasting and another was ignited by lightning. In fact the last two are probably the only incidents which were not caused by friction in one form or another.

Another statistical survey of the South African coal mining industry reveals that, from 1980 to 1989 there were 55 reported ignitions. In 49 of the incidents details of the ignition sources were known, and the distribution of ignition sources was shown to be:

Continuous miner picks	- 43 %
Coal cutter picks	- 29 %
Electricity	- 11 %
Explosives	- 11 %
Unknown	- 6 %

The unknown category will include 3 of the goaf incidents reported in the present survey, and since these are likely to have been caused by friction of rock on rock, this increases the

proportion of incidents caused by friction to around 75 %. Frictional ignitions are also a major source of concern in other countries, notably in the United Kingdom and Australia.

Another cause for concern is the fact that hybrid mixtures of methane and coal dust have flammable characteristics which are different from those of methane alone, and in particular, that the lower limit of flammability of those mixtures is lower than that of methane. Since such hybrid mixtures are likely to occur close to the cutter picks, the question of what constitutes a non-flammable mixture needs to be addressed.

Research in South Africa has been described recently in a paper by Phillips and Landman. They assessed the flammability characteristics of methane and coal dust mixtures using volumetric sources of ignition (fairly widespread sources) which could be characterised by a blown out shot, and point sources of ignition, which could be characterised by cutter picks striking material such as pyrites. It was found that, in a coal dust concentration of 150 g/m^3 (typical of the concentration at the cutting drum of a continuous miner) ignition could be produced by chemical igniters (volumetric sources of ignition) with no methane present and by point sources of ignition at a methane concentration of 1,5 %. While the coals used in these tests were particularly reactive and perhaps not typical of South African coals in general, the statutory limit of 1,4 % flammable gas, which was probably decided on relative to a lower flammable limit for methane of 4 - 5 % in air, appears not to have such a wide margin of safety relative to hybrid mixtures.

South African collieries generally do not have heavy flammable gas emissions when compared with coal mines in many other countries. The presence of flammable gas can be intermittent rather than constant and this requires workers and officials to be especially vigilant in order to avoid being caught off guard. A further point to remember is that, wherever flammable gas occurs, in however small a quantity, it leaves the strata in a virtually pure state. It is subsequently diluted by ventilation to a very low concentration and it therefore follows that, between the point of emission and the point of final dilution, it always passes through the nominal explosive range of 5 % to 15 % in air. The control of flammable gas in the context of preventing ignitions is entirely concerned with controlling the position and extent of this explosive fringe, and the most effective agents in this control are air velocity and turbulence close to the point(s) of emission. Consideration should again be given to the use of controlled recirculation of air (and to some extent this is being done). Flammable gas ignitions are seldom caused, these days, by failure to supply sufficient ventilation to dilute the gas to below the lower flammability limits; but by inadequate air velocity and turbulence at the point of emission. Local controlled recirculation systems can often be the means of providing such velocity and turbulence without prejudicing the general ventilation system.

Regulation 10.8.1.1 requires the manager to provide sufficient ventilation continuously in all accessible workings so that the general body of the air does not contain more than 1,4 % flammable gas. Inaccessible workings, which are exempt from this proviso, are by definition, goaf areas and sealed off workings, and by implication, places such as the kerf, and the cutting zone of continuous miners and longwall shearers.

It follows then that inaccessible workings, defined or undefined, need to be treated specially and somewhat differently from accessible workings.

(i) Accessible workings

A well planned and well operated ventilation system is the single most important component in a strategy to control flammable gas and to prevent accumulations. Despite the maximum specified limit of 1,4 % flammable gas in the general body of the air, the system must be planned on a much lower limit in practice, so that the specified limit would only be reached in the event of a serious disruption of the system. Once the system has been put into operation with adequate fan capacity and a good airway network, frequent checks on the flow and distribution of air are vital. Minimisation of leakage and airway obstruction are essential, for economic as well as safety reasons. Short term and longer term mining plans must include detailed ventilation plans. In particular, provision must be made for the full ventilation of spare sections so that, if a section is stopped temporarily because of say, faulting, it can be ventilated fully in addition to its replacement section. The changeover of ventilation from a mined-out section onto its replacement section must be planned in detail as the system is very vulnerable to flammable gas accumulations at the time of such changeovers. There is ample statistical evidence for this. It is extremely dangerous for a section to be half ventilated at any time.

The pressure difference across every set of airlock doors in the mine should be measured and recorded, and trends in these readings should be followed closely. A wealth of information can be gleaned from such readings by an experienced ventilation specialist, especially in the case of South African collieries, where the workings are largely in a single plane and where there are no wide variations in temperature and elevation to complicate the interpretation of the readings. A very high proportion of the data normally gained from complex and time consuming pressure surveys can be obtained from a scrutiny of door pressures around the mine circuit.

(a) Statutory requirements

The regulations on the detection and control of flammable gas as a means of preventing accumulations are quite comprehensive, and are augmented by regulations governing the use of correct mining practices and equipment with a view to eliminating the risk of ignitions.

More specifically, the regulations call for the definition of hazardous areas, these being return airways, areas within 180 m of a working face, and any other area where there may be a risk of igniting gas, dust, vapour, or any other explosive material. In addition, work is prohibited where the flammable gas concentration exceeds 1,4 %.

The use of electricity is similarly prohibited, operators of electrically operated machines must test for flammable gas before the machine is taken into the working place and shall continuously monitor the atmosphere for flammable gas.

Every working place which has been idle for more than 6 hours, or in which blasting has taken place must be examined and made safe before persons are allowed to enter.

Where a section is giving off flammable gas freely a limit is placed on the number of work places under the charge of an individual miner, and inspections must be carried out at hourly intervals. Special arrangements are specified for testing where there is no shift boss.

There are specific requirements regarding testing for flammable gas before firing a charge, and stricter requirements where flammable gas has been detected in the past 24 hours.

The regulations specify where only permitted explosives may be used.

Charging and firing by electricity must stop during a thunderstorm.

All persons must be withdrawn from any part of the mine where flammable or noxious gas is present in dangerous concentrations, while the flammable gas concentration under normal working conditions where persons are required to work or travel shall not exceed 1,4 %.

The manager shall notify the regional director of the presence of flammable gas in any part of the mine, if it is detected for the first time, or if it has not been detected in the previous three months.

The manager must compile a code of practice, to be approved by the regional director, which shall set out fully the means to be adopted to comply with the provisions requiring him to dilute gases and dust within statutory limits. The code must also specify how worked out areas shall be sealed off and must detail the responsibilities of the shift boss relative to ventilation. If the regional director deems it expedient a continuous gas monitoring system must be installed, and details of this system must be included in the code.

Airflow measurements must be taken monthly and dust concentration measurements must be taken six monthly

Return airways must be fenced off from disused workings and must be examined at intervals not exceeding 14 days by a competent person and at intervals not exceeding three months by a shift boss.

If the ventilation in any part of the mine stops, weakens noticeably or reverses, electricity must be switched off and not restored until it is safe to do so.

The main fan must be on surface, have an automatic alarm which sounds if the fan stops, and must be supplied with power via a ring circuit.

If the main fan stops persons must be withdrawn from affected parts of the mine and the electricity switched off. Persons shall not return, nor the electricity be restored, until it is safe to do so.

No underground fan shall be started until thorough tests for flammable gas have been made in the vicinity of the fan, motor and accessories and the atmosphere found to contain not more than 1,4 % flammable gas.

Provisions are included relating to doors, regulators, stoppings and the use of non-flammable brattice.

Comprehensive procedures are required with a view to preventing coal dust explosions.

In fiery mines diesel engines used underground must be of a design and construction approved in writing by the Director General and operated under conditions and subject to such restrictions as he may specify. Diesel engines may not be operated if the flammable gas concentration at the place of operation exceeds 1,4 %, and the operation of the diesel engine shall stop if the ventilation current at the site of operation stops, weakens noticeably or reverses.

At least one flammable gas measuring instrument shall be provided for every 20 underground employees and sufficient flammable gas warning devices shall be provided to ensure that every operator of an electrically driven machine for cutting, drilling, breaking or loading can be issued with one. Such instruments shall be calibrated, maintained and tested according to a set procedure, and every person issued with an instrument shall be trained in its use.

A record must be kept of each person to whom either type of instrument has been issued and all instruments shall be returned to the point of issue at the end of each shift.

No light or lamp other than an enclosed and locked or sealed light or lamp of a design and construction approved by the Director General shall be allowed, except such stationary lights or lamps at such places as may be permitted in writing by the regional director.

No contraband is allowed underground and regulations provide for searches in order to ensure that contraband is not taken underground.

Welding, flame-cutting, grinding, vulcanising, soldering or similar equipment can only be housed or used in a workshop established with the permission of the regional director and subject to the conditions he may deem necessary. Further regulations provide for the proper operation of such workshops.

Electrical apparatus used in a hazardous area, as defined in Regulation 1(11A) shall be explosion protected and certified as such in a test report provided by an inspection authority approved by the Director General. Further regulations provide for the identification of such apparatus, the keeping of relevant test reports and identification numbers by the manager, and the forwarding of copies of test reports to the regional director. Repairs

and modifications carried out by non-licensed organisations would nullify the original approval.

Persons who operate, run and maintain explosion protected apparatus must be properly instructed in the conditions and requirements contained in appropriate specifications in accordance with which the apparatus was tested.

The foregoing is a brief summary of what is contained in the regulations. For a complete and exact account of statutory requirements it is necessary to study the regulations themselves.

The relevant regulation numbers are given below:

1(11A)(i)(ii)	8.10.13	10.9.3	10.25.5(d)
8.5.1	8.10.4	10.9.4	10.25.5(e)
8.5.2	9.2.9	10.11.1	15.4.2
8.6.1	9.3.2	10.11.2	15.4.3
8.6.2	10.3.1	10.13.2	15.5.2
8.6.3	10.6.5	10.13.3	15.9.1
8.7.1	10.6.6	10.14.1	15.9.2
8.7.2	10.6.8	10.14.2	15.10.2
8.9.1	10.8.1.1	10.14.3	to
8.9.2	10.8.1.2	10.15	15.10.9
8.9.4	10.8.2	10.16.1	21.17.1
8.9.6	10.8.3	to	21.17.2
8.9.8	10.8.4	10.17.4	21.17.3.1
8.9.9	10.8.5.1	10.24.1	21.17.3.2
8.9.10	10.8.5.2	to	21.17.4
8.10.12	10.8.6	10.24.11	21.17.5
			21.17.6

(b) Further comment on inspection

The regulations provide a framework for the prevention of flammable gas fires and ignitions, and this can generally be regarded as a minimum requirement. Testing for flammable gas as prescribed is generally confined to places where people are working and where mining operations are proceeding.

There is a need, in addition, to keep an eye on the mine wide flammable gas situation by taking flammable gas readings throughout the return airway system, as an increase in the flammable gas level in a return airway could be a sign of significant change taking place in the 'active' workings or in a mined out area. Return airway testing procedures can be written into the statutory code of practice required in accordance with Regulation 10.8.2.

The same regulation empowers the regional director to prescribe that a continuous monitoring system be instituted. Arrangements should also be in place for monitoring trends in the surface barometric pressure. The influence of barometric pressure changes on flammable gas emission has been well documented over the years, but the studies and publications of Dr C.J. Fauconnier, linking the occurrence of mine explosions to barometric pressure trends of longer duration than the usual diurnal changes, are of particular significance in the South African context.

A phenomenon which is perhaps not as well documented nor as well known is that of gas locking. Through ventilation circuits at higher elevations than the general workings will remain clear of flammable gas accumulations because of normal ventilation flow. If the ventilation is disrupted for any reason, flammable gas can accumulate, forming an inverted U-tube full of gas. On the restoration of normal ventilation, if the pressure difference across the circuit is insufficient to overcome the buoyancy of the accumulated flammable gas, the ventilation will remain blocked by the gas and the accumulation will not be removed. Officials carrying out inspections after ventilation stoppages should be aware of this possibility in circuits above the general elevation of the workings.

The prospect of flammable gas roof layers forming above slow moving ventilation currents has been publicised 'ad nauseam' over the years but it is worth further repetition. The presence of a roof layer may have little influence on general body flammable gas levels and an increase in the general body concentration may not always be an indication that there is a source of flammable gas at the point where the increase occurs. The increase may be due to the fact that a flammable gas layer has started to mix with the ventilation stream, probably at a point where the air velocity increases, due to a reduction in cross sectional area. The actual source of the flammable gas could be some distance away, at the point where the layer forms. Flammable gas measurements in the general body should always be accompanied by corresponding measurements at roof level.

A few years ago there were a few occurrences in the United Kingdom in which flammable gas accumulated beneath armoured face conveyors when they were stopped, and which ignited when the conveyors were started.

Such possibilities cannot be discounted, and it may be worthwhile to test for flammable gas occasionally beneath armoured face conveyors. The test method might pose practical problems, though.

Finally, there seems to be some correlation between flammable ignitions and the presence of burnt coal. Extra vigilance is necessary at places where the emission of flammable gas is to be expected. This could include places where there are geological disturbances such as faults and dykes, particularly where there is more than one seam present, and where there is evidence of burnt coal.

(ii) Inaccessible workings

(a) Sealed off workings

Regulation 10.8.1.2 defines accessible underground workings as 'underground workings of a coal mine which have not been sealed off in a manner prescribed in the code of practice referred to in Regulation 10.8.2 or which have not caved owing to longwall or pillar extraction mining methods'. Regulation 10.17.2 requires that 'all stoppings shall be robust and built in such a manner as to prevent leakage. At least one side of every stopping shall be kept accessible for inspection'. While this is intended to refer to stoppings between intake and return systems, it is equally true that stoppings used to seal off disused workings should be designed to prevent leakage and to control the passage of gases, and that the outside of such stoppings should be ventilated and accessible for inspection. Mined out sections which are merely fenced off to deny human access must be ventilated to the same standard as the 'live' workings.

The regulations do not prescribe that inaccessible parts of the mine shall be sealed by explosion-proof stoppings but this should be considered if there is a risk of an explosion in a mined out section. This risk could be present in a section with a moderate flammable gas emission and a high propensity to spontaneous combustion. Where flammable gas does accumulate in a sealed off area it is important that the explosive fringe, referred to earlier, should be outside the seals and not inside, so that it can be dissipated by the

ventilation system. It would be good practice to install sampling pipes through at least some of the seals in order to measure the normal build up of gas after sealing. This would provide valuable information which could be used to interpret abnormalities which might occur when a section in the same part of the mine is sealed off to control a fire.

As part of this routine sealing off procedure, any surface boreholes which intersect the sealed area should be blocked and borehole casings removed. Before final sealing all roadways in the vicinity of the seals should be stone dusted thoroughly.

No matter how robust the seals are (even if explosion-proof) a sealed off section will 'breathe' in and out under the influence of barometric pressure fluctuations.

For this reason the outside of the seals should be adjacent to the return airway system. This is particularly important where a working section runs alongside a sealed off section. A substantial barrier pillar should be left between the two.

(b) Goaf

- Longwall

It must always be assumed that there is flammable gas in the goaf of a longwall face, and it is generally easier to deal with when advance mining is practised as the gas bleeds naturally and 'measurably' into the return airway system, and methane drainage, if it should be necessary, is easier to apply. However, this is academic as retreat longwall mining is the preferred method in South Africa. The guiding principle in dealing with flammable gas is that the explosive fringe should not be close to likely ignition sources. It must be assumed that there are more ignition sources in the vicinity of the face than there are in the goaf itself, and therefore the explosive fringe should be some distance behind the face. The most likely ignition sources in the goaf itself would be friction of rock on rock and possibly lightning, via boreholes to surface. The latter source of ignition should be eliminated by sealing such boreholes and removing the borehole casings.

The risk of frictional ignition could possibly be assessed on the basis of rocktype. The presence of quartzitic rock in the strata above the seam would tend to increase the risk. The presence of coal seams above or below the seam being mined would also make matters worse as the goaf would then have its own source of flammable gas, in addition to any gas which migrated into the goaf from the seam being mined. Effective roof control, with regular and systematic caving, is also very important. If the goaf 'hangs up' and caving is spasmodic there is greater chance of open voids developing in which flammable gas can accumulate, to be ejected when caving does take place. Modern longwall roof support systems are usually effective.

As always, the essence of good flammable gas control is good ventilation, with face velocities high enough for some air to course along the back of the face supports, if there is space, thus tending to push back the explosive fringe. At the intake end of the face, however, the air should be controlled so that it does not sweep into the goaf at the expense of the face. If possible, tests for flammable gas should be carried out behind the face supports, using a methanometer and probe. The option of using a gas bleeder system through the goaf, to a return connection near the start point of the face, is a good one provided the flow through the system is always under complete control and provided there is no appreciable risk of spontaneous combustion. The required degree of control is often difficult to achieve. In the absence of a gas bleeder system any gas in the goaf will tend to migrate towards the point of lowest ventilation pressure, that is, the return end of the face. Since this zone can be one of high energy activity it is necessary to direct the gas away from the face. If a localised gas bleeder system, as shown in Figure 4, is used the 'current goaf connection' will become the point of lowest pressure. Goaf gas, and some of the air from the face, will flow to that point. The system to aim at is one in which there is a generous flow of ventilation along the face, with a flow along the back of the supports (unless the goaf is caved right up to the supports) and a localised gas bleeder system at the return end of the face, combined with a good roof control system which precludes the formation of continuous cavities in the goaf. Such a system should have no flammable gas readings higher than 1,4 % within a probe's length of the back row of supports and there should be no possibility of an ignition on the face being transmitted to the goaf.

Having set up such a system, it would be prudent to monitor continuously the airflow distribution, particularly in the localised gas bleeder system, and the flammable gas content at the intake and return ends of face. Carbon monoxide should also be monitored.

In general, South African longwall systems, with their simple, high capacity ventilation circuits and relatively low flammable gas emissions, should not constitute a serious threat provided the systems are well monitored and controlled.

It is worth bearing in mind, though, that there has been a flammable gas fire in a longwall goaf this year. The fact that there was no explosion seems to indicate that the explosive fringe was confined to a very narrow layer along the edge of the accumulation, so the control system was inherently sound.

- **Pillar extraction (Rib pillar)**

The mining situation in pillar extraction sections changes much more rapidly than it does in longwall sections, so greater demands are made on the ventilation system in keeping up with the mining changes. While the splitting of pillars requires an auxiliary ventilation system using fans and ducting, means must also be provided to course air along the goaf edge, which often means establishing a circuit around the pillars being split or waiting to be split, depending on the layout.

A gas bleeder system is probably easier to establish in rib pillar mining than in longwall mining, as pillar extraction areas are usually surrounded by pillared workings. The question of control still remains and continuous monitoring of the air from the gas bleeder system is vital, firstly as an indication that the system is working satisfactorily, but also to detect the earliest possible signs of spontaneous combustion. Should this occur a reappraisal of the situation would be required, and the gas bleeder system may have to be changed or even abandoned, perhaps combined with nitrogen injection into the goaf. Increasing the flow in the gas bleeder road, as an alternative, might cool down the heating.

(c) **Other inaccessible places**

Since the regulations make a distinction between, and apply different criteria to, accessible and inaccessible workings, it is logical to assume that places such as the cutting zone of the cutter, continuous miner or longwall shearer, can be treated the same as other inaccessible places.

Ignitions in the cutting zone are invariably due to friction and the major precautions should be aimed at preventing them. In some countries attempts are made to classify workings in terms of the degree of risk of frictional ignitions. The classification is done in terms of the strata to be cut, with quartzitic rocks and pyrites being recognised as the high risk minerals. Where they are present, more stringent ventilation standards are imposed.

- **Conventional cutters**

There are many cases on record of flammable gas being ignited in the kerf, usually due to friction caused by the cutter picks striking pyrites or other similar material. The best way of preventing such occurrences is to ensure that picks of the best design are used and that they are replaced at appropriate intervals before they become blunt. Water can be applied to reduce dust production and quench any ignition that does occur, and this, combined with correct pick usage should provide all the protection necessary. However, diligence is required to ensure that these measures are adhered to properly. Suggestions are sometimes made for ventilating the kerf, and in fact the action of the cutter chain itself will entrain some air and take it into the kerf. However, attempts to ventilate the kerf more positively could make matters worse rather than better by diluting a non-explosive rich flammable gas mixture, and converting it to an explosive mixture.

- **Continuous miners**

Continuous miners cut coal at a very high rate, in the order of 15 tons per minute. Large quantities of dust are released and even in seams with a relatively low in situ methane content the methane release will also be high. This forms a highly potent hybrid mixture. No working place ventilation system can be guaranteed to deliver sufficient ventilation to the cutting zone to dilute these mixtures, and on-board

ventilation systems are necessary. Such systems are usually designed primarily to reduce inhalable dust levels in the working place, and invariably incorporate dust collectors. Computer simulation methods, using computational fluid dynamic techniques, to predict velocity profiles around the machine and in the cutting zone, are being developed in South Africa. The primary components of on-board ventilation systems can be air powered air movers, water powered air movers, hydraulic fans or water sprays.

The working place ventilation system must be designed to supply sufficient air to prevent flammable gas accumulations and to protect the operator from high dust concentrations. A forcing system is effective in dissipating gases but the operator can only be protected adequately against dust concentrations if the dust collection system on the machine is very efficient. An exhaust system would have to induce an intake flow velocity of at least 0,5 m/s to prevent dust 'rolling back' towards the operator.

The system to be aimed at is one in which there is adequate working place ventilation, and an on-board system which induces flow, preferably by the use of water, into the cutting zone and across the machine. The flow induced by the on-board systems should be in a direction which augments the main working place ventilation. If the air enters the working on the right and exits on the left, the flow induced by the on-board system should be from right to left.

The on-board system should be operating before the machine starts to cut, particularly if a cut has been started and the machine has withdrawn and is moving back in to complete the cut. In these circumstances a 'cubby' has been created which can fill up with flammable gas, and this gas must be dispersed before cutting recommences.

If the working place ventilation is disrupted, or if the on-board system is not working correctly (or not at all) a dangerous situation can develop quickly if cutting continues. Cutting should therefore stop as soon as a disruption occurs, for whatever reason. Cutting should also stop if the flammable gas emission rises abnormally, overwhelming an otherwise satisfactory ventilation system. The

working place flammable gas level can be monitored by the statutory flammable gas measuring instruments and warning devices.

The flammable gas concentration in the cutting zone should be monitored continuously and an alarm given if the concentration exceeds a predetermined figure, or preferably the machine should be stopped automatically. There are practical difficulties, however, in getting sensors to operate in such a hostile environment. One answer may be to draw a continuous sample from the cutting zone and monitor the flammable gas concentration in the sample. If this system is adopted, care must be taken in setting the alarm or cut-off level, since the concentration at the sample point, where complete mixing has taken place, may be somewhat less than the concentration in the cutting zone, where mixing is not complete. The alarm or cut-off level would have to be set on the low side.

Another option would be to monitor some other parameter, such as airflow or water pressure. The latter is most important as water can play several vital roles, viz. inducing airflow, suppressing dust, cooling picks and, as a non-flammable substance, preventing ignitions.

The whole question of on-board ventilation systems in South African conditions needs further research. Systems optimised for most effective dust suppression may not operate optimally in diluting flammable gas in the cutting zone.

Pick design should also be kept under review as modifications to pick design in the past have had some success in reducing frictional ignitions. Water should be applied as a spray rather than a jet, and aimed behind the pick and not above it. A great deal of information on this, and other topics, is contained in 'Mine gas and coal dust explosions and methane outbursts - their causes and prevention': An M.Sc Dissertation by J D Flint.

- Longwall shearers

Shearer drums are available in a variety of designs, usually with ventilation systems in which airflow is induced by water pressure. There is significant airflow and spray water in the cutting zone, and

as long as the systems are working while cutting is taking place there is little likelihood of a frictional ignition caused by cutting.

(iii) Final comment on the choice of ventilation system

The conventional ventilation system in use at South African collieries is the exhaust system, in which the main fan is situated at the top of the upcast shaft or incline and air is drawn into the mine from the atmosphere. This is the correct system to use because a negative pressure is created throughout the mine, that is, the pressure throughout the mine is lower than that which would prevail in the 'no flow' situation. A fan stoppage would cause a rise in pressure and a resulting temporary reduction in gas emission (but a temporary increase in gas emission on restarting the fan).

The opposite system, in which the main fan pressurises the downcast shaft, with the upcast shaft open to atmosphere, is not unknown in mining but should not be used at fiery mines. With this system the pressure throughout the mine is higher than that which would prevail in the 'no flow' situation. A stoppage of the fan in this case would result in a lowering of the pressure throughout the mine. The resulting stoppage in ventilation would therefore coincide with a temporary increase in gas emission, making the mine, and the people in it, very vulnerable due to the heightened risk of incidents involving noxious or flammable gas.

4.2.4 Other

This category covers the widest range of possible causes of fires, even though the total number of fires in the category is of the same order as those in the individual categories 'Spontaneous Combustion', 'Electrical' and 'Flammable Gas'. It is gratifying to note the absence of fires due to flame cutting and welding. Most of the fires in this category took place remote from the face areas and were therefore upstream of a comparatively large number of people. For this reason they had a high life threatening potential.

(i) Belt conveyors

The safe operation of belt conveyors is covered by Regulations 11.4 and 11.5.

Regulation 11.4 applies on or in every mine and at a works where a belt conveyor is used in a confined space.

Regulation 11.5 covers coal mines specifically.

Regulation 11.4.1 requires the manager to enforce a code of safety practice, approved by him, for the installation, operation, maintenance and patrolling of the belt conveyor system.

Regulation 11.4.2 calls for adequate means for extinguishing fires to be available for immediate use along every belt conveyor and at the driving head of every belt conveyor.

This is usually taken care of by the provision of the fire hydrants and water points, either at 200 m intervals with at least 200 m of hose kept at the drive head, or at 100 m intervals with at least 120 m of hose at the drive head. The hose should be capable of reaching fires in roadways adjacent to the conveyor roadway. Fire extinguishers are also supplied to augment the fire fighting system.

It is important that specific persons are detailed to examine and report on the state of these systems at prescribed intervals.

Regulation 11.4.3 requires that means shall be available to stop the conveyor from any point along its length, or where an attendant is stationed permanently at the driving head, to signal to the attendant from any point along the belt.

Regulation 11.4.4 refers to the use of belt conveyors in series, and requires the provision of sequence interlocking to automatically stop all conveyors feeding on to a conveyor which has stopped and to prevent a conveyor starting until the conveyor on to which it feeds is moving. Since this measure is designed to prevent spillage it is, in effect, a fire prevention measure.

Regulation 11.5.1 requires that conveyor belts in coal mines be made from incombustible or fire-resistant material. The material should also be non-static. Allied to this requirement should be measures to ensure that other combustible materials do not accumulate. Over greasing of pulleys should be avoided and surplus grease must be removed.

Regulation 11.5.2 requires measures to prevent the accumulation of coal or coal dust on or around moving parts of any belt conveyor where friction is likely to cause heating.

Conveyor installations must be designed to allow for easy inspection and washing down of drive head areas, transfer points and tail ends. Conveyor structure must be

installed with reasonable clearance above the floor to facilitate inspection and cleaning.

Cables and plastic pipes should not be installed alongside belt conveyors if this can be avoided.

The correct alignment of conveyors is also important and care should be taken to ensure that conveyor belts do not rub against structures and objects, particularly timber ones.

Regulation 11.5.3 requires every conveyor to be equipped with a device which will stop the drive automatically should the belt break, jam or slip excessively.

Belt conveyor drive structures should be of integral metal construction and adequately earthed.

(ii) **Mobile machines**

Mobile machines constitute a fire risk because of their mechanical action and because of their energy supply. Fires can be caused by brakes binding or by overheating, and they can be spread by grease accumulations or oil spills. Such fires can be avoided by good maintenance, cleanliness and training. In some cases flammable materials can be changed for less flammable or non-flammable ones. A case in point is hydraulic oils. There are non-flammable hydraulic oils available and their use should be seriously considered. Each machine should be equipped with at least one 9 kg fire extinguisher, preferably two, one at either end of the machine, and where machines operate along specific, well defined routes further fire extinguishers should be placed along these routes. Equipment such as plastic pipes should not be placed along routes used by mobile machines unless this is unavoidable. Fires which break out on machines, particularly those with tyres, can escalate in minutes if not dealt with promptly and the provision of fire extinguishing equipment should be planned with this in mind. On board extinguishing systems activated automatically or by means of a 'panic button' should also be considered.

(iii) **Oils and fuels**

These materials have comparatively low flash points:

Paraffin	35°C
Diesel oil	70°C

Mineral hydraulic oil	210°C
Lubricating oils	160°C - 250°C

The temperatures relate to 'pure' substances. If a substance is contaminated by another substance with a lower flash point, then the resultant flash point will be lower than the 'pure' one. Open fires can reach temperatures of 400°C and over so that oils and fuels can easily become contributors to a fire even if they are not the source of the fire.

Regulation 10.25.7 governs the delivery of diesel fuel to underground filling stations. When the fuel is piped underground the pipes shall be drained each time after use. The fuel shall be stored in robust closed containers which do not leak. The quantity of diesel fuel stored underground shall not exceed 3 days' estimated consumption.

Regulation 10.25.8 requires underground filling stations to be adequately ventilated, constructed of non-flammable materials and to have a smooth impervious floor which must at all times be kept clean.

The design of a filling station should incorporate adequate means for washing down the floor with water and draining away the wash down water. It is preferable for the ventilation from the filling station to pass directly to a return airway, and for the entrance to be equipped with steel doors which can be closed in the event of a fire. The entrance should also have a sill to prevent diesel fuel from spilling out.

Regulation 10.25.9 decrees that diesel powered mobile units shall be refuelled only at properly established filling stations.

Regulation 10.25.10 calls for an effective system for extinguishing a fire where diesel engines are refuelled. This is usually achieved by having four 9 kg fire extinguishers positioned on the intake side of the refuelling station, often backed by a fire hydrant. The diesel storage tanks must be protected from accidental damage either by placing barricades between the tanks and the vehicles being refuelled, or by arranging that the vehicles do not enter the station to be refuelled but use a hose supply while standing outside. The diesel storage tanks should be fitted with metering devices to keep a check on receipts and issues of fuel.

Regulation 10.25.13 governs the fire prevention features of diesel servicing and repair stations, and is basically the same as that for a filling station.

Similar provisions should apply to oil stores. Spillage must be avoided if possible and where it does occur it must be absorbed with sand or other non-flammable material and then cleaned up. Oil drums and containers should be mounted on racks and provided with spill trays. Consideration should be given at any of these installations to the provision of fire detectors and possibly automatic spray systems.

When filling stations or stores which have contained oils or greases are to be abandoned, they must be cleared of all combustible material or rubble and spillages of oil, grease or diesel fuel.

(iv) Tyre stores

Permanent tyre stores should exhaust to a return airway, if possible, and consideration should be given to the installation of a steel door at the entrance, for quick sealing in the event of a fire. The floor must be kept clean and free of oil and it should not be possible for a fire from other flammable materials to reach the tyres. The number of tyres in storage must be kept to a practical minimum. A supply of water for fire fighting as well as fire extinguishers must be close at hand.

(v) Plastic piping

Where high density polyethylene piping is in use fire breaks should be installed along its length. The fire break could consist of a 5 m length of metallic pipe or UPVC pipe and the suggested spacing between fire breaks is 30 m. It should be noted that the function of a fire break is to stop a fire which starts in a section of unprotected pipe from spreading to the next section of unprotected pipe. A fire break will not arrest a well developed fire if the temperature of the fire gases is above the ignition temperature of the HDPE.

HDPE pipes should not be installed along conveyor roadways.

(vi) Flame cutting and welding

Regulation 15.10.1 states that at every fiery mine no welding, flame cutting, grinding, vulcanising, soldering or similar equipment shall be taken into or used in the workings except in a workshop established with the permission of the regional director subject to the conditions he may deem necessary.

Regulations 15.10.2 to 15.10.9 set out requirements for the design and safe operation of such workshops, essentially from the fire prevention point of view. Where cutting

or welding is carried out, with the permission of the regional director, outside a designated workshop, the conditions laid down by the regional director must be obeyed to the letter.

Typical precautions would include:

- (a) Measures to ensure that ventilation remains adequate and constant throughout the cutting/welding operation.
- (b) Measures to ensure that an adequate water supply is available before operations commence and during the whole operation.
- (c) The cleaning of the operations site before work commences, particularly the removal of oil and grease accumulations.
- (d) The continuous application of water to the floor below the material being cut or welded.
- (e) The provision of sufficient sand or stone dust to smother any fire which may develop.

Flash-back arresters should be used on all cutting equipment.

Precautions similar to those listed above would also apply where cutting/welding is being carried out in or above a shaft, except that close supervision is required both at the site where the operations are being carried out and also at any place which might be in danger from sparks or molten material. Continuous communication must be maintained at all times between the two supervisors. When work is carried out above the shaft a drum or bucket of water must be placed immediately below the site of the cutting/welding to catch sparks or molten material. When the work is complete a guard should be detailed to keep watch on the site for a period of not less than four hours, to ensure that there is no latent burning.

(vii) **Blown out shots**

Blown out shots can occur where delay detonators are used. A detonator can be dislodged by an earlier shot and it could detonate in the open, after it has been dislodged. This is unlikely to be of consequence if milli-second delay detonators are used. A blown out shot can also be caused when the burden exceeds the strength of the tamping, either because the burden is too great or the tamping is not up to

standard. The tamping will then be ejected from the holes, followed by gases heated by adiabatic compression. A methane or dust explosion could ensue if preventive measures (adequate ventilation and stone dusting) have not been taken. Correct blasting procedures, in relation to the placing and loading of holes as well as the use of high quality tamping material and techniques, should be followed. Stone dusting should be carried out right up to the face where blasting is likely to be particularly difficult, such as when blasting through faults.

(vii) Surface fires

A number of fires in the period under review started on surface and were 'carried' underground. The only safeguard is the removal of combustible material from the vicinity of mine entries, especially remote ones, more particularly when the dry season is approaching.

(ix) Compressors

Coal dust and oil accumulations must be cleaned up regularly.

Inspections should include a check on high temperature trip out circuits. (At least one compressor fire in the period under review was due to the compressor failing to trip when it began to overheat).

Air should be drawn from the coolest and cleanest source available and the inlet air filter must be cleaned out regularly.

The whole installation must be inspected regularly for the presence of oil and carbonised oil.

Chapter 23 of the Regulations must be complied with, especially Regulations 23.14.1 to 23.14.4.

(x) Timber

Comparatively little timber is used in South African coal mines. Its use should nevertheless be discouraged and any timber which cannot be removed should be covered with fire resistant or non-flammable material. The rubbing of ropes or belts on timber must not be allowed.

(xi) Wash down water

Water which has been used to wash down areas where oils or greases may have accumulated must be pumped out of the mine and not allowed to settle on the floor of old workings or airways, otherwise pools can be formed with oil floating on top.

4.2.5 Final comments on fire prevention

Fire prevention is a question of analysing a situation and visualising what could happen before it does happen. It often requires consideration of a combination of events rather than a single event and the concept of an event logic tree analysis is useful in this regard. This is highly developed in 'high risk' industries such as nuclear power generation and the petrochemical industry to 'design out' as far as possible the risk of catastrophic events. While space does not permit a complete run down of an actual situation analysis, the reader is referred to 'A Qualitative Analysis of the Inherent Fire Safety/Fire Risk in a Coal Mine' U.S. Department of Commerce, National Bureau of Standards. NBSIR 86-3502.

The end result of a situational analysis is the facility to simulate, by computer, the end results of a single event or a combination of events. The usual picture which emerges is how seemingly unrelated events in combination, can lead to disaster. Mining history is full of such 'happenings', as is life itself. Even without resorting to the sophistication of event logic trees analysis, much can be achieved by comprehensive inspections and audits, using very detailed check lists, based on the Mine Safety Rating Scheme, or adaptations of FPA Bulletin Number 47 "A fire prevention inspection guide" published by the Fire Protection Association of South Africa. Mines with sufficient resources may appoint specialists to 'take care' of fire related matters, but this should not absolve the line officials and indeed the whole labour force from responsibility. Specialists (in any field) are there to provide expertise and to cultivate interest in, and enthusiasm for, their particularly speciality. They should not take over except at the specific request of mine management.

The regulations call for persons to be appointed by the manager to be responsible for specific tasks e.g. the examination of fire fighting equipment, but general responsibility is vested in the person(s) appointed under Regulation 2.16.1.1 (a) or (b), who shall examine or cause to be examined the environmental conditions at all places in or on the mine concerned, where persons travel or work, and report in writing to the manager on: (inter alia) 'measures taken to prevent and detect mine fires'.

Inspections and examinations of equipment, and the consideration of likely fire situations, can be used to ensure that everything is in place to prevent fires, or to prevent them from

spreading. In the final analysis, however, it is the attitude and actions of officials and the labour force which count the most.

Training and the promotion of fire consciousness are of vital importance. Even on mines with a good fire record, where complacency might develop, events at other mines, in South Africa and overseas, can be used to maintain enthusiasm on the basis that 'if it happened there it could happen here'. Fire prevention is, or should be, an ongoing process. Everybody should be familiar with the measures required and put them into practice every day. However, as was stated earlier, the occurrence of a fire, particularly a major one, is a comparative rarity on many South African coal mines and immediately puts all concerned into an unfamiliar situation. The detection, location and extinguishing of such a fire is a great test of competence and training. These matters are covered in the following sections.

4.3 Detection

Regulation 11.6. 'At every mine or part of a mine, not exempted in writing by the regional director, where there is a longer interval between shifts than 6 hours, the manager shall provide for the early detection of any fire or spontaneous heating which may be taking place in the underground workings.

Some authorities would regard 6 hours as too long an interval to be used as a criterion for a fire detection system to be instituted. It is widely held amongst mine rescue personnel that a fire which is not brought under control in the first two to three hours will probably have to be sealed off. Thus a mine with a 5 hour interval between shifts and which, complying with the letter of the law, did not regard a fire detection system as necessary, could have a major problem if a fire started at the beginning of the interval between shifts.

A fire detection system is, in fact, a virtual necessity at any mine and it is important to strike the right balance between manual detection and automatic detection.

An automatic system keeps a continuous vigil at fixed points even when the mine is unoccupied, but can merely alert people if signs of fire are detected. On the other hand a human being is the only type of fire detector which, having detected a fire, can start to deal with it, or at least give a meaningful situation report.

Another question in the debate as to the relative merits of automatic fire detection and manual fire detection is sensitivity. Can the human nose detect an incipient fire at an earlier stage of its development than an electronic detector can? In most cases it can, but sense of smell can vary from one individual to another and can vary in a single individual over time.

A survey published in the National Geographic magazine indicated that 2 out of 3 people suffer a temporary loss of smell, that 1 in 100 cannot smell at all, that age and tobacco diminish odour perception and that prolonged exposure to strong odours often diminishes a person's sensitivity to that odour. Human beings often mistake one smell for another, but then, some gas detectors mistake one gas for another.

In the sections that follow, fire detection will be covered from the point of view of determining whether or not a fire has broken out. The use of fire detection techniques to follow the progress of a fire will be covered under 'Control' of fires.

4.3.1 Manual detection

A time honoured South African mining practice is the use of fire patrols, and such patrols have served the industry well. In fact, some gold mines with a particularly high fire risk employed continuous fire patrols until the risk had been reduced systematically. Fire patrols still have a role to play.

(i) Selection of personnel

The qualities required are intelligence, reliability, and physical fitness and, preferably, literacy. Physical fitness would include an unimpaired sense of smell. Selection would probably be made from long serving members of the mine labour force, who are already familiar with the layout of the mine.

(ii) Training

Fire patrol personnel must be trained in first aid and additional training should be given in the functions of fire patrols, including the route to follow and what to do in the event of a fire being detected. Trainees must be subjected to the smells which emanate from coal mine fires, and other signs of fire. They should also be trained in the use of any equipment with which they may be issued. This would probably include a self-contained self-rescuer, a personal CO monitor and a continuously operating methanometer.

(iii) Operating routine

Two men usually patrol together, performing patrols throughout the week-end and on public holidays. The first patrol should start soon after the end of the last working shift before the week-end shutdown, as the first hour after shutdown is usually regarded as the most critical period for the onset of fires (apart from spontaneous

combustion, which is a continuous threat on the mines it affects most). The patrolmen's departure and return must be logged in the lamp room, and if a patrol is overdue without explanation the most senior official on duty must be alerted. Routes must be planned and marked out by the mine overseer(s) and at each point inspected by the patrol there should be means of verifying that the point has, in fact, been covered. If patrolmen are issued with personal CO monitors they should be instructed to react to the first indication of fire (CO or smell). They should not wait for a CO indication to be confirmed by smell or visa versa. Fire patrol routes must be kept safe and all telephones must be kept in good order. Patrolmen must know where all the telephones are. Patrolmen should also be familiar with the ventilation system along their routes and should check for abnormalities e.g. ventilation more sluggish than usual, doors left open, reverse flows. They should also check that electricity is switched off and trailing cables are disconnected.

(iv) Action in case of fire

In the event of detecting a fire, patrolmen must return to fresh air by the shortest route and report by telephone as soon as possible to the designated person on the surface. They must then return to the surface to make a full report.

(v) Belt conveyor patrols

Belt conveyor fires are invariably caused by friction. The frictional heating can take place at the drive head or the tail end, where the belt is rubbing against timber or other flammable material, or where the belt rubbing against a jammed idler, heats up a layer of coal dust or oily waste. The belt itself will not ignite while it is moving, and excessive slip at the drive head should cause the conveyor to trip automatically. The most likely cause of a belt fire is the frictional heating of coal dust during the shift which, if not detected and dealt with, will spread after the conveyor has stopped and could eventually set fire to the coal on the belt. There could then be sufficient fuel burning to create lethal conditions, and perhaps even cause a fire resistant belt to burn. The critical time for the belt patrol is immediately after the conveyor has been stopped, especially if the conveyor was not run empty at the end of the shift.

Patrolmen should be on the look-out for smouldering layers of coal dust, and they should be aware that combustion can spread along a layer of coal dust without any visible sign on the surface of the layer. All layers of coal dust should be disturbed with a stick or other object to see if burning is taking place beneath the surface. During the shift patrolmen should look out for excessive spillage, discarded waste, tom belting, jammed rollers and the belt rubbing against timber or other objects.

They should also pay particular attention to the tail end and the drive head if it is not manned, as could happen in the case of tandem belts.

4.3.2 Automatic detection

This section on automatic detection is a description of available options. Some of these options have not yet been used in South African coal mines, and they are listed as suggestions for consideration.

The most commonly used fixed position automatic fire detectors in South African coal mines measure either carbon monoxide concentration or smoke density. Detectors monitoring other parameters are available, however, and they will be covered in the following sections:

The parameters which can be measured are:

- Products of combustion

These can be measured by a semi-conductor type sensor and would include a range of gases including hydrogen, alcohols and phenols. These gases can be produced by an overheating conveyor belt or electrical insulation. The 'products of combustion' detector usually incorporates a carbon monoxide sensor.

- Carbon monoxide

This is usually measured by an electrochemical cell sensor.

- Thermal noise (heat detection)

The sensor is a thermistor whose electrical resistance varies with temperature. In normal use a constant current is passed through the thermistor and the DC voltage produced is used to indicate the ambient temperature. When the thermistor is sited in a flowing air stream, small fluctuations of AC are superimposed on the DC signal. The fluctuation has a frequency of a few tens of Hertz and is referred to as 'Thermal Noise'. A small addition of heat upstream of the sensor increases the amplitude of the AC component. This change is detectable long before there is any measurable change in the ambient temperature level, and is used as the basis of heat detection.

- Smoke density

This is measured by the well known ionisation type detector.

(i) **Typical applications**

(a) **Conveyor installations**

The 'products of combustion' type sensor could be used near the drive head as it can detect gases which emanate from overheating conveyor belting. Equally a thermal noise sensor could be used to cover a very limited zone near the drive head, again to detect overheating.

However, since most conveyor belt fires follow the sequence:

smouldering coal fire

flaming, small coal fire

combined coal and conveyor fire, increasing rapidly in intensity

and since the final stage can reach lethal proportions very quickly, the correct strategy is to detect the small coal fire and extinguish it while it is still developing slowly. The choice for this task lies between the carbon monoxide detector and the smoke detector. Recent work in South Africa indicates that the smoke detector responds more quickly to smouldering combustion and flaming combustion of coal than does the carbon monoxide detector. In either event the detectors would be spaced out along the conveyor system and the spacing would depend on the sensitivity of the sensor, the response time of the sensor, the air velocity and the air quantity in the conveyor roadway. Typical spacings seem to be 300 m to 500 m. Where conveyor systems are frequently patrolled or continuously manned the spacing can be increased.

(b) **Intake airways**

There should be few intake airway fires other than conveyor fires. However, fires could occur in fixed installations such as sub-stations, pump stations, workshops or compressor stations. If these installations are not vented to return airways and not separately monitored the fires essentially become intake fires. If there is a need to detect fires in intake airways, the choice of detectors lies between carbon monoxide and smoke detectors.

(c) **Return airways**

Detectors in return airways act as sentinels for whole sections and, eventually, the whole mine, and have to indicate the occurrence of all types of fires, including spontaneous combustion. Carbon monoxide detectors should be used for this purpose.

(d) **Inbye workings**

The air entering the production sections can be monitored by carbon monoxide detectors, or smoke detectors. Account must be taken of any diesel equipment which may be in use and which could have an effect on the detectors. The carbon monoxide and the smoke detector would both be affected to some extent but it is usually possible to distinguish between diesel exhaust effects and fire. Detectors on the return side of the workings would also be affected by shot firing fumes, which would similarly be taken into account.

(e) **Fixed installations**

These could be conveyor drive heads (which have already been covered to some extent), electrical sub-stations, battery charging stations, machinery workshops and diesel filling stations, cutting and welding workshops and underground compressor stations. Much would depend on the degree to which installations are manned.

Carbon monoxide or smoke detectors could be used but consideration could also be given to product of combustion or thermal noise detectors. There are advantages in using these other detectors in some specialised applications, where the general mine detection system is based on more common types such as carbon monoxide and smoke detectors. A sort of 'electronic second opinion' could be provided by these less common detectors. The thermal noise type detector would seem to be suitable where the potential fire source is fixed absolutely e.g. electrical sub-stations, compressors. Electro-chemical carbon monoxide sensors should not be used to monitor battery charging stations because hydrogen causes a response on such sensors

(see later). The detectors should be placed down stream of the fixed installations, far enough to ensure that complete mixing of gases or smoke has taken place. A thermal noise detector would have to be much closer to the potential fire source than would a carbon monoxide or smoke detector.

If an installation is vented direct to return the detector would be placed immediately down stream of the potential fire source.

Some types of installation lend themselves to a sprinkler or spray system linked to a fire detector e.g. an underground store containing flammable materials. Care must be taken in these cases that the detector cannot possibly react to a fire in any position other than one at the actual site of the sprays. If the detector did react to a 'nearby' fire, the sprays would try to put out a 'misplaced' fire, thus diverting much needed water from the actual fire.

(ii) Installation

(a) Siting of detectors underground

Planning the positions of detectors can only be done by persons with a comprehensive knowledge of the mine, its potential fire sources and its ventilation systems. All potential fire sources must be covered with a practical minimum number of detectors, bearing in mind the maintenance load placed on the mine's resources by monitoring schemes. However, where possible the system should be designed such that a fire at a particular position will trigger at least two detectors. This is a great help in 'homing in' quickly on the position of a fire. Cognisance should also be taken of the population patterns of the mine (which parts of the mine will be populated and when). Changes to the monitoring system must be planned ahead as part of the general mine planning routine.

The actual positioning of individual detectors must be such that the air flowing over a sensor is fully representative of the air stream it is intended to monitor. Detectors monitoring general conditions (as opposed to detectors monitoring individual items of equipment) should not be placed at intersections where air streams meet, but far enough down stream of such intersections to ensure that complete mixing of the air streams has taken place. This position can be ascertained by tests with chemical smoke.

Where detectors are placed so as to protect individual potential fire sources such as conveyor drive heads and other stationary equipment, the positioning of detectors is more critical. Detectors must be positioned such that air containing the 'fire indicator medium' from all potential fire sources passes over the sensors. In the case of a conveyor drive head, potential fire sources could be brakes, the loop take-up or the drive drums.

Airflow patterns have been shown to be affected by the size and shape of the airway, the size and shape of the machinery and the air velocity. The exact position of a detector should be determined by the chemical smoke tests. Where necessary, deflectors may have to be installed to direct the flow over the detector.

Consideration should be given to external influences such as diesel fumes, shot firing, solvent vapours and the passage of men, materials and vehicles. The ease of inspection and maintenance of detectors should also be taken into account.

Smoke detectors should be positioned after chemical smoke tests are completed, as the chemical smoke may coat the sensor and reduce its sensitivity.

The final setting up of detectors should be carried out by the supplier or in accordance with detailed instructions furnished by the supplier.

(b) Cross-sensitivity of electro-chemical detectors

The rates of reaction of electro-chemical cells, including those designed to detect carbon monoxide, are temperature dependent. Most of the detectors have some form of built-in temperature compensation, but there may be inaccuracies in the event of temperature extremes. Such extremes would be unusual in South African coal mines except, perhaps, in the event of a fire.

They also react to other gases to a greater or lesser degree as shown below:

Interfering Gas	Concentration of Interfering Gas	Indication in ppm CO
Hydrogen	110 ppm	< 30 (at 20°C)
Carbon dioxide	25 % v/v	< 1
Methane	1 % v/v	> 1
Hydrogen sulphide	10 ppm	40
Sulphur dioxide	100 ppm	70

The cross sensitivity effect with hydrogen increases with increasing temperature. At 40°C, a 100 ppm concentration of hydrogen would lead to a CO indication of < 90 ppm.

(iii) Surface arrangements

Most comprehensive monitoring systems are linked to a central control facility on surface, where the whole network can be under constant surveillance. Such a surveillance system, for economic reasons, can be combined with a mine-wide methane monitoring system and a production data gathering and control system. The underground fire detectors usually have visual and/or audible alarm indicators on site as well as triggering alarms on surface. The surface facility can be used to increase the sophistication of the whole network, by using computer technology. Alarm levels can be set for the detectors, with perhaps different alarm levels for different detectors. More than one alarm level can be set for individual detectors. These could be Preliminary Alarm, Warning and Action Level. The appropriate response can be programmed into the computer system. The setting of alarm levels should be based on historical data so that influences such as diesel fumes, shot firing fumes, and changes in carbon monoxide levels in the downcast air, can be taken into account. This last point could be important at some Highveld collieries, as temperature inversions during the night and early morning, raising surface pollution levels, are a feature of Highveld winters. When a new detector is installed underground, or a detector's position is changed, and no historical data exist, the readings from the detector should be monitored for a period before the alarm level strategy is decided upon. In this way the normal variations in readings, including those caused by activities other than a fire, can be taken into account. The final setting of alarm levels should be such as to avoid false alarms while ensuring that genuine alarm conditions are not missed. Alarm levels can possibly be reduced during periods such as week-ends, when external influences, arising from production activities, are absent. On the other hand they could be raised when seasonal variations lead to an increase in carbon monoxide entering the downcast system. A rule of thumb in the case of carbon monoxide detectors is to set the first alarm 5 ppm higher than the average 'normal' level, having taken external influences into account.

On a more sophisticated plane, involving more expensive computer hardware and software, readings can be subjected to statistical analysis before a 'decision' is made as to whether or not an alarm condition has arisen. The rate of change in the measured parameter would be taken account of, as would the duration of the peak before indications of a return to 'normal'. In this way a distinction can be made between a genuine fire and some other source of the particular 'fire indicator'.

An external influence would usually exhibit a 'change - transient peak - return to normal' pattern with different numerical values for different types of influences, while a fire would exhibit a steady, continuing change, with no sign of a return to normal. The use of advanced computer technology could enhance a monitoring systems role in following the progress and effectiveness of fire fighting activities.

Alarm level setting strategy should be reviewed fairly frequently and should take cognisance of changes in the mining situation.

(iv) Tube bundle systems

Tube bundle systems have largely been replaced by systems using underground transducers with direct electronic transmission of the readings to surface. However, they still possess valuable advantages over the more modern systems. Since a tube bundle system involves drawing continuous gas samples from a wide variety of sources underground (including, if appropriate, sealed off areas) and all the gas analysis is carried out on surface, the gases being monitored can be changed to suit any special circumstances which may arise. Also, since a tube bundle system does not require any power source underground, it could remain functional in a near disaster situation, where an 'electronic' system would not. Even if the system were incapacitated, restoration of the underground sampling lines would be much easier than restoring sensors and electronic transmission lines.

The admitted disadvantage of the tube bundle system, namely that the response time is longer than that of the electronic system, is sometimes exaggerated.

(v) Personal CO monitors

The increasing use of personal CO monitors can make a contribution to the overall underground detection system by making a person's judgement of possible fire symptoms less subjective.

(vi) Maintenance

The benefits of continuous monitoring are undoubted and the prompt detection of a fire can be markedly cost-effective, but there is a price to pay in terms of maintenance. Transducers should be checked on site at regular intervals. This check should include cleaning, calibration and replacement of defective equipment. In addition, detectors should be returned to the supplier for more comprehensive maintenance, at intervals specified by the supplier. All in all the increased use of

electronic equipment on mines has changed the make-up of mine maintenance departments.

4.3.3 Detector tubes

Detector tubes are not monitoring devices in the true sense, in that a physical procedure has to be followed to obtain a reading. Nevertheless, they can serve an extremely useful purpose as indicators of abnormal situations. They also have extreme flexibility in the variety of gases which can be catered for. Cognisance should be taken of possible cross-sensitivity effects e.g. a carbon monoxide detector tube is affected by hydrocarbons.

Details of possible cross-sensitivity effects are included in technical literature available from the suppliers.

There are arguments as to whether tubes from one manufacturer can only be used with a pump supplied by the same manufacturer. A series of tests was carried out in which tubes and pumps from various manufacturers were tested against 'guaranteed' gas mixtures. Results from matched pumps and tubes were compared with results from unmatched pumps and tubes. Some were grossly unmatched in that the nominal displacement of the pump was not the gas volume on which the tube design was based. The conclusion reached was that unmatched pumps and tubes should not be used, and that if they are, the reading obtained should be regarded only as qualitative. This finding requires further comment.

When laboratory tests are carried out to measure the accuracy of matched detector tubes and pumps, under almost ideal conditions, the acceptable tolerance in the readings is $\pm 25\%$. In other words, if the 'true' gas concentration is 100 ppm, any detector tube reading between 75 ppm and 125 ppm is regarded as an acceptable quantitative assessment of the concentration. In the less than ideal conditions underground, the range of readings can be expected to be even wider. If, in spite of such wide tolerances, matched detector and pump tests constitute acceptable quantitative assessments, one must ask where the borderline between quantitative assessments and qualitative assessments actually lies.

All the detector tubes available in South Africa are designed for a sample volume of 100 ml or multiples thereof, and the nominal displaced volume of all the pumps is 100 ml.

A rather limited exercise was carried out recently in which detector tubes and pumps from all local suppliers were tested on two made up concentrations of carbon monoxide. All possible matched and unmatched combinations of tubes and pumps were tested. While, because of the limited nature of the tests, the results could not be classed as statistically significant, any experienced mine official, faced with any or all of the readings, would have made the same qualitative judgement, and would not have been misled by any of the readings. The first

series of readings would have indicated that the carbon monoxide level was higher than normal and there was reason to be concerned, and the second series would have indicated a further deterioration in the situation.

These judgements would have led to a more refined assessment of the situation, which would have necessitated the use of a sampling/analysis method other than detector tubes.

4.4 Evacuation and Control

While this section is devoted to 'Evacuation and Control', the three sections 'Prevention', 'Detection' and 'Evacuation and Control' overlap to some extent.

If a small fire is controlled very quickly and extinguished, this act of control has prevented the occurrence of a major fire. Similarly, if a flammable gas accumulation is detected and dissipated, this has prevented a possible ignition.

When an underground fire has been detected and reported, management immediately has two main priorities.

(a) Regulation 11.8

'In the case of a fire occurring in any coal mine or of a fire due to or resulting in the ignition of flammable gas at any mine all persons except those dealing with the fire or in services in connection therewith shall be withdrawn from the whole of the underground workings affected and shall only be allowed to return when safe conditions have been restored.'

(b) The other simultaneous, priority is to locate the fire and bring it under control as soon as possible.

The two priorities, therefore, are evacuation and control.

4.4.1 Evacuation

There are three main elements of a good evacuation procedure, viz: timely warning, a convenient refuge bay or safe place to go to if it is not practicable to leave the mine immediately and completely, and a self-contained self-rescuer to wear if there is a noxious atmosphere on the way to the refuge bay or safe place.

(i) **Timely warning**

Regulation 24.20.4 requires the manager to provide a device at a working place when somebody is working thereat, whereby that person can be warned against impending danger associated with a fire or explosion. Since there is no specific definition of a working place in the regulation, working place in this context must be taken to mean a place where work is taking place, no matter what the nature of the work might be, and no matter where it might be. Looked at in this light, compliance with Regulation 24.20.4 is barely practicable for some categories of workers, but is reasonably practicable for the majority.

The specification of 'a device' leaves the choice of device open, and seems to imply that, as technology improves current 'devices' should be replaced by improved ones.

At present there is no practical method of giving everybody advance warning of the approach of products of combustion no matter where the fire might be. A straight forward case could be postulated where an intake fire well outbye of the production sections triggered a fire detector (of any type) and then sent an instantaneous signal ahead to alarms in the production sections; but the same warning (in terms of time) could not be sent from fires in other positions.

Where personal CO monitors are distributed to groups of workers, the instruments should have the capability of being set to two alarm settings, a low one and a higher one, or, if instruments capable of only one alarm setting are used, two instruments should be issued to each group of workers. One instrument would be set to a low alarm level and the other to a higher one.

Care should be taken in fixing alarm levels, particularly the lower one. If diesel exhaust gases and shot firing fumes cause the warning light to flicker on and off, the alarm setting is probably too low (or there is something wrong with the ventilation system). It seems logical to set the lower alarm at about 100 ppm CO. This is well above what the normal level should be and it is the upper limit allowable in terms of Regulation 10.6.6 where persons have to work or travel, under normal conditions. According to the literature a person could work in a concentration of 100 ppm for 2 hours without getting a headache. When the lower alarm level is reached, work should stop and workers should proceed to the waiting place. The shift boss, mine overseer and mine manager should be informed and CO concentrations should be checked using a CO measuring instrument. If the CO level continues to rise its rate of increase should be timed and reported to the management. Based on this, the manager may order the evacuation to begin. If the upper alarm level is triggered, the

evacuation should begin, workers having donned their self-rescuers. The upper alarm limit should be set at about 400 ppm. This is the so called Short Term Exposure Limit, to which persons can be exposed, in the normal course of their work, for four separate 15 minute 'excursions' in an eight hour shift, provided their Time Weighted Average exposure for the whole shift does not exceed 50 ppm. A person would start to feel dizzy after working for about 45 minutes (without wearing a self-rescuer) in a CO concentration of 400 ppm, and would be close to collapse after about 2 hours. This procedure is offered as a suggestion only. The final procedure adopted obviously has to be prescribed by the mine manager. Obviously, if fire is detected by smell or other physical means, withdrawal must take place immediately.

In a fire situation a person's chance of survival can be affected adversely if he dons his self-rescuer too soon, or if he dons it too late.

A fire warning system based on electronic equipment, fixed or portable, can only work properly if maintenance and calibration schedules are adhered to meticulously. This can be a considerable burden on the mine's resources.

(ii) **Refuge bays and safe places**

The ultimate safe place is surface, or at least a downcast shaft with access to surface which can be negotiated by personnel.

At least one mine has a policy of providing all workers with direct access to surface not more than 1 500 m from any working area. This mine has a substantial number of downcast shafts augmented where necessary by drilling 1,83 m diameter raise bored holes equipped with ladderways. These raise bored holes range in depth from 25 m to 95 m, and each one is equipped with a chamber containing first aid equipment, telephones and fresh water.

This option is not available to most mines and they provide refuge bays or caches of long duration self-rescuers. Where depth and the nature of the surface terrain permit a borehole is drilled from surface into the refuge bay. In many cases the borehole is comparatively small (100 mm - 150 mm diameter, lined), capable of handling ventilation, and can be used to pass food down to the occupants of the refuge bay. The boreholes will always downcast under the influence of the mine ventilation system provided it is conventional (i.e. a surface exhaust fan, no underground booster fans and no controlled recirculation systems), and the refuge bays are kept cool and fresh. The downcast flow will seldom be sufficient to meet the needs of the fully occupied refuge bay and must be boosted by a fan or blower when the refuge

bay is in use. The booster fan is usually on surface, but one mine has fans at the bottom of the holes, in the refuge bays. Such a fan would be started to boost the flow when the first 'refugees' arrive, and stopped when surface personnel have reached the top of the hole and started the surface fan. Refuge bays should not be accommodated in bords, as the ventilation pressure could induce a flow through them. They are usually situated in an oversized pillar so that the chamber is completely surrounded by solid coal apart from the entrance. All the facilities specified in Regulation 24.20.2.2 must be provided but additional equipment such as toilets, gas testing instruments, mine plans, and lights are provided. In order to help people find the refuge bay when visibility is bad, a siren and flashing light are provided at the entrance. At least one mine is supplying air to refuge bays from cylinders of compressed air, with cylinder capacity based on 15 l/minute per man. Space is allocated on the basis of 1 m² per man.

This document does not prescribe what equipment must be provided in refuge bays. Ideas on the subject vary, and a study of what is actually supplied at various mines shows that a great deal of thought has gone into the matter. Refuge bays are usually sited about 20 minutes travelling distance from the working places, taking account of the difficulty of the route to be followed. There is some debate as to whether the route should be in the intake airways or the return airways. The choice is perhaps not so critical in coal mines as the intake and returns usually run alongside each other and access between the two is provided at regular intervals.

As a general rule refuge bays should be situated on the routes taken by the majority of the workers on their way out of the mine in the normal course of their duties. In an emergency they literally stop off at the refuge bay instead of carrying on out of the mine as they normally would. However, there should be ready access between intakes and returns adjacent to each refuge bay so that a worker who finds himself in the 'wrong' airway does not have to make a long detour to get to the refuge bay.

Some mines have exemptions from the need to provide refuge bays and instead provide safe places in the form of caches of nominal 90 minute SCSR's. Workers are equipped with the normal belt worn nominal 30 minute SCSR's, which they use to get them to the appropriate cache of 90 minute sets, which they can then use to vacate the mine. The caches of longer duration sets are usually placed 200 m - 300 m from the working places i.e. nearer than refuge bays would normally be.

In some cases the belt worn sets are the Ocenco M20, which have a shorter duration than the nominal 30 minute sets.

One of the disadvantages of using caches of long duration sets instead of refuge bays is the complication of workers having to change over from a belt worn set to a long duration set. This may have to be done in an irreparable atmosphere and, while it can be done safely, there is little room for error.

(iii) **Escape routes**

Regulation 24.20.3 requires the manager to draw up a code of practice for rescue operations and ensure that anybody who goes underground is adequately trained in the procedure necessary to ensure his survival in the event of an explosion, fire or other emergency.

In recent incidents, accounts have been given of the total disorientation that people experienced when faced with the real thing. The training they had received had not prepared them for the 'poor visibility, disorientation and panic' to quote one recent paper.

This will be referred to again under 'Training'. In the meantime it seems that escape routes, having been decided upon after careful thought and reflection, must be demarcated in such a way that they can be negotiated in poor visibility. The layout of bord and pillar mines does not lend itself to this. The escapee has to negotiate an intersection every few metres, and it is easy to take a wrong turn at an intersection in the dark without knowing it. Escapees must be guided by features they can feel or hear rather than see. Such features could be conveyors and pipes, and indeed it is stipulated that conveyor roads shall be official escape routes at some mines. In such cases route markers are fixed along the conveyor. At one mine a special 'blue line' is provided along the length of the conveyor, with metallic arrows fixed to the line to indicate the right direction. Tests are being carried out on the use of individually powered flashing lights and sirens, to be placed at conveyor junctions, and to guide persons to the conveyor roadways.

Lifelines provide another option, especially when the escape route is in a roadway which does not contain specific identifiable equipment. Such a lifeline should be such that an escapee can identify the right direction by feel. It should be at thigh or hip height so as to be used by a person walking upright, or by one crouching or crawling at floor level. It is well known that in smoke-filled airways the visibility and oxygen level is better close to the floor.

When an escape route is not in the same roadway as the refuge bay, escapees should be physically prevented from travelling past the connection leading to the

refuge bay. This could be done by placing a screen across the roadway, or if a lifeline is used, it must turn at the right point.

Escape routes must be kept as obstruction-free as possible and roof conditions must not be allowed to deteriorate. An obstruction which can be negotiated easily in calm conditions with good visibility can be almost insurmountable when visibility is poor or when one is conscious of being in extreme danger.

Finally, in designing features of escape routes which will help people who cannot see, it would seem that good practical advice could be obtained from Blind Institutes.

(iv) **Self-rescuers (ResQpacs)**

The generic term ResQpac has been adopted to denote a self-contained self-rescuer of any make and which is issued as a standard item of equipment to every person who goes underground. The use of the ResQpac is integrated into the escape strategy in that refuge bays or other safe places must be within easy reach of workmen and 'within the limits of protection afforded by a self-rescuer device'.

Every type of ResQpac must be approved by the Director General and in order to be approved it must pass a strict test protocol. In particular, it must be tested to establish the relationship between duration and breathing rate and the performance curve illustrating this relationship must pass through a point which denotes a duration of 30 minutes at a breathing rate of 30 litres a minute. This breathing rate corresponds to a moderate work rate. However, mental stress caused by fear can cause the breathing rate to increase well above that associated with a moderate work level even when physical conditions would not normally demand such a work level. Other factors affecting the breathing rate are physical size and degree of fitness. The 'procedure to ensure survival' required by law applies to everybody. The biggest, unfittest and most panic-stricken individual should be able to reach safety before his oxygen runs out.

The source of oxygen in a ResQpac can be chemically generated or simply stored in the set as compressed oxygen.

In a chemical oxygen set the user breathes into and out of a sealed bag. The exhaled air flowing in and out of the bag passes through a canister of potassium superoxide (KO_2), CO_2 and moisture from the exhaled air react with the KO_2 to produce oxygen, which reconditions the air available to the user. This inspired air should contain not more than 2,5 % CO_2 and over 21 % O_2 . The amount of KO_2 is

determined by the rate of absorption of CO₂. More oxygen is generated than is needed and surplus air is expelled via a relief valve. Heat is produced by the chemical reactions and this must be dissipated to prevent the inspired air becoming too hot.

This type of set has no high precision moving parts and inspections are limited to visual integrity checks with occasional vacuum checks. Special sets have to be used for training because the cost of recharging a set with KO₂ after it has been used is a high proportion of the original cost.

Compressed oxygen sets also work on the basis of recycling the user's exhaled breath. Oxygen is supplied from a pressurised vessel via a demand valve. The CO₂ from the exhaled breath is absorbed by soda lime or lithium hydroxide in a special absorption unit. The available air has to be dehumidified to prevent it becoming intolerable even though the air temperature is generally lower than that from a KO₂ set. The replenishment cost for a compressed oxygen set is much lower, relatively, than for a KO₂ set, but the compressed oxygen set contains high precision moving parts and the oxygen container is pressurised to 25 000 kPa. Inspection and maintenance are more onerous than in the case of a KO₂ set.

The test protocol and field trials tested many features, including design, ease of donning and use, ease of maintenance and durability.

Several successful escapes have been carried out by men using ResQpacs and have been documented in greater or lesser detail. There are, however, misgivings about the lack of retention of skills learned during basic training, when judged on the criterion of being able to don a ResQpac, in total darkness, in 15 seconds or less. It might be added that this lack of retention is by no means confined to South African mineworkers.

(v) Training

The whole series of requirements covered by Regulations 24.20.1 to 24.20.5, which refer to the supply of ResQpacs and refuge bays, the provision of early warning systems and the specifying of escape routes, will not ensure survival without adequate training.

Regulation 24.20.3 thus specifies training in 'the use of a self-rescuing device' and in 'the procedure necessary to ensure survival' and specifies that training shall be

repeated at intervals not exceeding six months and that any person who has been absent from the mine for a continuous period of more than 30 days shall be retrained.

All the topics referred to in this section on 'Evacuation' are topics on which training must be focused. All workers are trained to don and use a ResQpac, and are tested to ensure they have reached the required standard of proficiency. They are also shown where the appropriate refuge bay is, what route to follow to get to the refuge bay, and are instructed, according to a documented list, on what to do (and not to do) when they get there.

All these matters, and more, are documented in the proceedings of a one day workshop held by the Collieries branch of the Mine Ventilation Society of South Africa, in May 1992, entitled 'A Review of the Rescuer Strategy in South African Coal Mines'.

The following are direct quotations from contributors at the workshop, all relating to training or the results of training.

- Following a goaf collapse which people thought was an explosion

63 % did not use the ResQpac while of the remainder, 98 % did not use it correctly. Only 2 % donned the sets correctly and would have survived had it been a real explosion.

- Following a fire at the same mine

All the ResQpacs were donned and used correctly. This is a significant improvement on the previous effort (the one above) and can be ascribed to the intensive training effort.

- Following a methane ignition at the same mine

All persons who were not directly exposed to the methane ignition donned their sets correctly. Of those in the immediate vicinity of the incident, only one used the set correctly.

- From an experienced ex-Proto member caught in the ignition

My first reaction was to escape to safety. Initially I had forgotten to don my ResQpac and only after I realised that it was difficult to breath, did I

remember the ResQpac. I donned the ResQpac but failed to inhale through it. I then found the mouth plug was not removed. I would not have escaped to safety had it not been for the ResQpac.

- **General comment on reaction to the ignition**

This was the first real practical experience in a methane-related emergency. Evaluation revealed that, in general, the personnel performed well under these adverse conditions.

- **Conclusions (based on the above ignition)**

The bottom line remains the human factor, and how the human responds to emergency situations. An appeal is made to the whole mining industry to investigate ways and means whereby personnel can be trained under conditions which simulate as closely as possible realistic emergency situations. The provision of a dark room proved to be inadequate as a substitute for an emergency situation.

- **Training (at another mine)**

All new underground employees are given practical training on how to use the ResQpac and the Ocenco. The meaning of the Mine Com alarms and the escape route to be followed in the event of an emergency are explained. Spot checks are done monthly in production sections. A person, selected at random, must correctly don a ResQpac, proceed to the Ocenco bay, describe the changeover from a ResQpac to an Ocenco and then proceed to escape via the recognised route.

- **Training (at another mine)**

Escape routes from sections to refuge bays are followed regularly (twice a month) by the shift boss, miner and section personnel. Additionally, a section per month is taken through escape procedures whereby one volunteer dons a ResQpac and the whole section follows the escape route to the refuge bay. Monitoring is done by the Environmental Control Department regarding correctness and speed of donning, time taken to walk to the refuge bay and the full time period of the ResQpac's capacity.

- **Training (at another mine)**

Careful consideration must be given to what action personnel are trained to take when one or more of their colleagues should collapse. This may affect the survival rate. Monthly emergency drills are essential. These drills must be analysed and anomalies recorded for feedback to the participating employees when they reach the place of safety.

- **Training (at another mine)**

Instructors from the Training Centre go underground regularly with training sets and ask workers randomly to demonstrate the donning procedure and the escape strategy.

Management and Safety Department question workers on the escape strategy and escape procedures during routine underground visits. All underground workers walk the entire escape route in their work areas once every 3 months. A procedure is being drawn up whereby a certain percentage of the ResQpacs will be tested for reliability. Persons will be selected to wear these ResQpack along the escape routes. This will serve as additional training and give additional information as to improvements that can be made to the escape routes.

- **Training (at another mine)**

Different scenarios and combinations of scenarios are considered and persons are taught to prepare themselves for specific responses. Courses of action are planned for incidents and combinations of incidents regarding:

Explosions

Spontaneous Combustion

Open Fires

Major Roof Collapse

Methane Outbursts.

One very important aspect is to remember to sit down and wait for help when vision is obstructed due to dust and disorientation, after being thrown about.

- **Training (at another mine)**

In the case of an explosion or where the employee is disorientated, it should be emphasised that he should wait until he has regained a sense of direction or until help has been rendered.

- **Retaining skills levels**

Research shows clearly that when an acquired skill is not used, the performance degradation curve is exponential. It also shows that the most beneficial way of maintaining that skill is to provide practise before the degradation becomes too great.

The success of implementing any organisational change programme is critically dependent on demonstrated management commitment.

Other questions which should be asked:

Is the desired behaviour punishing?

Is it more rewarding not to perform?

Does it matter if the trainee doesn't perform (in tests)?

Do all trainees understand why they are being asked to perform in the desired way?

(vi) **Final comments on training**

There could be fundamental misunderstanding as to why people are asked to demonstrate, when there is no fire, what they would do if there was a fire. Perhaps that is the real issue, and the people concerned are convinced in their own minds that if the real thing ever happened, they could, and would, take the correct action. This comes back to a point quoted earlier. There is a need to be able to simulate the sense of shock and confusion produced by an explosion, or the helplessness when one is engulfed by smoke.

During the black-out in the Second World War, vehicles' head lamps were covered by black screens with two or three narrow, hooded, slits positioned such that the limited amount of light which did penetrate was directed downwards. No light was allowed to shine upwards. Perhaps simulated escapes could be carried out with all

fixed lights extinguished and all cap lamps blacked out so as to emit very little light, and in one direction only.

4.4.2 Control

(i) Types of fires

Fires are classified according to a convention recognised in South Africa, which applies equally to underground and surface fires. The classification also determines the method(s) of extinguishing the fires.

Class A

These fires involve solid materials, usually carbonaceous, such as coal, wood or paper. They produce glowing embers and, finally, ash.

Class B

These fires involve liquids or liquifiable solids such as petrol, oil, grease and cooking fat.

Class C

Fires involving materials covered in Classes A, B, D and E and occurring in the presence of live electrical installations.

Class D

These fires involve metals such as aluminium and magnesium, and pose special hazards.

Class E

Fires involving flammable gases stored under atmospheric or higher pressure.

(ii) Fire extinguishing agents

(a) Water

This is the cheapest, most efficient and readily available fire fighting agent. It has a high heat absorption capacity and it is easily transported to fires and applied to them. It works by heat reduction and oxygen removal. On application it turns to steam, extracting latent heat from the fire, thus cooling it, and expands by a factor of 1 700, displacing air and reducing the oxygen content to about 9 %. It can be applied as a jet or as a 'fog'. The quantity of water required will depend on the size of the fire, and a major mine fire can consume up to 145 l/s, requiring 15 to 20, 63 mm diameter fire hoses. Water is most effective on Class A fires. It is not effective on flammable liquid fires as these liquids are invariably lighter than water, and they will float to the surface and carry on burning. Water should not be used on electrical fires while the circuits are live. Burning metals react with water and this can cause the fire to intensify. Similarly water is not effective on flammable gas fires. However, water is valuable for cooling down the surroundings of any fire and thus preventing it from spreading.

(b) Sand and stone dust

These materials extinguish fires by smothering. They have no cooling effect but do not promote any dangerous chemical reactions with burning material. They cannot be applied in large quantity and are therefore effective on small Class A fires, but not Class B. They should not be used on machinery, particularly sand as it is abrasive. Sand can be used on burning metal fires as the heat converts it to molten silica which eventually forms a crust.

(c) Foam

Mechanical foam is produced by mixing liquid foam concentrate with water and aerating it to produce bubbles. It has a low specific gravity and therefore floats on burning flammable liquid pools, thus extinguishing them.

Low expansion foam has an expansion ratio of 10:1 and can be used for direct attack on an open fire. It can also be used to cover falls of rock with burning rubble underneath it, or for filling small paddocks where burning material is inaccessible.

Medium expansion foam has an expansion ratio of 80:1. It can be applied directly to fires and larger areas can be filled than by low expansion foam. It can also be used in inaccessible areas where there is smouldering under falls of ground.

High expansion foam has an expansion ratio of 1 000:1. Large volumes of foam can be produced, capable of filling large areas of the mine.

Advantages of foam

- It has the same cooling properties as water.
- It is an excellent shield against radiant heat.
- The continuous breaking of bubbles causes water to be deposited on all surfaces.
- Foam has been used to create a ventilation barrier, either temporarily while seals were being built, or permanently, in place of seals.
- Remote operation removes people from bad conditions.
- Large volumes can be filled.
- The equipment can be operated by mine personnel.

Disadvantages of foam

- If underground water is used on coal beneficiation plants, foam in the water can affect the washing process.
- Foam compounds are detergents and remove lubricants from pumps.
- The rate of settlement in water is reduced by foam, leading to dirtier water.
- Foam tends to take the line of least resistance, and its effect is dissipated if there are many splits.
- Skin irritation can result from prolonged contact.

Foam can only be effective if it reaches the fire and keeps it covered.

Preliminary surveys are necessary to ensure that the prospects of success are good. It is expected that prospects will seldom be good in South African coal mines because the wide open areas will make it difficult to concentrate the foam where it is needed.

(d) **Carbon dioxide**

This can be used in fire extinguishers or in large volumes and it can be used to render a whole fire area 'inert'. When used for the latter purpose it is best used where the fire is below the point of injection. It will flow downhill against the ventilation current and will not mix with the air. It will also flow through rocks to get to hot spots. In general it will act like water but it can be used where the use of water would be inappropriate.

(e) **Nitrogen**

Nitrogen is also good for inerting large volumes. Because its specific gravity is almost equal to that of air, it will mix readily with the ventilation current, so the ventilation can carry it to the fire. A tanker of liquid nitrogen yields 50 % more gas than a tanker of carbon dioxide.

(f) **Fire extinguishers**

- **Dry powder. (sodium bicarbonate, potassium or ammonium phosphate base).**

These extinguishers are used in Class B and Class C fires and smother the fires.

Advantages

- **They are rapid in action.**
- **The dry powder does not conduct electricity.**
- **The extinguishers are not subject to freezing.**

Disadvantages

- **The chemical has no lasting blanket effect on flammable liquids and ignition sources must be overcome.**
- **The chemical has no cooling properties and care must be taken that re-ignition does not occur.**
- **The powder forms sticky deposits, which are detrimental to delicate machinery and equipment.**

The extinguishers discharge a dense stream of powder 2 m to 5 m for 10 to 30 seconds.

- **The water CO₂ extinguisher**

The extinguisher can be used on Class A fires where cooling and wetting is required, but not on oil, grease, flammable liquid or electrical fires. The 10 l model will empty in one minute and has a range of 6 m to 8 m.

Advantages

- The expelled liquid is pure water only.
- Recharging is quick.
- It has excellent cooling properties.
- Water will penetrate to deep seated fires.

Disadvantages

- Water is a conductor of electricity and cannot be used on electrical fires.
- It will cause Class B fires to flare up and may cause explosions when used on Class D fires.

- **Carbon dioxide extinguisher**

This is a high pressure cylinder containing liquid carbon dioxide. When the extinguisher is activated the liquid carbon dioxide flows through a diptube, a valve and a small hole at the base of the horn, where it is turned into gas and snow. Care must be taken not to touch the 'wrong' parts of the horn as the very low temperature will burn the hands. The effective range of 1,5 kg to 50 kg units is 1 m to 2 m and the discharge time is 15 to 60 seconds.

The extinguisher is used on Class B and Class C fires.

Advantages

- It is very rapid in action and is independent of ambient temperature.

- It is clean. The gas dispenses quickly and produces no chemical action. The very cold gas, however, can damage articles which are susceptible to cold.
- The gas is a non-conductor of electricity and leaves no harmful deposits on electrical equipment.
- It may be used in the early stages of flammable liquid fires. The liquids are not contaminated by the gas.

Disadvantages

- The cylinder is very heavy (to withstand the high pressure of the liquid CO₂).
- The contents of the cylinder cannot be checked visually and the extinguisher must be weighed accurately.
- The cylinders must be sent away for recharging.
- The cooling properties of CO₂ gas are limited and re-ignition of fires could occur.
- The effective range of the extinguisher is limited, especially in windy conditions.
- Direct impingement of the gas onto delicate equipment could cause cold shock damage.
- Under dry conditions the discharge of CO₂ extinguishers generates static electricity, which is uncomfortable to the user.

Smoke density increases with increased humidity. CO₂ extinguishers have been discharged into smoke laden airways to cause condensation and lower the humidity, thus reducing the smoke density.

- BCF (Bromochlorodifluoromethane)

BCF is liquified gas which 'kills' fires by interfering with the flame chain reaction. It can be used on all fires except Class D (metals) and is particularly useful on live electrical fires. It is discharged at high velocity, which gives the extinguisher a long throw. The operator can therefore be at a safe distance from the fire. The gas is hazardous to human beings and has a marked ozone depletion potential. BCF extinguishers are banned in some countries.

Advantages

- Particularly suitable for electrical fires.
- The liquid vaporises to dense gas which blankets the fire.
- The gas leaves no mess after discharge.

Disadvantages

- Has limited cooling properties, so cannot be used on a deep seated fire.
- Cannot be used in confined spaces, where there is a risk of breathing in the products of decomposition.
- The acrid smell could force the user to evacuate the fire area before the fire is extinguished.
- The direct impingement of the medium on delicate equipment could cause cold shock damage.

(iii) Fire fighting (technical aspects)

This section deals only with the technical and scientific aspects of controlling fires. It does not address the management organisation and the standard emergency procedures which are necessary to ensure that every person knows his duties and responsibilities, that decisions are based on sound information and judgement, and that there is no confusion as to who must implement the decisions. These matters are largely documented in the codes of practice required in terms of Regulation 11.1.2.(b).

(a) Small fires

A small fire is considered to be a fire which a single person is reasonably able to extinguish, without assistance, in three to five minutes. Immediate attack is essential with whatever materials are at hand. If the fire is put out it will become the subject of an enquiry and an incident report. If it is not put out it is by definition no longer a small fire and will probably be handled by the mine's fire fighting organisation, assisted if necessary by the Rescue Training Service. Evacuation of affected parts of the mine will also be necessary.

(b) Factors which contribute to more serious fires

Of the 120 fires and ignitions reported between 1986 and April 1992, only 25 involved rescue brigades and only 9 required more than two teams. A very high proportion of incidents were small fires. While this is reassuring, it would be prudent to consider the potential for more serious fires and how such fires would be handled. A recent example of such an occurrence is one in which a Load Haul Dump vehicle caught fire. The fire could not be extinguished while it was still confined to the vehicle, and it eventually spread to such an extent that a sizeable portion of the mine was sealed off. This shows that the potential for more serious fires exists and it also illustrates the view, held by many mine rescue officials, that a fire which is not brought under control in the first two to three hours will have to be sealed off. Sealing off is not critical if the fire can be confined by sealing off a fairly small portion of the mine, but if the fire were to be in one of the main arteries, virtually the whole mine would have to be sealed off. It is therefore vital that fires are dealt with in such a way that they do not get out of control. A fire is out of control when the fire fighters cannot apply extinguishing agents to it in the right places and insufficient quantity to arrest its propagation.

This may be because conditions are so hot that the fire fighters cannot get close to the fire, or because visibility is limited due to smoke rollback or recalculation of smoke laden air back into the intake. The fire can also be considered to be out of control if the fire fighters are at risk because an explosion may be imminent. Such a situation might arise if there is danger of gaseous distillates released by the fire reversing back through the fire. There may also be an explosion danger where tanks of flammable liquids are involved. Lastly the fire may be inaccessible because of roof falls.

It is often said that coal mines are vulnerable to fires because they are literally full of fuel. This could be said to apply even more to South African coal mines because the whole mine, including the main airway system, is usually developed in coal, unlike European collieries in which the main arteries are developed in rock.

However, in a coal mine fire, it is volatile gases and tars, driven off from the coal by heat (pyrolysis), which form the fuel which is added to the fire. These volatiles will only be driven off if the temperature is high enough, and the hottest place is probably the top 200 mm immediately below the roof downstream of the fire. The fuel producing region therefore is the roof and

the top 200 mm of the coal sides. If there is no coal in the immediate roof, then the fuel producing area is reduced. The presence of timber downstream of the fire will make matters worse as timber gives off more volatiles than coal, and at a lower temperature.

In any event, the application of water against the roof, in the form of fine mist, downstream of the fire, reduces the temperature, thus reducing the release of volatiles and robbing the fire of some of its fuel.

Obviously this will have to be done as soon as possible after the fire breaks out as it will be impossible if the fire becomes highly developed. Failure to get downstream sprays in position soon enough could be a reason for the fire eventually getting out of control.

Timber will give off gases and volatiles at a temperature of 300°C. The temperature at which coal gives off volatiles, tars, etc. and the rate at which they are given off, depends on the volatile content of the coal. Methane, hydrogen and carbon monoxide are distilled freely at 300 - 400°C and are burnt at 650 - 750°C. Coal burns generally at temperatures in excess of 650°C.¹

The total strategy in fighting an open fire then, is to apply water directly to the fire from the upstream side to cool the fire (remove heat), to smother the fire (remove oxygen), and to cool the roof downstream to retard pyrolysis (remove fuel). The three sides of the well known fire triangle are therefore tackled simultaneously. Water is able to reduce the oxygen content at a fire because, in turning to steam, it expands by a factor of 1 700, thus displacing some of the air. This is obviously very effective when the fire is in a confined space. It would be less effective in a main airway with a substantial airflow, unless a great deal more water was used. This raises the question of reducing the airflow over the fire.

¹ Flame is a gas phase phenomenon, and, clearly, flaming combustion of liquid and solid fuels must involve their conversion to gaseous form. For burning liquids this is normally simple evaporative boiling at the surface, but for most solids, chemical decomposition or pyrolysis is necessary to yield products of sufficiently low molecular weight, that can evolve from the surface and enter the flame.

Reducing the airflow can have the effect of reducing the 'fanning of the flames'. It can also slow down the passage of noxious gases around the mine, which can be of benefit. However, it can increase smoke rollback, it could possibly cause reversal of gases through the fire, cause changes in ventilation distribution in other parts of the mine which would need to be assessed in advance, and finally it could possibly lead to a fuel rich fire. All these factors could contribute to the fire getting out of control, so a decision to reduce the airflow, while it should always be considered, is not one to be taken lightly.

Smoke rollback from the fire will occur in a level airway if the air velocity is too low. This is a convection effect, and if the airflow is downdip the effect will be greater. If the airflow is updip smoke rollback is unlikely to occur. The height of the airway has an effect. For a given fire intensity the higher the airway the more pronounced the convection will be.

Table 5 MINIMUM AIR VELOCITY REQUIRED TO PREVENT SMOKE ROLLBACK

Height of Airway (m)	Minimum Air Velocity (m/s)		
	0 Dip	10 % Dip	20 % Dip
1,2	1	1,2	1,5
1,8	1,25	1,45	1,8
2,4	1,4	1,7	2,1
3,1	1,6	1,9	2,8

A further danger is the heat accompanying the smoke. This heat can loosen roof bolts and cause roof falls. Smoke rollback can be prevented, or reversed, by erecting hurdles to increase the velocity near the roof, as one would do to dissipate layers of methane. Care must be taken not to restrict the airflow too much, thus reducing the total flow, especially if the velocity is particularly low already, say 0,5 m/s. Smoke is made up of carbon particles and water vapour. The higher the relative humidity the denser the smoke. Materials containing carbon black tend to produce dense smoke when they burn, as the very fine carbon particles are liberated more easily than the carbon in wood or coal. Smoke density also increases as oxygen content increases, and finally, smoke is always accompanied by carbon monoxide. While smoke itself is not toxic, it can disable people by making it distressing to breathe. Smoke hazards can be alleviated somewhat by reducing the relative humidity, which is not easy given the amount of water present at a

fire, and reducing the oxygen level, but the most important factor in keeping smoke away from fire fighters is air velocity.

Reversal of airflow goes hand in hand with smoke rollback. It is only likely to happen if air velocities are marginal. It should be remembered, though, that the convection effects created by the fire, and which contribute to smoke rollback, impose a restriction on the circuit, so that slight disruptions of the ventilation system, such as might arise from leaving ventilation doors open, could cause a reversal. That is why it is so important, during a fire, to avoid unplanned and uncontrolled ventilation changes.

Where a fire occurs in a dipping airway, reversal is almost inevitable and contingency plans should be made, while the fire is in its early stages, to enhance the flow in the normal direction. This can be achieved by regulating other circuits or by boosting the flow in the airway where the fire is. Fans in open circuit on the intake side of the fire could be used for this. This option is not so easy to exercise in multiple entry mines like South African coal mines. The danger attributed to a reversal of air through the fire is a possible explosion of the fire gases contained in the air. But what are the chances of gases around a fire being explosive? Strang and Mackenzie-Wood, of the Southern Mines Rescue Station, New South Wales, have this to say, 'In non-gassy or slightly gassy coal seams, the usual concern of those involved in fighting or sealing fires is exposure to smoke and gases. The possibility of explosive gases being generated receives less attention. A blazing fire in the confines of a coal mine entry shares much in common with the operation of a beehive coke oven. In beehive cooking, coal is placed in the lower portion of a pre-heated oven. The heat in the oven initiates the release of gases (methane, carbon monoxide and carbon dioxide) which collect above the coal and are mixed with a controlled amount of air, where they are burned and discharged into the atmosphere. The heat from the controlled burning of the gases permeates downward, progressively distilling the volatile matter contained in the coal until it is finally coked. The same coke oven reactions take place in and around a mine fire except one is under controlled conditions while the other (the mine fire) is semi-controlled. To compare the similarities of a raging mine fire burning in the confines of an entry to that of a coke oven makes possible a better understanding of the unappreciated hazards of combating or sealing mine fires'. The interpretation of gas samples taken from sealed areas always receives a lot of attention, and will be referred to later in this report. The taking and interpretation of samples downstream of an open fire is not so well documented. When a fire is fully developed and is

oxygen rich, it reaches a state of equilibrium. The production of fire gases is proportional to the flow rate so that the concentration of the gases will remain fairly constant irrespective of the flow rate, once equilibrium has been regained following a change in flow rate. If the flow rate is reduced the gas concentration will increase, but as the fire intensity adjusts to the reduced airflow, the concentrations tend to regain their previous values. There is a time lag between one equilibrium situation and the heat, during which the gas concentrations are heightened. This is why sudden, pronounced changes should not be made to the ventilation flow. If changes are planned, particularly reductions in ventilation, they should be made in small, progressive steps. If the gas concentrations do not regain their previous values when a new equilibrium point should have been reached, it could be a sign that characteristics of the fire are changing, which is almost the same as saying that a change in gas concentrations with a constant ventilation is a sign that the fire characteristics are changing.

It is important that trends be followed, and that decisions are not based on specific values. Airflow trends should always be considered, in confirmation with gas trends. One authority advocates frequent inspections downstream of the fire as a means of assessing how the fire is progressing. Great significance is attached to pulsations in the airstream as a sign of an impending explosion. Such pulsations could be due to the fire 'searching' for air as it grows rapidly in size, or they could be due to minor ignitions, indicating the presence of flammable gas at the fire, and which could be a prelude to a bigger explosion.

If a reversal of airflow is carried out purposely, perhaps to change the side from which the fire is to be attacked, it must be remembered that there may be unburned volatiles downstream of the fire and that they would have to pass through the fire on reversal. It is important that the downstream zone over which reversal takes place should be as short as practicable so as to limit the quantity of gas which has to pass through the fire. The reversal should be carried out while the fire is of modest size, and if it is delayed until the fire has grown in size, it might be better not to carry it out at all.

The concept of fuel rich fires was first suggested in 1969 by Dr A F Roberts. Mention has been made previously of heat downstream of a fire causing pyrolysis of the coal and/or timber. The pyrolysis or distillation products are fuels which feed the fire and which are generally consumed provided there is sufficient oxygen present for combustion to take place. This is called an

oxygen rich fire. In the case of a very big fire which consumes all the available oxygen, there will be no combustion beyond the point at which the oxygen is exhausted. The temperature will still be very high downstream of the fire and pyrolysis of coal or timber will continue. The gases produced will no longer be burned because of the lack of oxygen. There is more fuel than can be consumed, so the fire is described as fuel rich.

The mixture of gases downstream of the fire, has approximately the same composition as the products of distillation. These are:

Timber	
Gas	Volume per cent
CO	30
CO ₂	60
H ₂	-
CH ₄	10

Coal	
Gas	Volume per cent
CO	6-14
CO ₂	1-3
H ₂	30-60
CH ₄	20-40

The atmosphere downstream of a fire which has become fuel rich is very hot, sometimes as hot as 1 100°C (Roberts records 1 300°C in controlled tests) and contains dense, tarry smoke. If the stream of gas mixes with another stream of air containing sufficient oxygen, it will possibly burst into flame and could explode. The same would apply to the gases reversing through the fire.

Conditions favouring the development of a fuel rich fire are therefore:

- an intense fire consuming large quantities of oxygen,
- a somewhat restricted airflow, and
- a continuous source of fuel downstream of the fire from which distillates can be produced.

The chances of a fuel rich fire occurring in a South African coal mine are probably not very high. The experiments conducted by Roberts all involved timber, and there is no experimental evidence relating to fuel rich fires involved in coal.

However, temperatures would certainly be high enough to promote distillation of coal. Two suspected fuel rich fires, both in the USA, involved rather extreme circumstances. In the first one the fire was started when over 200 litres of hydraulic oil ignited. Downstream was an area of highly volatile coal.

Rescue workers attempting to enter return airways about 600 m downstream of the fire told of hot, dense smoke bursting into flames when the doors were opened.

In the other fire hydraulic oil was ignited, and a figure of 700 litres of oil is mentioned. The main fan was stopped in the hope of preventing smoke spreading to inbye sections. Smoke rollback forced fire fighting teams to retreat. The fire continued to burn with virtually no ventilation, and when the fan was restarted there was an explosion. The most likely scenario for a fuel rich fire in a South African coal mine would probably involve a vehicle or a continuous miner close to other fuels, perhaps accompanied by a ventilation breakdown.

Mention has been made of possible reversals of the ventilation caused by fires in dipping roadways. The convection effects that cause this can cause other disturbances such as reverse leakage through doors. Essentially a fire in an inclined roadway acts like a booster fan, with the updip side of the fire simulating the higher pressure (discharge) side of the fan and the downdip side simulating the lower pressure (suction) side of the fan. Just as a badly sited booster fan can cause air to flow from return to intake, thus contaminating the intake, a fire can have the same effect. A booster fan or a fire would also have a more marked effect inbye, where the ventilation pressure between intake and returns is relatively small, than it would have further outbye, where the ventilation pressure is greater. This should be borne in mind where possible fire sites (sub-stations, workshops, etc.) are vented to return. If these sites are well inbye where the ventilation pressure is low, a fire may boost the return pressure to a point where air, and smoke, leaks back into the intake, which is what the system is designed to avoid. The relative flatness of the workings limits buoyancy effects, in South African coal mines.

The increase in mechanisation in the coal mines has brought with it a special fire risk, just as trackless mechanised mining has in the gold mines. The machines themselves contain fuel oil or a high energy electrical power source, together with hydraulic oil, lubrication oil and tyres, all of them flammable to a greater or lesser degree, while the back-up facilities such as stores, fuelling stations and workshops also contain flammable materials. These back-up facilities should be situated on surface. Fires on these vehicles have the potential to cause extensive fires, as recent experience

shows, and it is essential that adequate and appropriate fire extinguishing equipment is on hand.

It is equally important that drivers and associated personnel are well trained fire fighters because they form the first and most important line of defence. In particular, fire extinguishers should be positioned so that the fire can be tackled from either side of the machine, so that at least one is always available to tackle it from the upwind side. Operators should be trained to render the machine immobile, to switch off the power supply, to use fire extinguishers, and to stand by the machine when the fire appears to be out to deal with any further outbreak.

Above all, operators should be told at what point to summon further help. Persons who work regularly in back-up facilities should be formed into fire fighting teams and trained to deal with fires specific to their facility.

Section 4.4.2(iii)(b) has thus far outlined some of the factors or situations which can cause fire fighting operations to be discontinued, allowing the fire to become out of control. An interesting development in the USA is designed to allow fire fighting operations to continue beyond the point where efforts would normally be abandoned. This is the development of a remotely controlled fire fighting vehicle, an adaptation of a research prototype vehicle developed by the US Navy for fighting aircraft carrier deck fires. Although a vehicle has been acquired and is undergoing modification the project as a whole is in the conceptual stage. It is intended that the vehicle will be controlled by an operator close to the fresh air base. The maximum range would be about 300 m. Communication between the operator and the vehicle would be by a fibre optic link. Information would pass from the vehicle to the operator and control instructions would pass in the opposite direction.

The vehicle would be equipped with a video camera, a through-smoke infra-red viewing system, various proximity, temperature and gas sensors, and one or more water nozzles. The remote control platform would contain a video camera and sensor output displays to enable the operator to navigate the vehicle through mine workings to the fire, and apply water or foam to the fire from close range. The vehicle in its present form is 1,22 m high, 1,78 m long and 1,42 m wide. It weighs 1 500 kg and is powered by two 4,5 kW motors. A similar type of vehicle exists in Russia.

(c) Fires other than those in the goaf

- Open fires at the coal face in bord and pillar extraction

Fires are likely to be caused by frictional ignition due to the operation of production machines, the ignition of oil or grease, electrical faults, or blasting, which may leave flammable gas burning in strata breaks (hanging flames).

Such fires can usually be tackled conventionally, using water or fire extinguishers. Foam might also be an option as the working place will usually form a ready made paddock. The main objective would of course, be to put out the fire, but a second important objective would be to stop the fire spreading to the 'through ventilation' system, from where it could spread more rapidly throughout the section. If the fire should spread to through ventilation and still continue unabated, contingency plans for sealing off should be set in motion. A close watch must be kept on flammable gas levels to detect signs that an explosion may be imminent.

If there is evidence that burning of flammable gas took place in roof breaks then, after the fire is out, inspections should be made for hanging flames. A hanging flame occurs when a small jet of flammable gas continues to burn, almost like a pilot flame. If a further build up of flammable gas takes place it could be ignited by the hanging flame. Signs of a hanging flame would include a slightly elevated carbon monoxide level, faint popping sounds which might be heard in almost total silence, and possibly slight luminosity, visible when the lights are switched off. Traces of carbon monoxide would be almost impossible to pick up after a fire had just been put out because the whole area could be filled with a highly complex mixture of gases. Hanging flames are more likely to occur in proximity to goaf than in 'solid' bord and pillar workings.

- Open fires at the coal face in longwall or pillar extraction

Types of fires likely to arise are:

- burning gas or coal dust caused by frictional sparking,
- fires on the face caused by an extension of the above,

- conveyor fires which ignite coal
- oil fires,
- electrical fires,
- fires caused by blasting (hanging flames again?),
- spontaneous combustion.

Direct fire fighting methods can be used but a close watch must be kept on flammable gas levels. Water jets or sprays can be used, or foam may be used if it is feasible to build a paddock using brattice cloth. If the fire shows signs of getting out of control, or if flammable gas is burning in roof breaks, it would be wise to seal off the section.

The basic requirements for sealing off would be as follows:

- Stoppings should be far enough from the coal face and sited such that the faces of the stoppings can be ventilated after sealing.
 - Adequate ventilation should be maintained through the stopping while it is being built and facilities for simultaneous sealing should be provided.
 - Arrangements should be made for taking gas samples through the seals and measuring pressure across the seals.
 - Stoppings should be completed as quickly as possible.
 - Stoppings should be of explosion proof construction.
 - Facilities should be provided for the subsequent re-opening of the sealed area.
-
- Open fires in an intake or return airway remote from the coal face and where there is little likelihood of flammable gas accumulations.

Fires in these situations would most likely be due to conveyors, vehicles or electrical faults. Direct attack should be adopted using fire extinguishers, water jets, foam or water spray barriers.

In a typical South African bord and pillar situation it may be possible to box in the fire by building seals inbye, outbye of the fire and in the bords on either side of the affected airways, leaving parallel airways to function normally. If this were done as a quick way of getting the fire out (not because the fire was out of control) and if there was no chance of an explosion, the stoppings would not need to be explosion-proof. The construction of the stoppings could be simple and they could be built in whatever sequence suited the practical situation. The intake stopping would probably be built first. This boxing in strategy could be combined with the use of foam.

Before deciding to box in a fire using non-explosion-proof stoppings, it would have to be ascertained that flammable gases had not been produced by the fire itself, or that, if they had, they had been mixed with products of combustion to form an inert mixture. The latter situation is usual in the case of oxygen rich fires where there is no natural emission of flammable gas. If the pillars in the vicinity of the fire were of low volatile content and were old, so that natural devolatilization had taken place, it is unlikely that the fire would have released distillation products in great measure.

If it becomes necessary to seal off the fire because it is out of control, then full sealing operations in the intakes and returns would be required. The final sealing arrangements would have to include explosion-proof stoppings, simultaneous sealing of intake and return stoppings, maintenance of adequate ventilation up to the time of sealing, remote sampling arrangements and the like. However, because there is no immediate threat of explosion, as the fire is well separated from sources of flammable gas, the intake stoppings can be quite close to the fire (assuming the fire is in an intake airway) so that depletion of oxygen is accelerated. It may also be feasible to reduce the ventilation quite drastically in the early stages of sealing. This will cause flammable gases to start accumulating at the faces, but will delay their arrival at the site of the return stoppings. The return stoppings should be well outbye of the faces so as to provide a 'cushion' between the sources of flammable gas and the stoppings.

If the fire is in a return airway, but well removed from the coal faces, the return stopping must still be well outbye the coal faces. It may be permissible to site the intake stoppings close to the faces.

- **Open fires in intake or return airways close to the coal faces and sources of flammable gas**

The situation here is much the same as for open fires at the coal face. Direct attack must be used but if this does not meet with early success, preparations for sealing must commence.

General sealing requirements are:

- **All stoppings should be well outbye of the fire and the coal faces to avoid men being exposed to the danger of explosion.**
- **The stoppings should be sited to permit ventilation up to the front wall after sealing.**
- **Adequate ventilation should be maintained through the stoppings whilst they are being built and facilities must be provided for simultaneous sealing.**
- **Arrangements must be provided for taking gas samples through the seals and measuring pressures across the seals.**
- **The stoppings should be of explosion-proof construction.**
- **Facilities for subsequent re-opening should be provided.**

(d) Goaf fires

Fires in the goaf are usually caused by spontaneous combustion or by frictional ignitions of flammable gas (especially when there is quartzitic rock in the roof). Alternatively, a fire on the coal face could spread into the goaf. On rare occasions, where a fire caused by spontaneous combustion is not too deep seated, direct attack is possible. The hot material must be excavated and wetted down or mixed with sand or stone dust before being removed from the mine. If too much air is admitted during digging out there is a danger of explosion or conflagration. While the use of water is essential, there are risks arising from its use.

- **Scalding steam can be produced.**

- Humidity can reach levels which make it impossible for work to continue.
- Water gas may be formed (this will be dealt with more fully later).

No matter how fires in the goaf originate, there is always a danger of flammable gas explosions, so direct attack is seldom wise. Trying to seal off the goaf area in isolation is also unwise because there is no 'cushion' between the probable explosion sites and the stopping sites. The two remaining options are inertization or complete sealing of the section.

- **Inertization**

This is the process of injecting inert gas into the fire area to lower the oxygen content, and in fact in some European mines it is used as a routine measure to prevent spontaneous combustion developing. A continuous bleed of inert gas (usually nitrogen) into the goaf can sometimes make the difference between mining a seam successfully and not mining it at all. Even as a fire control measure rather than a fire prevention measure, inertization has found greater use in European mines where longwall mining, with a very limited number of entries, is practised, than in the multiple entry mines of South Africa or the USA.

Inertization is used to render explosive gas/air mixtures non-explosive. This means reducing the oxygen concentration to a level below which explosion cannot take place. This level is temperature dependent and the figures shown below are for methane/air mixtures.

Temperature	Oxygen percentage by volume below which ignition cannot occur
20	12,2
100	10,8
150	10,4
200	9,9
150	9,5
300	9,0

By bringing the oxygen concentration down to these levels, rescue brigadesmen can be protected during sealing or recovery operations.

It is also used to restrict the development of the fire or to extinguish it completely. Flaming combustion is arrested first as the oxygen level

drops, followed by smouldering. The flame of a safety lamp will go out when the oxygen level drops to 17%. Flaming combustion of coal can continue down to 5% oxygen, while smouldering can continue virtually to zero oxygen. The extinction of spontaneous combustion certainly requires an oxygen concentration of less than 2%.

Carbon dioxide is used for inertization when its high density is important. It could be used to inertize an underground fire from surface as it would stay at the lowest point. It is supplied to the mine in liquid form and vaporised before use. However, carbon dioxide is soluble in water and could therefore be 'lost'. It can also be converted to carbon monoxide by contact with incandescent material, which leads to problems with toxicity and explosibility, and it is absorbed by coal.

An indirect disadvantage is that gas samples taken to indicate the state of the fire are rendered meaningless by the presence of injected carbon dioxide. The only gas which could be used as an indicator would be oxygen.

Nitrogen is the most common inert gas used in mines. The nitrogen is delivered to the mine in liquid form, and it has to be evaporated and heated to ambient temperature before being piped into the mine. If attempts were made to pump the liquid nitrogen into a borehole and straight into the mine, the thermal shock would cause the borehole casing and surrounding strata to shatter, thus blocking the borehole. The use of nitrogen also causes fire monitoring parameters to become invalid. Any parameter based on oxygen deficiency is meaningless, but the ratios CO_2/CO , CO/CH_4 or CO_2/CH_4 can be used (More of this in Appendix I).

A system available in Poland and used widely in Eastern Europe uses combustion gases in a modified jet aircraft engine and after-burner known as the GAG-3 engine. The exhaust gases are used as inert gas after being cooled by water. The engine uses 0,7 kg/s of paraffin and 19 l/s of water to produce $30 \text{ m}^3/\text{s}$ of gas with a composition of 63% water vapour, 30% N_2 , 2-5% CO_2 , 1% CO and 1% O_2 .

An airflow of 15 m³/s has to be available at the operating site, so that there is sufficient oxygen to burn the paraffin.

This system can be used where nitrogen is in short supply or where piping nitrogen underground becomes impractical because of depth (the system was used in a South African gold mine recently).

Advantages

- High flows of inert gas are available. This can be used to fight large open fires as well as fill a sealed area quickly.
- It generates up to 30 000 kW of power, which can be used to drive fuel and water pumps.
- Running costs are fairly low.

Disadvantages

- The initial cost is high.
- Large quantities of water are required.
- The hot, wet gas can cause deterioration of the strata.
- The interpretation of the fire's progress by gas analysis is virtually impossible. (Although the oxygen concentration can be used with some meaning).
- Highly specialised staff are needed to operate the system.

The engine is, in effect, a powerful and quite high capacity booster fan which can have a marked effect on the flow distribution in the fire circuit and other circuits. A high degree of ventilation engineering skill is required to ensure that these flow distribution effects are not allowed to create dangerous situations.

At the outset it must be decided whether inert gas is to be added to the air stream flowing to an unsealed fire to reduce the oxygen level (object inertization) or whether it is to be pumped into a sealed area to create an inert atmosphere (space inertization). The latter is far more common. Having decided on space inertization, it must be decided whether the objective is to extinguish the fire or to keep the gas mixture in the panel in a non-explosive state so as to afford protection to the fire fighters.

Leakage into the airstream out of the panel must be tightly controlled as excessive leakage will prevent the inert gas from reaching its target, and will impose a limit on the maximum achievable concentration of inert gas. The required flow rate of inert gas and the total quantity of gas required must be calculated in advance, and a leakage allowance must be built into the calculation.

- **Water gas reactions**

When steam is passed over incandescent carbonaceous material, chemical reactions occur which can produce mixtures of hydrogen and carbon monoxide. When produced commercially, water gas typically contains 45% H₂, 45% CO and smaller amounts of carbon dioxide, methane, nitrogen and oxygen. The lower flammable limit of water gas in air is 6-9% and the upper flammable limit is 70% (both by volume.)

The process requires a lot of heat and takes place at a temperature of 1 200°C to 1 400°C. The production of substantial quantities of water gas in a mine fire is not very likely, and a water gas explosion is less likely. The biggest danger would arise from using water on a fire in a confined space. The best practice is to use water lavishly so that if water gas is produced, there will still be surplus water to prevent or quench any ignition.

Using water sparingly in order to prevent a water gas explosion is probably the most likely way to promote one, and if water is in short supply it would be better to resort to inert gas or sealing.

- **Sealing off**

The main purposes of sealing off fires in a coal mine are twofold viz. to deny the fire sufficient air to maintain combustion and to contain any explosion which might occur. Sometimes there is little chance of an explosion occurring but a great urgency to control the fire, while at other times the opposite will apply. In most cases both purposes assume equal importance and only when the fire is safely confined behind explosion-proof stoppings can the rest of the mine resume normal operations, even though the fire may still be very active.

The explosion hazard arises from the normal emission of flammable gases near the fire, the generation of distillation substances from the coal due to the high temperature and the use of water for fire fighting purposes. The build-up of these gases must be controlled by ventilation and it is for this reason that the ventilation must be maintained at near-normal levels until sealing takes place. It is essential that the rate of emission of flammable gases is known and used in planning the sealing programme. Routine measurements of flammable gas emissions are rather rare in South African coal mines, partly because the general body concentrations in the return airway are so low that calculated figures could be inaccurate. A more realistic assessment could be made by seeing how long it takes for a routinely sealed off section to fill up with flammable gas.

An oxygen rich fire usually consumes any fire gases which are produced, but a highly developed fire, perhaps tending towards fuel rich, may not consume all the gases, but these remain downwind of the fire. Any sudden reversal of these gases could lead to an explosion, as was mentioned earlier, and is another reason for maintaining steady and adequate ventilation right up to the moment of sealing. It is also one of the reasons why the period immediately after sealing is one of the most critical in respect of an explosion inside the sealed area. Other critical periods occur as a result of the natural build-up of flammable gas and the consumption of oxygen which takes place after sealing.

Because the completion of only one of a series of stoppings can stop the ventilation, leaving the workers at the incomplete stoppings exposed to a possible explosion, it is essential that all stoppings be constructed so that the ventilation can be maintained up to the last instant, and that all stoppings are closed simultaneously. The positions of stoppings should be far enough from the fire to allow a certain dissipation of an explosion between the ignition point and stoppings. This dissipation would be brought about by stone dust and stone dust barriers. In spite of the need for stoppings to be sited in such safe positions, the area sealed off must be kept to a practical minimum, both from the point of view of not 'sterilising' too much of the mine, and to minimise the time taken for the atmosphere throughout the whole sealed off area to become extinctive. The stoppings and surrounding strata should be made as airtight as

possible (a leakage of 0,05 m³/s is more than enough to sustain a heating) and if they are not sufficiently airtight, pressure balancing should be used (See Appendix III). This is a means of bringing the faces of all the stoppings to a common pressure to minimise flows between one stopping and another. If an individual stopping breathes in or out excessively, a pressure balance chamber can be built in front of it, and the pressure inside the chamber varied to counteract the breathing in or out. Facilities for monitoring the atmosphere inside the sealed area should be incorporated in all the stoppings.

It is a distinct advantage to have partially built stoppings at pre-selected positions. These are provided most often in areas known to have a high risk of spontaneous combustion, but this need not be the only criterion. Advance planning can identify parts of the mine in which fires would be particularly disruptive. Preparations could then be made to deal very promptly with any fires which occurred at these sites. The positions of partially built stoppings should be marked on mine plans. In unforeseen circumstances new stopping positions could be substituted for the prepared ones. The exact sites of stoppings should be based on:

- The normal emission of flammable gas and the probable points of emission in relation to the position of the fire.
- The quantity and distribution of the ventilation and the likely variations which may occur during the construction of the stoppings.
- The gradients in the area, and the possible effect the fire might have on the ventilation. In particular, can the direction of flow change so as to pose a threat to the men constructing the stopping?
- The coal dust hazard. If there is risk of a coal dust explosion the stoppings may have to be moved further away from the fire.
- The length of time it would take to build the stoppings, which would be determined largely by the ease with which materials

could be brought to the site, although some materials should be left there in advance.

- The state of the ground. If it is disturbed, advance provision must be made for grouting, etc.
- The proximity of other workings, existing or abandoned and sealed.

Preparatory work would consist of selecting sites for two walls at least three metres apart if the stopping is to be concrete, barring down loose material to get to solid ground, and then partially building the walls, leaving an opening for the passage of ventilation, men and materials.

The length of an explosion-proof stopping depends on the force which the stopping will have to resist, the shear strength of the material from which the stopping is built, or the strata into which it is constructed, whichever is the lesser, and the area of contact between the stopping and the strata.

$$\text{Length} = \frac{(P_o \times A_m)}{2(w + h) \times f_s}$$

Where P_o = maximum explosion pressure (MPa)
 A_m = cross sectional area of stopping (m^2)
 w = width of stopping
 h = height of stopping
 f_s = shear strength of concrete or coal, whichever is lesser

f_s for concrete = 15 to 25 MPa

f_s for coal = 5 MPa

The shear strength for coal is used and a safety factor of 10 is applied to arrive at a design shear strength of 0,5 MPa.

The maximum pressure generated by a coal dust explosion is 0,7 MPa, but a safety factor of 3 is allowed to take account of the rapid rate of pressure build up (15 MPa/s).

$$L(\text{Length}) = \frac{H(\text{Height}) \times W(\text{Width}) + 0,6}{2}$$

subject to a minimum length of 3 m.

The length (concrete) of 6 m x 3 m stopping given by this formula would be
9,6 m.

The material sets to compressive strength of 3,54 - 4,14 MPa after two hours and 11,72 - 13,8 MPa after 24 hours.

Access tubes with armoured doors at each end are set into the stopping, allowing ventilation to pass through right up to the time of closing the seals and allowing all seals to be closed simultaneously. A sampling pipe and a pressure measuring pipe are also built into each stopping.

The usual routine is to close all the seals simultaneously and to vacate the mine for 24 hours, the period during which an explosion is most likely to take place. One of the ways of telling whether an explosion has taken place is to connect a U-tube containing mercury to the sampling pipe through the stopping. A rubber tube is then attached to the open end of the U-tube with the other end of the rubber tube placed in a bottle on the floor. If an explosion takes place the mercury in the U-tube will be displaced and will fall into the bottle.

It is usual to go back after 24 hours and do further work to improve the seals, such as grouting the strata to prevent leakage. Sampling and pressure measuring routines can be set up and, if necessary, pressure balancing measures can be taken. (See Appendix III.)

Sources of flammable gas emission would have been noted before stopping sites were finalised. These should be finally checked, if possible and decisions reached about possible layering after the ventilation is stopped. All electrical apparatus must be disconnected.

As a precaution against a coal dust explosion all conveyors should be run until empty before they are disconnected. All roadways, starting as close to the fire as possible and working outbye to the stopping sites, should be liberally stone dusted, especially in transport roadways. Stone dust close to the fire will give proportionally more protection than stone dust immediately inbye of the stoppings, which an explosion would reach only after it had built up considerable momentum. Decisions as to whether to leave doors and regulators intact in the sealed area would have to be made on an individual basis. Leaving doors open might speed up the extinguishing of the fires. Leaving them closed may help in restoring the ventilation when the time comes to re-open. Opening the doors in the connection closest to the stoppings (inside the sealed area) might help pressure balancing.

Sampling tubes should be left through each stopping and be suspended so that they do not sag and become blocked against the floor. Normally they would not extend more than a few metres inbye of the stopping. Where feasible longer tubes should be left in to sample as close to the fire as possible. If such pipes are left they should be inspected as late as possible before the seals are closed.

The need for pressure balancing should be identified soon after sealing. This can be done by means of a pressure survey taking in the faces of all the stoppings. It may be possible to deduce whether the pressures need to be balanced by measuring the pressures across ventilation doors serving the area.

If a stopping is consistently breathing in, it may be necessary to build a pressure chamber in front of it and apply suction to the pressure chamber to retard the breathing in. A stopping which consistently breathes out is not as serious unless the breathing out causes gas to be emitted at a rate which the ventilation system cannot handle. In this case a pressure chamber would be built in front of the stopping and pressurised to retard the breathing out.

If a stopping breathes the third sample out of these samples should be accepted. When sampling is conducted in the general body of air, the whole roadway must be traversed.

When samples are taken in stagnant places a probe or sampling line should be used. Apart from the safety aspect, a sampler moving around in a stagnant atmosphere can cause turbulence which will lead to constantly changing mixtures. When a sampling probe or tube is used in any situation care must be taken to ensure that the tube is fully evacuated before the final sample is taken. At least three times the internal volume of the tube must be evacuated. Samples taken to monitor fires should not be taken too close to the fire as the gas may not be completely mixed with the air. The samples should be taken at least ten roadway widths downwind of the fire. Samples at places where the direction of flow changes (borehole, stoppings) should only be taken when the flow is outwards, towards the sampler.

When rescue teams are taking samples, one man should do the sampling. If other team members are present, oxygen from their suits can 'falsify' the atmosphere.

In South African coal mines samples are taken in cylinders. A hand pump is used to pump the sample into the cylinder. As the cylinder can be pressurised a large sample can be packed into a small space.

Gas analysis is carried out in a Mobile Gas Analysis Laboratory (MOGAL) which analyses the concentrations of hydrogen, oxygen, methane, carbon monoxide and carbon dioxide. The rest of the sample is assumed to be nitrogen.

The analysing instruments are as follows:

- A single beam infra-red methane instrument with two ranges 0-10% and 0-100%.
- A single beam infra-red carbon dioxide analyser (UNOR) with ranges 0-2% and 0-20%.
- A single beam cuvette infra-red carbon monoxide instrument with ranges 0-100, 0-1 000, 0-10 000 and 0-100 000 parts per million.
- A bio-marine Draeger oxygen sensor with a range 0-25%.

- A hydrogen thermal conductivity analyzer (THERMOR) with ranges 0-10% and 0-100%.

A MOGAL has the facility to give a print out of the analysis plus the parameters required to plot the sample on the USBM Explosibility Diagram.

The MOGAL lacks the facility to analyze for the higher hydrocarbons (ethylene and propylene). This has proved to be rather a serious deficiency (See Appendix V).

It would also be an added benefit if the MOGAL were to calculate and print out the Trickett Ratio (See Appendix I). Not only does this ratio give useful information as to the state of the fire, but it is of further value because a figure greater than 1,6 is impossible. If an analysis indicates such a figure it can be concluded that the sampling routine or the analysis is incorrect and the result is invalid.

Hand held instruments can be very useful in giving instantaneous readings provided they are used correctly and are maintained and calibrated properly. The ones which incorporate electrochemical cells are not specific to one gas and can be affected by gases other than the target gas. These would include the carbon monoxide instruments, which would be affected by hydrogen, and the higher hydrocarbons. Filters can be incorporated to filter out low concentrations of the interfering gases, but this does not solve the problem entirely. Users must be aware of this limitation.

Oxygen meters must also be used with care. Some oxygen meters, and all human beings, react to the partial pressure of oxygen in the atmosphere. Although the volume per cent of oxygen remains constant at 20,93 up to an altitude of 90 km, the oxygen partial pressure reduces at higher altitude and increases at lower altitude. The oxygen partial pressure at sea level is 21,2 kPa i.e. about 21% of the atmospheric pressure (101,3 kPa). At 400 m below sea level the oxygen partial pressure will be 22,2 kPa, although the oxygen content is still 20,93% by volume. An instrument with a partial pressure sensor, calibrated at sea level, would show a reading of 22% O₂ in fresh air at 400 m depth, while an instrument with a volumetric sensor would show the correct reading of 20,93%. An

instrument with a partial pressure sensor should be reset in fresh air to 20,93% at the depth where it is going to be used. It is important for the user to know whether his instrument has a volumetric sensor or a partial pressure sensor.

Stoppings are fitted with sampling pipes so that the atmosphere in the sealed area can be monitored. This can give information as to whether the fire is dying and also whether the atmosphere is explosive. A sampling routine should be set up. Samples taken when a stopping is breathing in are of no value because they will reflect the composition of the gas which has just passed from outside the seal to the inside. The most meaningful samples are those taken at the end of a sustained period of breathing out. The greatest influence on the breathing in and out of stoppings is the barometric pressure. Stoppings will tend to breathe in when the barometric pressure rises and will tend to breathe out when it drops.

The barometric pressure on the Highveld and at most places a long way from the coast is very stable and highly predictable. It follows a sinusoidal pattern containing one falling cycle and one rising cycle in 24 hours, with one high peak and one low peak. It has been suggested that sampling times at stoppings be fixed according to the trend in barometric pressure, with samples generally being taken at the end of the downward cycle, just as the pressure flattens out and starts to rise.

However, not all stoppings react the same way to changes in barometric pressure. The flow through or around a poorly constructed stopping will reflect changes in the barometric pressure, while, in the case of a tight, well constructed stopping, what little flow does take place will lag behind the barometric pressure. Factors other than barometric pressure influence the pressure patterns at stoppings. These factors will normally be associated with the movement of vehicles in narrow airways, and should be minimal in South African coal mines because of the multiplicity of airways. The most reliable sampling practice is to take samples at fixed intervals and take pressure readings at each stopping at similar intervals. Greater credence would be given to samples taken after prolonged breathing out. Better still would be to attach the pressure line to a

recording manometer and the sampling line to a tube bundle system, if available.

Much of what follows with respect to interpreting sample results is discussed further in Appendix I. It cannot be stressed often enough that trends are always more meaningful than specific values for any gas or ratio. It is also accepted that frequent results from a few sampling points are of more value than a few scattered results from many sampling points. Frequent sampling at salient points in the mine, and the calculation of the fire indicating parameters should be carried out when there is no heating or fire, so that 'normal' values are established and all personnel concerned with 'fire watching' are alert to the values and any changes in them. Where continuous monitoring systems are in operation alert conditions, as distinct from alarm conditions, can be calculated and displayed.

The most common fire products are carbon monoxide and carbon dioxide, with hydrogen appearing at higher temperatures, particularly when water is being used on the fire. The production of carbon monoxide, quantified by multiplying the carbon monoxide concentration in an air stream by the quantity of air flowing, is a useful measure of the activity of a fire throughout its life cycle. Care should be taken when the air quantity changes. If the quantity increases and the CO production stays unchanged, the CO concentration will be reduced in relation to the increase in air quantity. If the increase in air quantity is accompanied by a reduction in static pressure, as could be expected, and the pressure change 'pulls' gas from goaf areas into the ventilation system, the CO concentration could increase temporarily, giving a false indication of an increase in CO production. When changes in air quantity take place, time should be allowed for gas levels to stabilise before new assessments are made. The Graham Ratio or CO/O₂ deficiency ratio can be used to give meaningful information. A figure of 10 denotes an open fire while figures higher than 10 could indicate that water gas is being produced. The CO/O₂ deficiency ratio can also be expected to rise and fall with increase and decrease in fire activity. There is always a problem with CO₂ in that it can be affected by agencies other than the fire, but if it can be ascertained that these other agencies are constant the CO₂ readings remain valid. This is another reason for relating readings taken during a fire to values established

in the 'no fire' situation. When nitrogen inertization is practised, all indices involving oxygen deficiency are invalid. CO production and CO₂ production will not be affected, and ratios such as CO/CH₄ and CO₂/CH₄ can be used. The last two ratios will give meaningful readings if the methane emission remains reasonably constant.

When an open fire is being fought directly in an area where there is a very low flammable gas emission, carbon monoxide and hydrogen can be produced by the action of water on the fire. In such cases the USBM Explosibility Diagram can be used and experience has shown that the R value is always close to 0,1. If this single triangle (corresponding to R = 0,1) is used, the tracing of explosibility becomes very simple, and can be used to judge when it would be appropriate to change the airflow through fire or start to use inert gas.

Immediately after a fire area is sealed the atmosphere inside the area starts to change. Since the object of sealing is to lower the oxygen content, the fire can become oxygen depleted or fuel rich. The difference between the two lies in the temperature, because if the temperature is high enough for pyrolysis to continue, fuel will still be produced. If the temperature is not high enough to support pyrolysis the fire will start to die.

Since conditions at the fire will be very turbulent, with pockets of high oxygen content even when the general oxygen level is dropping, the presence of pyrolysis produced gases can lead to an explosion. In the simplest terms, though, when the seals are closed the oxygen starts to be consumed and adsorbed, while flammable gas emission will continue. Carbon monoxide and carbon dioxide will increase initially. As the fire dies down the carbon dioxide will be increased by oxidation and decreased by adsorption onto coal and solution in water, resulting in a net decrease. As the fire zone cools further, carbon monoxide will cease to be formed and will gradually be oxidised to carbon dioxide, while hydrogen will be reduced by bacterial oxidation. The CO/O₂ deficiency ratio (Graham Ratio) is regarded as an indicator of whether a sealed fire has died out, even though Graham only intended it to be an indicator of the development of spontaneous combustion.

Oxygen is an almost universal indicator as to the state of a fire. It is generally accepted that active flaming combustion ceases when the oxygen concentration falls below 12%, that slow combustion can be sustained by 5% and smouldering can continue in 1% to 2% oxygen. Readmission of oxygen can lead to a resurgence of combustion even after the oxygen has been less than 2% for long periods. Appendix I contains more information on these matters.

Before sealing is complete, the atmosphere will consist of fresh air plus products of combustion, flammable gases and inert gases, but the biggest component will be fresh air. Immediately after sealing the consumption of oxygen will result in the replacement of air by inert gases, while the emission of flammable gas will displace both the air and inert gas. The result will be a changing mixture of air, inert gas and flammable gas. If the oxygen is consumed rapidly the mixture may never become explosive but if the build-up of flammable gas is rapid, the mixture may well become explosive.

These mixtures of air, inert gas and flammable gas, and the changes they undergo, can be displayed on explosibility diagrams. Two such diagrams are in use in the mining world, the Coward Diagram or its derivatives, and the USBM Diagram. The USBM Diagram is preferred in South Africa and the USA but the Coward Diagram seems to be preferred in all other countries. Either of the two diagrams can be used to show whether the atmosphere is explosive, or whether it is tending towards explosibility.

A method has been developed to produce time related trend diagrams to show whether or not an atmosphere is tending towards explosibility and if so, when it is likely to become explosive. These time related trend diagrams can be derived from Coward Diagrams or USBM Diagrams, and the final diagram appears in the same form in either case.

The process now needs to be computerised for easy use. All these matters are dealt with in full in Appendix II, and in a paper entitled 'A Re-look at Explosibility Diagrams' by W Holding presented at the Fifth International Mine Ventilation Congress' held in Johannesburg in October 1992.

- **Extinction and reopening**

It is difficult to be certain when a fire is out. Because the strata in a coal mine generally consist of good insulators, the heat generated by a fire takes a long time to disperse and an oxygen content of 1% to 2% can maintain a heating almost indefinitely. It cannot be known how far the fire spread after sealing and very hot spots may remain which can start up a fire quickly or ignite flammable gas.

Fires involving large masses of material not freely open to the mine atmosphere may take years rather than months to cool. Fires closer to the roadways would cool more quickly. Some attempts have been made to cool the fire area artificially by bleeding air out, cooling it and bleeding it back in, adding nitrogen in the process, but this does not have wide application.

If there is no pressing need to reopen a section, it should not be attempted. However, if it is important to reopen, it should not be attempted until conditions have been completely stable for at least one month and preferably about three months in the case of small fires, and much longer where the fire was large.

Assessments must be made of roof conditions, as far as possible, as a fire which has been buried will take longer to cool down. One slight advantage enjoyed when reopening is that, if there is a resurgence of the fire, the fire fighters will know, in general terms where it is going to be, and they can make their preparations accordingly. There are two basic methods of re-establishing the ventilation around a sealed area. The first method is to re-establish the ventilation in total and in one step, and the second is to re-establish the circuit in steps. Direct re-establishment would be used where the likelihood of rekindling the fire is not very high. The access tubes of one intake stopping and one return stopping would be opened. A forcing fan would be connected to the access tube of the intake stopping, with facilities for varying the air quantity. The fan would be started to re-establish the ventilation, with arrangements for monitoring flow and gas concentrations at the return stopping. The principal gases to be monitored would be flammable gas (methane) and carbon monoxide. The flow would be regulated so that the flammable gas content in the

'receiving return airway' did not exceed 1,4%. The operation would be carried out at week end.

The gases from the sealed area would probably come out as a plug, after which the concentration would 'tail off'. Once the concentration had stabilised a decision would be made either to increase the flow to lower the concentration further or to start an inspection of the section.

The carbon monoxide production would be logged, and if there was any sign of a resurgence in combustion, fire fighting teams would enter the section to deal with the situation. It would be important to know what the carbon monoxide production due to oxidation had been when the section was working normally.

If there was no resurgence of the fire and an inspection was carried out, the section would be made safe and all flammable gas layers would have to be dispersed before full and normal ventilation could be restored.

For step by step reopening the access tubes of one intake stopping and one return stopping would be opened and a rescue team would go through, closing the access tubes behind them. They would travel to the first connection between intake and return. They would build temporary stoppings inbye of the connection, then open the doors in the connection and open the access tubes in the main stoppings. A ventilation circuit would therefore be established as far as the first connection. This process would continue step by step until a ventilation circuit had been formed around the whole section. The section would be made safe and flammable gas roof layers would be dispersed. Full ventilation would then be re-established.

Fire fighting equipment would be kept on hand, and if there was a resurgence of the fire it would either be dealt with on site, or the team would retreat and the section would be resealed.

4.5 Final Comment

Some of the information contained in this report is based on experience, but a great deal of it has been derived from other sources. A study of these sources would obviously add greatly to the value of the report.

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APPENDIX 1

FIRE INDICATORS

The main indicators of underground fires are gases and smoke. The gases can be divided into several groups. The first group consists of the distillation products, that is, gases that are present in coal and are emitted naturally, but which are liberated more freely as the temperature rises. The principal gases in this group are methane (CH_4), ethane (C_2H_6), and propane (C_3H_8) and their concentrations increase until the temperature exceeds 150°C . At this point flaming combustion starts and the distilled gases are consumed, lowering their concentrations. The second group of gases consists of combustion products such as carbon dioxide (CO_2), carbon monoxide (CO), hydrogen (H_2), ethylene (C_2H_4), propylene (C_3H_6) and sometimes acetylene (C_2H_2). Carbon dioxide is a product of complete combustion and is predominant only when the fire has reached an advanced stage, while the others are products of incomplete combustion and are therefore useful indicators of the early stages of spontaneous combustion. As oxidation accelerates into the first stage of spontaneous combustion and then accelerates further as the temperature increases, carbon monoxide is produced first, followed by hydrogen, ethylene, propylene and acetylene in that order. The presence of these gases can therefore be used as an indication of the temperature of the heating. In the gases from an active heating there will always be more CO than H_2 , and more H_2 , than the hydrocarbons C_2H_4 , C_3H_6 and C_2H_2 . When the oxygen becomes depleted to 2% or less, the H_2 can fall below the level of the hydrocarbons. The above is based on the work of Chamberlain and is illustrated in Figure I.1.

The third group of gases comprises the constituents of the air itself viz. oxygen, nitrogen and carbon dioxide. (The other trace gases are assumed to be part of the nitrogen). As oxygen is consumed by oxidation and combustion, the normal proportions of 20,93% O_2 , 19,04% N_2 and 0,03% CO_2 become distorted and an excess of N_2 and CO_2 , relative to the depleted O_2 , is created. This mixture of excess N_2 and excess CO_2 is known as black damp (because it is extinctive). Since slow oxidation of carbonaceous material is a normal feature of every coal mine, blackdamp is almost always present and has led to many a fatality.

When oxidation speeds up to become spontaneous combustion and open fire, the proportions of the gases change more rapidly and much useful information can be gathered from following the trends in concentrations of these gases.

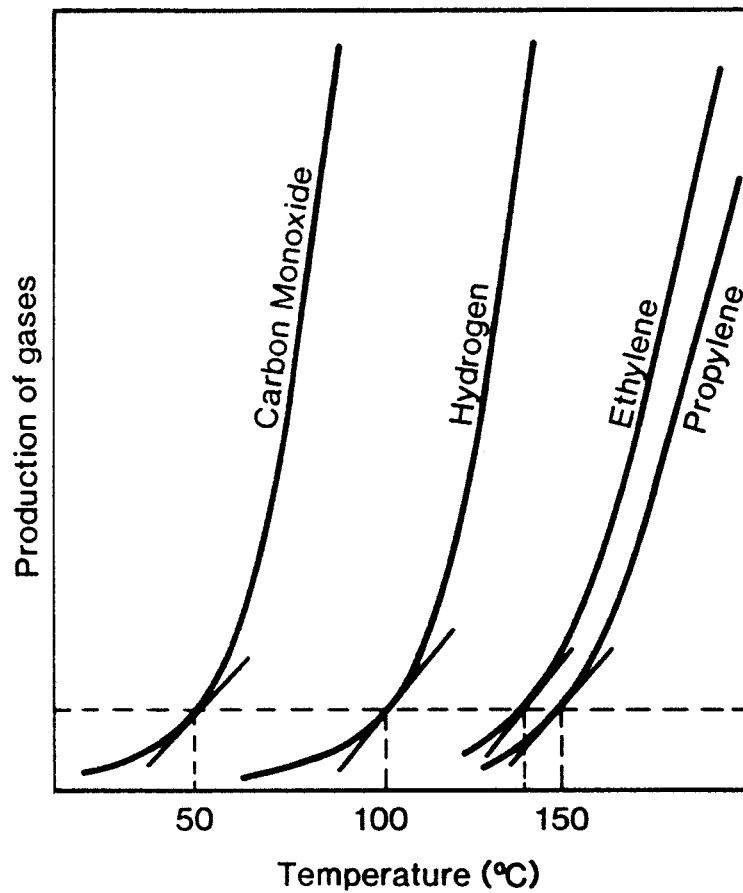


Figure I.1 Production of gases during active heating

Since N_2 is neither consumed nor created in the combustion process, it provides a steady base with which the other gases can be compared. Trends in the ratio of N_2 to other gases can, therefore be useful.

The whole situation can be complicated by the fact that some of the gases used as fire indicators sometimes occur due to causes completely unrelated to the fire. CO_2 is sometimes released from coal seams continuously in the same way as CH_4 . It can also be produced by the action of acid water on carbonates. On the other hand it can be depleted by dissolving in water. In some cases N_2 has been known to be emitted from coal seams in small quantities, while CO is a product of diesel engines and the use of explosives.

A further point to remember is that conclusions drawn from a study of fire gases are based on combustion theory, which in turn is based on the assumption that the air feeding the

combustion process is normal. If this is not true, particularly if the air is already deficient in oxygen, care must be taken in interpreting trends, especially in making decisions based on indices involving oxygen deficiency e.g. Graham Index.

Smoke is also a sensitive fire indicator. In the case of fires other than those due to spontaneous combustion, smoke is a more effective early indicator than carbon monoxide.

In tests carried out locally, diesel engine and conveyor belting burning at a rate of 800 g per minute was detected by a Beacon ionisation detector at a distance of 2 500 m. The carbon monoxide concentration was 9 ppm, while most carbon monoxide sensors have a background 'noise' signal level corresponding to about 3 ppm. It is unlikely that a CO sensor would react reliably to such a small increase in concentration above the background 'noise' level.

When a fire is inaccessible or too small to exhibit discernible physical signs, the composition of the gases produced by the fire can be used to interpret its state. While the simple analysis of the gas composition and trends in the composition can yield some useful information, some workers have used the ratio of one gas to the others as an interpretative tool.

$$\text{Graham Index} = \frac{100 \text{ CO}}{0,265\text{N}_2 - \text{O}_2} \text{ per cent}$$

As oxidation proceeds carbon monoxide is produced and the ratio of carbon monoxide produced to oxygen consumed increases. This continues through the oxidation stage to the heating stage to open combustion. At this stage carbon dioxide becomes the dominant gas and the Graham Index can actually decrease. The index is not affected by dilution with air, but oxygen deficiency ($0,265\text{N}_2 - \text{O}_2$) values of 0,2 or less reduce the reliability of the reading because a small increase in carbon monoxide can cause an exaggerated increase in the index. The index is not valid if the air feeding the heating is itself deficient in oxygen.

Typical values are:	0,4 or less	-	normal
	0,5	-	check the situation
	1	-	probable heating
	2	-	serious heating
	3	-	open combustion

It is better to observe trends than absolute values as the characteristics of coal seams differ. The best procedure is to establish 'normal' figures at selected points in the mine and be alert to changes from the 'norms'.

The Graham Index can be used under some circumstances, to indicate the state of a sealed off fire. If the remaining carbon monoxide is absorbed or oxidised after the fire has stopped producing carbon monoxide, the Graham Index will diminish rapidly once combustion has ceased.

There are coal seams in which the carbon monoxide is not absorbed or adsorbed when combustion ceases, and the Graham Index remains high. This is documented by Willett (1951). In such circumstances carbon dioxide can be used as an indicator as to whether the fire is out or not. This is complicated by the fact that carbon dioxide often occurs as a natural seam gas in the same way as methane. If there is not fire, carbon dioxide will maintain a constant relationship with the other naturally occurring gases. If there is a fire, extra carbon dioxide will be produced and the relationship will change. This is the basis of Willett's Index.

$$\begin{aligned} \text{Willett's Index} &= \frac{100 \text{ CO}_2}{\text{Excess N}_2 + \text{Total Combustibles} + \text{CO}_2} \text{ per cent} \\ &= \frac{100 \text{ CO}_2}{(\text{N}_2 - 3,78\text{O}_2)(\text{CH}_4 + \text{H}_2 + \text{CO}) + \text{CO}_2} \end{aligned}$$

There are no universal 'typical' values by which Willett's Index can be interpreted. It is prudent to establish trends and values for Willett's Index in sections sealed off in the normal course of mining and then use these values for guidance when sections in the same seam and locality are sealed off to extinguish fires.

The burning of 1 kg of coal produces, typically 0,031 m³ (31 litres) of carbon monoxide, so that measuring the air flow rate and the concentration of carbon monoxide, to calculate the production rate of carbon monoxide, is an indirect method of measuring the rate of combustion. A carbon monoxide production rate of > 10 litres per minute is a cause for concern, while > 20 litres per minute is a definite sign of spontaneous combustion.

As in the use of the Graham Index, care must be taken to ensure that there is no carbon monoxide in the air feeding the heating, or if there is, to take account of it. This would be done by calculating the Graham Index on the intake side and the return side of the suspected fire or heating, thus arriving at the increase in Graham Index across the heating. The same routine could be applied to carbon monoxide production.

The Trickett Index is based on the calculation of the theoretical gaseous products from a pure methane explosion and a 'pure' coal dust explosion. A pure methane explosion would result in a Trickett Index of 0,5 while a 'pure' coal dust explosion would result in a Trickett Index of 0,87. The original intention of Trickett and Jones (1954) in carrying out this work was to be able to distinguish between a methane explosion and a coal dust explosion.

If there is no fire, carbon dioxide will maintain a constant relationship with the other naturally occurring gases. If there is a fire however, further work on the Trickett Index arising from fires as well as explosions has led to the following:

For explosions:

Trickett Index =	0,5	methane is the only fuel
=	0,87	coal dust is the only fuel
=	0,5 - 0,87	methane and coal dust are both involved

For fires:

Trickett Index =	< 0,4	no fire, gases probably residual
=	0,4 - 0,54	methane is the only fuel
=	0,55 - 0,8	methane and coal involved
=	0,8 - 1,0	coal is the only fuel
=	> 1,0	wood is the probable fuel
=	> 1,6	impossible

The fact that it is impossible for the Trickett Index to exceed 1,6 makes it a very useful tool for checking whether sampling and analysis procedures have been carried out correctly. It is therefore worthwhile calculating the Trickett Index as a 'litmus test' of these procedures.

$$\text{Trickett Index} = \frac{\text{CO}_2 + 0,75 \text{ CO} - 0,25 \text{ H}_2}{0,265 \text{ N}_2 - \text{O}_2}$$

Or, if ethylene (C₂H₄) or ethane (C₂H₆) are present:

$$\text{Trickett Index} = \frac{\text{CO}_2 + 0,75 \text{ CO}_2 - 0,25 \text{ H}_2 + 0,5 \text{ C}_2\text{H}_4 + 0,25 \text{ C}_2\text{H}_6}{0,265 \text{ N}_2 - \text{O}_2}$$

The Morris Index can be used to estimate the temperature of a heating. The index increases as the temperature increases, reaching a peak at a temperature of 100°C, after which it decreases with further increase in temperature, levelling off at a temperature of 250°C. It is important to trace the trend from the beginning as a particular value can occur on the upward cycle at a fairly low temperature, and will recur on the downward cycle at a considerably higher temperature. This index is not valid if nitrogen inertisation is being used to suppress the fire.

$$\text{Morris Index} = \frac{\text{N}_2}{\text{CO} + \text{CO}_2} \text{ per cent}$$

The CO/CO₂ ratio is of particular value where nitrogen inertisation is taking place. It rises rapidly up to the temperature at which flaming combustion commences after which it remains constant as the temperature increases further. The actual values are less than unity.

Sometimes the reciprocal is used, in which case the values are greater than unity and the trend in values with increasing temperature is reversed. The results can be spurious if either of the gases is from a source other than a fire.

The following ratios are also useful indicators:

$$\frac{\text{CH}_4}{\text{C}_2\text{H}_6 + \text{C}_3\text{H}_8} = \frac{\text{Methane}}{\text{Ethane} + \text{Propane}}$$

In the early stages of a heating the emission of ethane and propane increases relative to methane, due to distillation, so the ratio will decrease. When the fire is diminishing the ratio will increase and when the fire is out it will stabilise.

When nitrogen inertisation is taking place either of these ratios can be used:

$$\frac{\text{CO}}{\text{CH}_4} \text{ or } \frac{\text{CO}_2}{\text{CH}_4}$$

When the readings are completely distorted, for instance, by inertisation with CO₂, the only meaningful reading is the oxygen content. Any value above 2% means the fire can still be active. 1% - 2% can sustain spontaneous combustion almost indefinitely.

APPENDIX II

SAMPLES FROM SEALED AREAS - THE PRODUCTION OF TIME RELATED EXPLOSIBILITY TRENDS

The explosibility characteristics of the atmospheres in sealed areas are usually displayed on a Coward diagram or a United States Bureau of Mines Explosibility Diagram (USBM diagram). Throughout the mining world the Coward diagram is by far the most commonly used. In fact it would appear that the USBM diagram is never used outside the United States of America and South Africa.

No matter which diagram is used the procedure is the same. The position of the sample is plotted on the diagram, and this position is noted relative to an explosibility triangle. A series of samples can be plotted to see if the atmosphere is explosive, moving towards explosibility or moving away from explosibility. There are two complicating factors. Firstly, the position of a sample is specific to the constituents of the sample, but the co-ordinates of the explosibility triangle are also specific to the constituents of the sample. If a series of samples is plotted, not only does the position of the sample change from one sample to the next, but the shape and position of the explosibility triangle also change from one sample to the next. The trend in the samples therefore has to be related to a 'moving target'. Secondly, because there is no time scale on the diagram, the rate of change of a series of samples towards or away from explosibility can only be judged by reference to separate time data.

The procedures set out below describe how to produce time related diagrams. In the case of Coward diagrams, a calculation method and a graphical method of establishing the co-ordinates of the explosibility triangle are proposed, while in the case of the USBM diagram, these co-ordinates are part of the diagram. The calculation method for establishing the co-ordinates of a Coward explosibility triangle can be computerised quite readily.

Coward Diagram

Calculation of the co-ordinates of the explosibility triangle

Figure II.1 shows a Coward diagram. The explosibility triangle is shown as PQR. P is the lower flammability limit, Q is the upper flammability limit and R is the nose limit, and each of these points can be defined in terms of its oxygen content and its combustible gas content. However, since P and Q are always on the line AB which represents mixtures of fresh air and combustible gas, they can be fixed in terms of their combustible gas contents only.

The combustible gas in a mine sample is usually assumed to consist of methane (CH₄), hydrogen (H₂) and carbon monoxide (CO). Table II.1 shows the lower and upper flammability limits and nose limits for these gases individually.

Table II.1 Flammable and nose levels for CO, H₂ and CH₄

Gas	Flammable Limits (Combustible Gas %)		Nose Limits	
	Lower	Upper	Combustible %	Oxygen %
CO	12,5	74	13,78	6,08
H ₂	4,1	74	4,3	5,12
CH ₄	5,0	14	5,93	12,24

The figures for gas mixtures can be calculated from the following:

$$\text{Lower Limit for Mixture (LL)} = \frac{100}{\frac{100 \text{ CO}}{(\text{CO} + \text{H}_2 + \text{CH}_4) \times 12,5} + \frac{100 \text{ H}_2}{(\text{CO} + \text{H}_2 + \text{CH}_4) \times 4,1} + \frac{100 \text{ CH}_4}{(\text{CO} + \text{H}_2 + \text{CH}_4) \times 5}} = \% \text{ combustible}$$

(12,5 = LL for CO) (4,1 = LL for H₂) (5 = LL for CH₄)

$$\text{Upper Limit for Mixture (UL)} = \frac{100}{\frac{100 \text{ CO}}{(\text{CO} + \text{H}_2 + \text{CH}_4) \times 74} + \frac{100 \text{ H}_2}{(\text{CO} + \text{H}_2 + \text{CH}_4) \times 74} + \frac{100 \text{ CH}_4}{(\text{CO} + \text{H}_2 + \text{CH}_4) \times 14}} = \% \text{ combustible}$$

(74 = UL for CO) (74 = UL for H₂) (14 = UL for CH₄)

Nose Limit (Combustible)

$$\text{for Mixture (NL - COMB)} = \frac{100}{\frac{100 \text{ CO}}{(\text{CO} + \text{H}_2 + \text{CH}_4) \times 13,78} + \frac{100 \text{ H}_2}{(\text{CO} + \text{H}_2 + \text{CH}_4) \times 4,3} + \frac{100 \text{ CH}_4}{(\text{CO} + \text{H}_2 + \text{CH}_4) \times 5,93}} = \% \text{ combustible}$$

(13,78 = NL - COMB for CO) (4,3 = NL - COMB for H₂) (5,93 = NL - COMB for CH₄)

LL and UL i.e. points P and Q in Figure II.1 have now been ascertained. The % combustible of point R i.e. (NL-COMB) has also been ascertained. It now remains to calculate the oxygen content at R i.e. (NL-oxygen).

Table II.2 shows the volumes of nitrogen required to render the individual flammable gases extinctive.

Table II.2 Volumes of nitrogen needed for extinguishing different flames

Flammable Gas	Extinctive Volume of Nitrogen
CO	4,15 volumes
H ₂	16,55 volumes
CH ₄	6 volumes

$$\% \text{ Excess N}_2 \text{ at Nose Limit} = \text{NL} - \text{COMB} \times \frac{4,5 \times 100\text{CO}}{\text{CO} + \text{H}_2 + \text{CH}_4} + \frac{16,55 \times 100\text{H}_2}{\text{CO} + \text{H}_2 + \text{CH}_4} + \frac{6 \times 100\text{CH}_4}{\text{CO} + \text{H}_2 + \text{CH}_4}$$

% Excess N₂ at Nose Limit can now be written as % O₂ (at the Nose Limit) i.e. NL-oxygen

$$\% \text{ O}_2 = 0,2093 (100 - \% \text{ excess N}_2 \text{ at Nose Limit} - \text{NL-Comb}).$$

- NOTES
- 1 If other flammable gases are present e.g. ethane, they can be accounted for in the same way as CO, H₂ and CH₄.
 - 2 The above method assumes that the CO₂ is equivalent to N₂ and the two gases are added together as a single figure for N₂. This is a slight error on the safe side.

While the above calculation is tedious it consists only of addition, subtraction, multiplication and division, and is easily handled by a computer or programmable calculator. See example below.

Sample analysis

O₂ - 14% CO - 1% CO₂ - 5,7% CH₄ - 0,22% H₄ - 2,46%

$$\begin{aligned} \text{Lower Explosive Limit} &= \frac{100}{\frac{100 \times 1}{(1 + 2,46 + 0,22) \times 12,5} + \frac{100 \times 2,46}{(1 + 2,46 + 0,22) \times 4,1} + \frac{100 \times 0,22}{(1 + 2,46 + 0,22) \times 5}} \\ &= \frac{100}{2,1739 + 16,304 + 1,195} \\ &= 5,083\% \text{ combustibles} \end{aligned}$$

$$\begin{aligned} \text{Upper Explosive Limit} &= \frac{100}{\frac{100 \times 1}{(1 + 2,46 + 0,22) \times 74} + \frac{100 \times 2,46}{(1 + 2,46 + 0,22) \times 74} + \frac{100 \times 0,22}{(1 + 2,46 + 0,22) \times 14}} \\ &= \frac{100}{0,367 + 0,9033 + 0,427} \\ &= 58,9\% \text{ combustibles} \end{aligned}$$

$$\begin{aligned} \text{Nose Limit Comtustibles} &= \frac{100}{\frac{100 \times 1}{(1 + 2,46 + 0,22) \times 13,78} + \frac{100 \times 2,46}{(1 + 2,46 + 0,22) \times 4,3} + \frac{100 \times 0,22}{(1 + 2,46 + 0,22) \times 5,93}} \\ &= \frac{100}{1,972 + 15,55 + 1,008} \\ &= 5,39\% \end{aligned}$$

$$\begin{aligned} \% \text{ Excess N}_2 \text{ at Nose Limit} &= 5,39 \times \frac{\frac{4,15 \times 100 \times 1}{1 + 2,46 + 0,22} + \frac{16,55 \times 100 \times 2,46}{1 + 2,46 + 0,22} + \frac{6 \times 100 \times 0,22}{1 + 2,46 + 0,22}}{100} \\ &= 5,39 \times \frac{112,77 + 1106,33 + 35,87}{100} \\ &= 5,39 \times 12,5497 \\ &= 67,64 \end{aligned}$$

$$\begin{aligned} \% \text{ O}_2 \text{ at Nose Limit} &= 0,2093(100 - 67,64 - 5,39) \\ &= 0,0293 \times 26,97 \\ &= 5,65\% \end{aligned}$$

Alternatively a set of graphs devised by Hughes and Raybould (1960) can be used. See Figures II.2 - II.5.

Time related explosibility trends

Time related explosibility trends can be produced by transferring data from a series of Coward triangles onto a single diagram with a time scale. The procedure is based on the following:

When the position of a sample and the explosibility triangle relating to the sample constituents have been fixed on the Coward diagram, adding or subtracting combustible gas of the same composition as the combustible gas in the sample will cause the sample position to move along a line joining the original sample position to the 100% combustible gas point (B). This line can be extended across the diagram to intersect the explosibility triangle at two points, or to intersect the line separating the 'non-explosive' segment from the 'explosive when mixed with air' segment, at one point. The combustible gas percentage at these points and the sample position can be read off the diagram and transferred to a new diagram with combustible gas content on the Y-axis. Similarly, if the air content of the sample changes, keeping all else constant, the sample position will move along a line joining the original sample to the fresh air point (A). This line can also be extended across the diagram to intersect the explosibility triangle at two more points. These points can also be transferred to the new diagram.

A series of samples can be plotted on Coward diagrams and the data transferred on to the new diagram. This series of samples would appear on the new diagram with time on the X-axis. The diagram would then portray the trend in the samples and their explosibility characteristics.

Figures II 6(a) to (f) illustrate the whole procedure. For the sake of simplicity it has been assumed that the co-ordinates of the explosibility triangle do not change throughout the series. The method works equally well if they do change.

USBM Diagram

When the USBM Diagram is used, the explosibility triangles for various gas mixtures are part of the diagram, so there is no need to calculate the triangle co-ordinates. This is a great advantage, but the diagram is complex to look at and even more complex to work with. The most important draw-back, however, is that the USBM Diagram is based on lower limits of flammability only. The line portraying the upper flammability limit is fictitious, although it covers the vast majority of practical circumstances. When Zabetiaklis and his co-authors introduced the USBM Diagram in 1959 they stipulated that the Diagram should not be used if the H_2 in the sample exceeds 5% and the CO in the same sample exceeds 3%. In such circumstances the Coward diagram would display the correct upper flammability limit, while the USBM Diagram would not.

Nevertheless, the USBM Diagram finds great favour in South Africa, and understandably so. It can be used to generate time related explosibility trends in the same way as the Coward Diagram. Figures II 7(a) to (f) illustrate the procedure.

An explosibility triangle has been used throughout corresponding to an R value of approximately 0,3 but the procedure would work equally well if the R value varied from one diagram to the next.

If the air content of the atmosphere is changed, the point representing the sample will move along a line joining its original position to the fresh air point (the origin of the graph). Extending this line across the explosibility triangle will fix two points of the triangle.

If the effective combustible content of the atmosphere is changed the point representing the sample will move from its original position to the apex of the diagram (off the scale) representing 100% effective combustibles. Extending this line across the explosibility triangle will fix two more points on the triangle. All these points, including the sample point, can be expressed in terms of their effective combustible gas contents and can be transferred to the vertical scale at the right hand side of the diagram. Figure II 7(a) to (f) shows the

sequence of the five samples (a) to (e) and the 'explosibility envelopes' generated by the procedure described. It will be noted that in the case of sample (c), the line joining the sample to the 100% effective combustibles point does not intersect the explosibility triangle, but it does intersect the boundary line between 'non-explosive mixtures' and 'explosive when mixed with air'. The position of sample (e) is such that neither the line joining the sample to the 100% effective combustible point nor the line joining the sample to the fresh air point, intersect the explosibility triangle.

Figure II.8 shows a sequence of samples taken at an actual fire, plotted on a composite time related trend diagram. A comparison is made between the diagram generated from Coward triangles and the diagram generated from USBM diagrams. It will be noted that while the sample positions and their explosibility characteristics are almost identical, the upper flammability limits generated from Coward diagrams are much higher than those generated from USBM diagrams. Where upper flammability limits are important, it is essential that Coward triangles are used.

The procedure outlined in this Appendix for the generation of 'explosibility envelopes', viz. plotting lines of changing air content or lines of changing combustible gas content, is an effective device, but in practice it is almost certain that both processes will be going on at the same time. Since, however, the two 'envelopes' overlap to a large extent, the procedure is valid for all practical purposes.

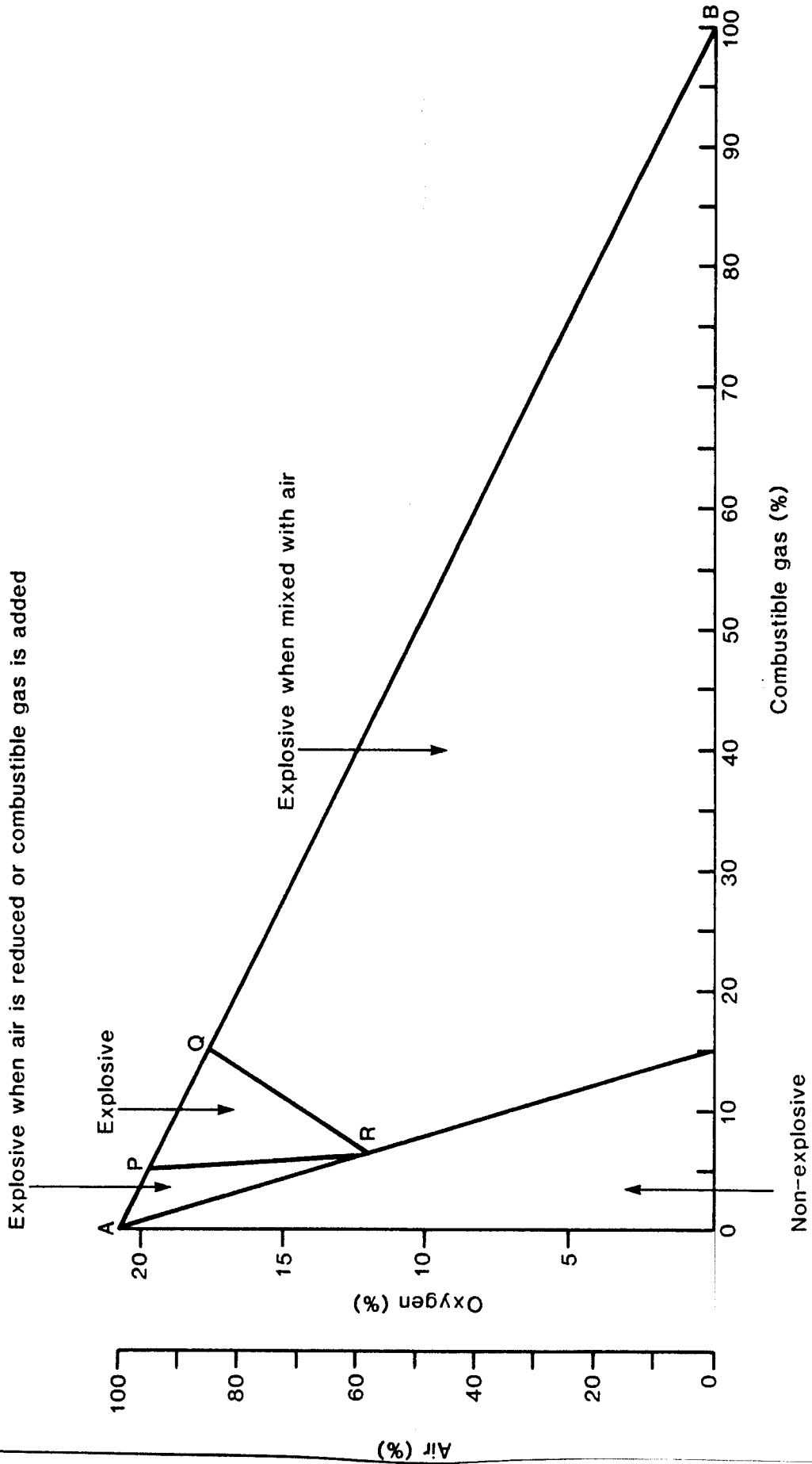


Figure II.1 Showing the Coward Diagram

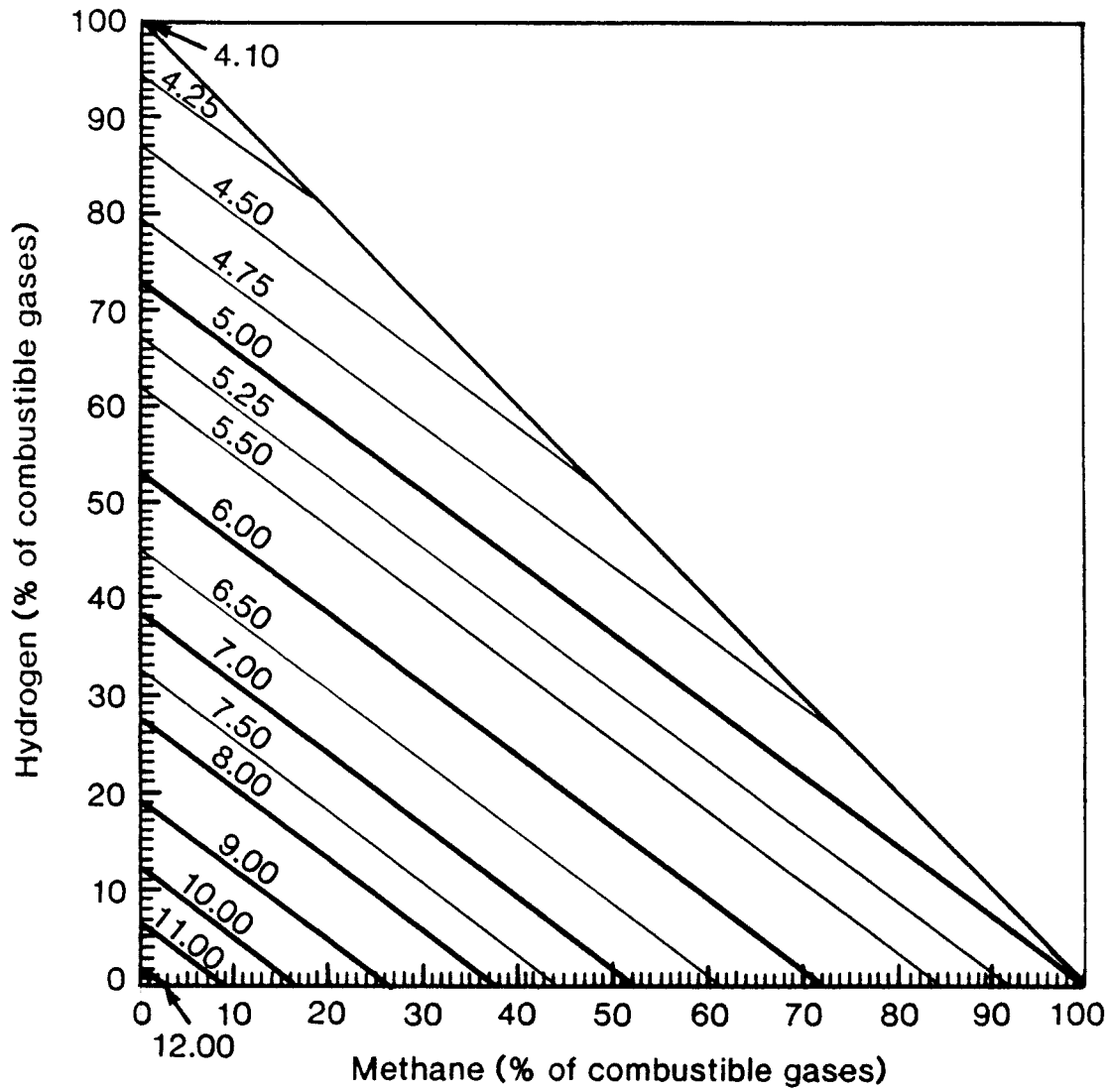


Figure II.2 Combustible content at the lower explosive limit of mixtures of CH₄, H₂ and CO in air. Carbon monoxide % = 100 - (CH₄ + H₂%)

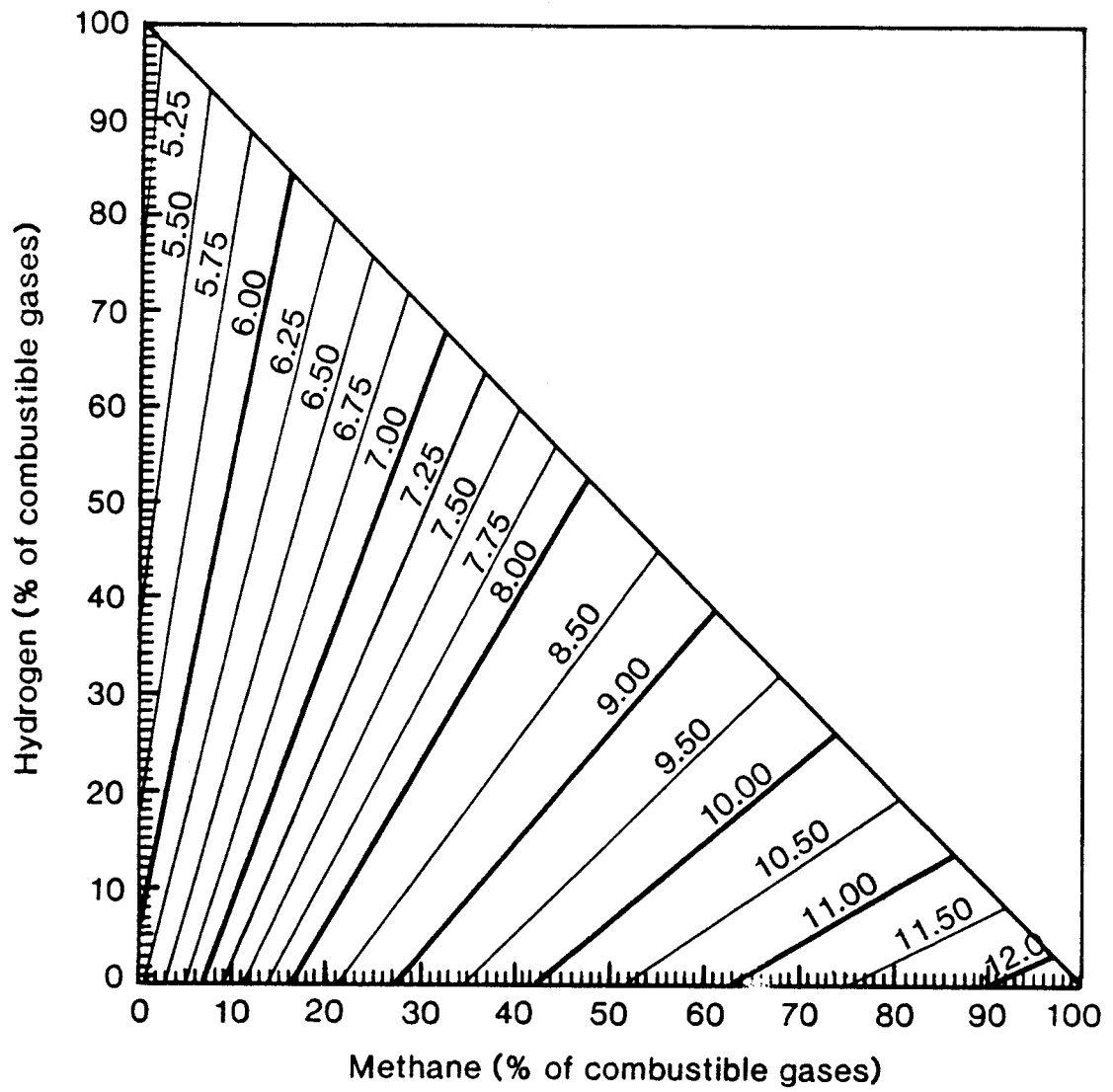


Figure II.3 Oxygen content at the nose limit of mixtures CH_4 , H_2 and CO in air and excess N_2 . Carbon monoxide % = $100 - (\text{CH}_4 + \text{H}_2\%)$

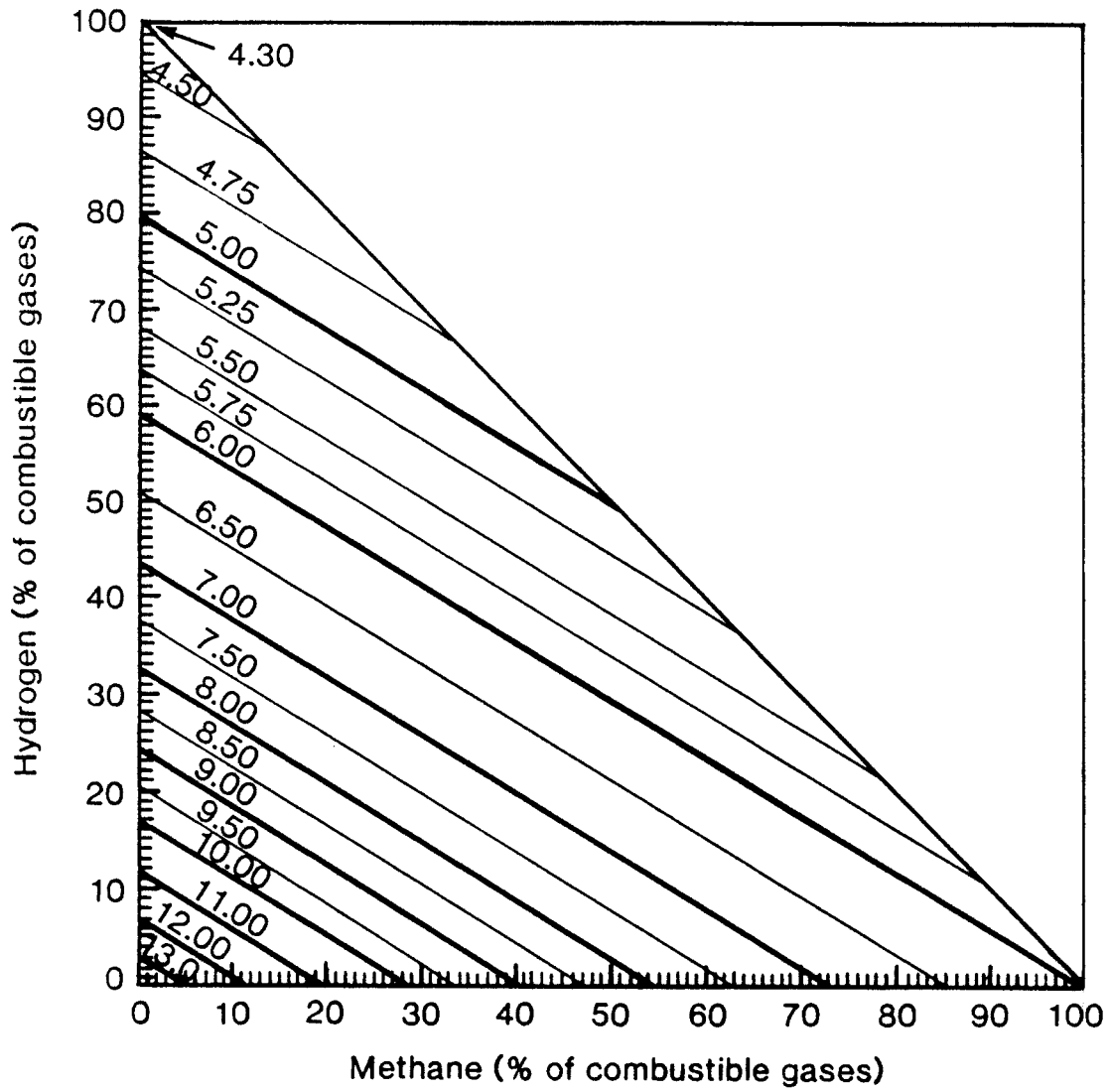


Figure II.4 Combustible content at the nose limit of mixtures of CH₄, H₂ and CO in air and excess N₂. Carbon monoxide % = 100 - (CH₄ + H₂%)

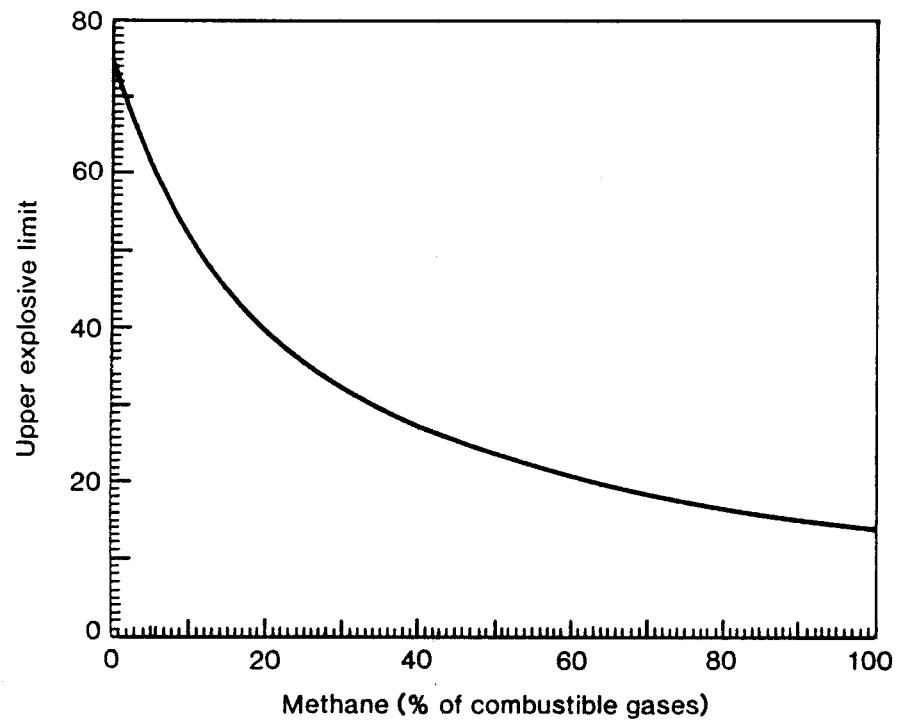


Figure II.5 Combustible content at the upper explosive limit of mixtures of CH₄, H₂ and CO in air

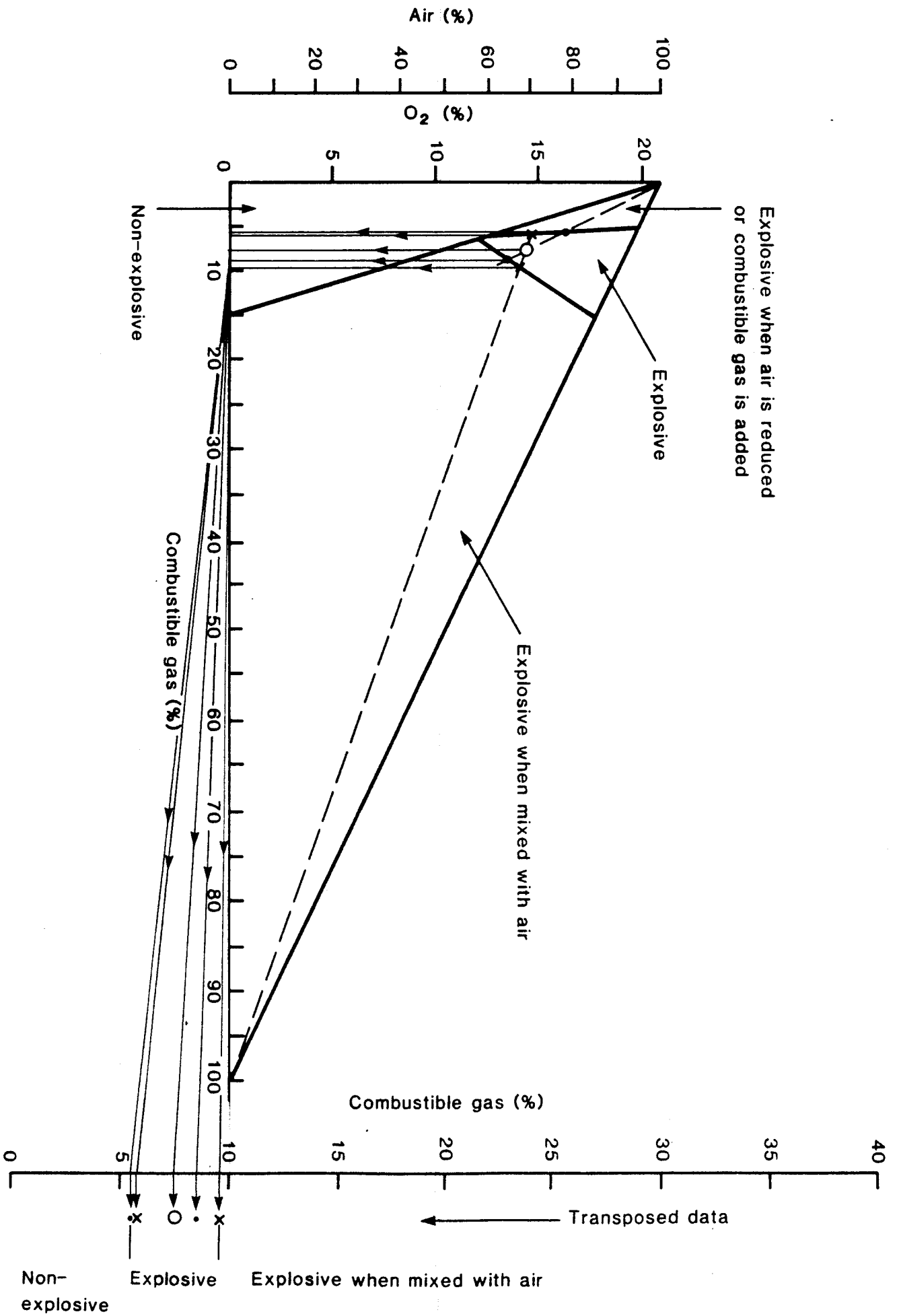


Figure II.6(b) Plotting sample 2 on the Coward Triangle

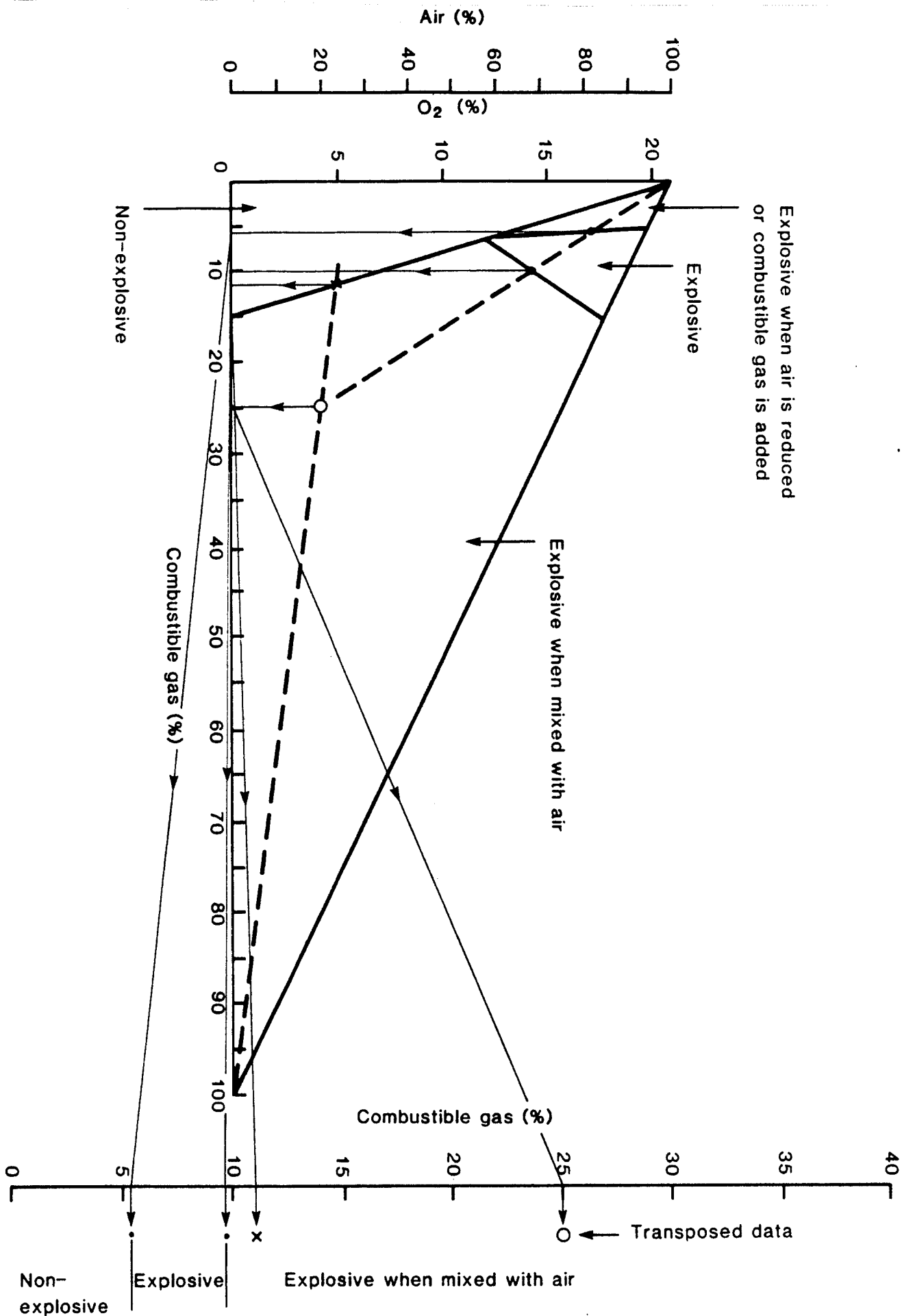


Figure II.6(c) Plotting sample 3 on the Coward Triangle

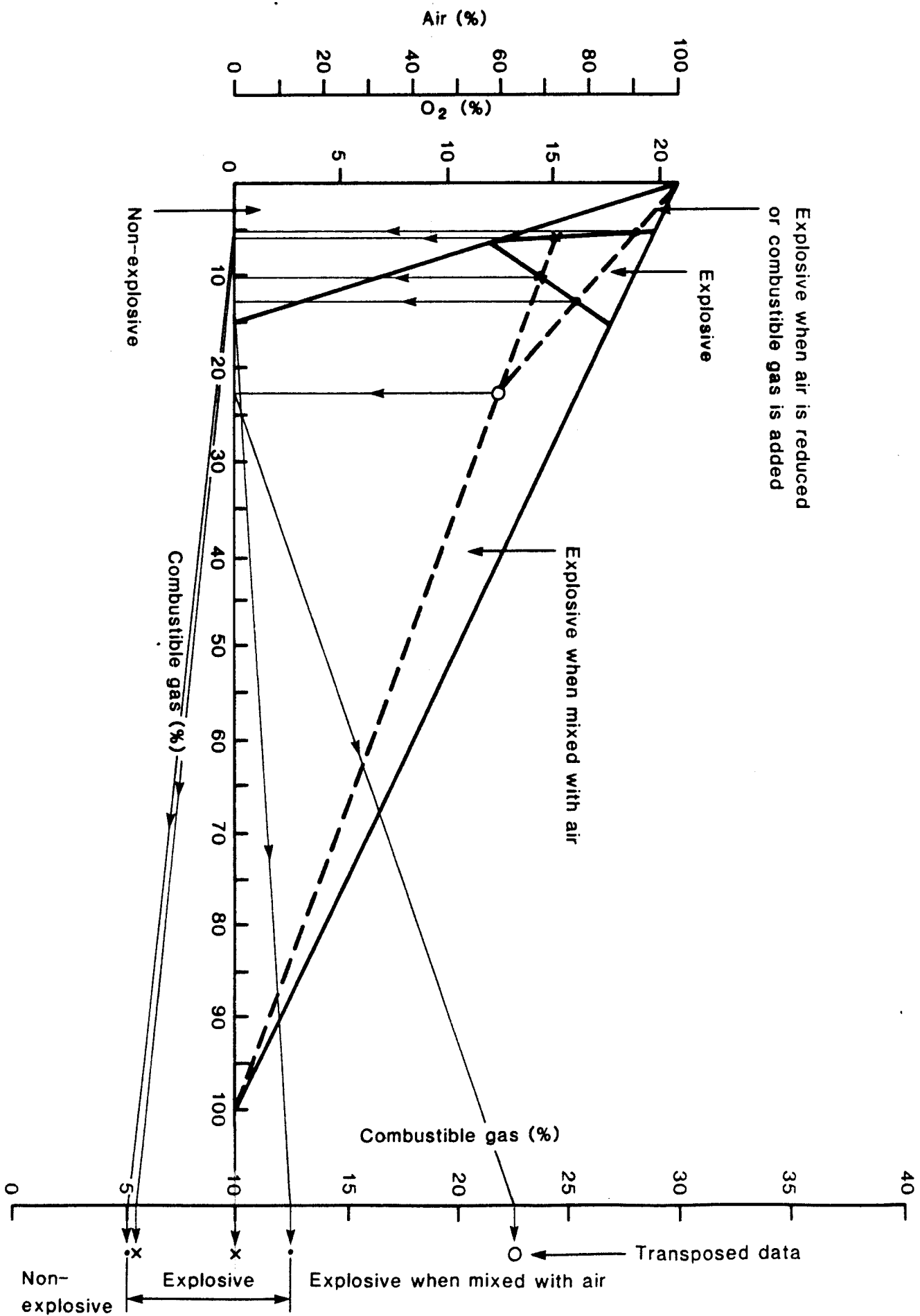


Figure II.6(d) Plotting sample 4 on the Coward Triangle

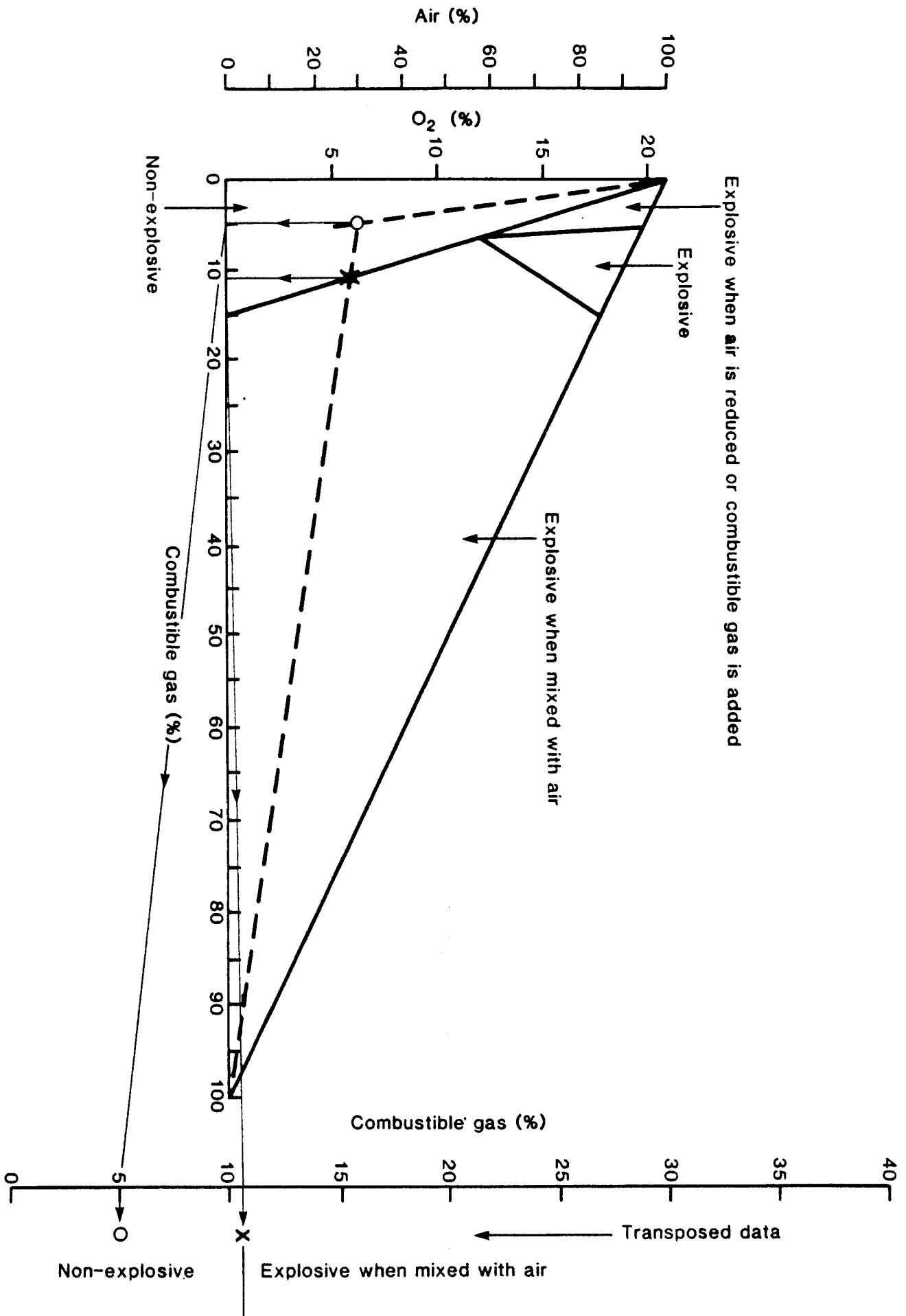


Figure II.6(e) Plotting sample 5 on the Coward Triangle

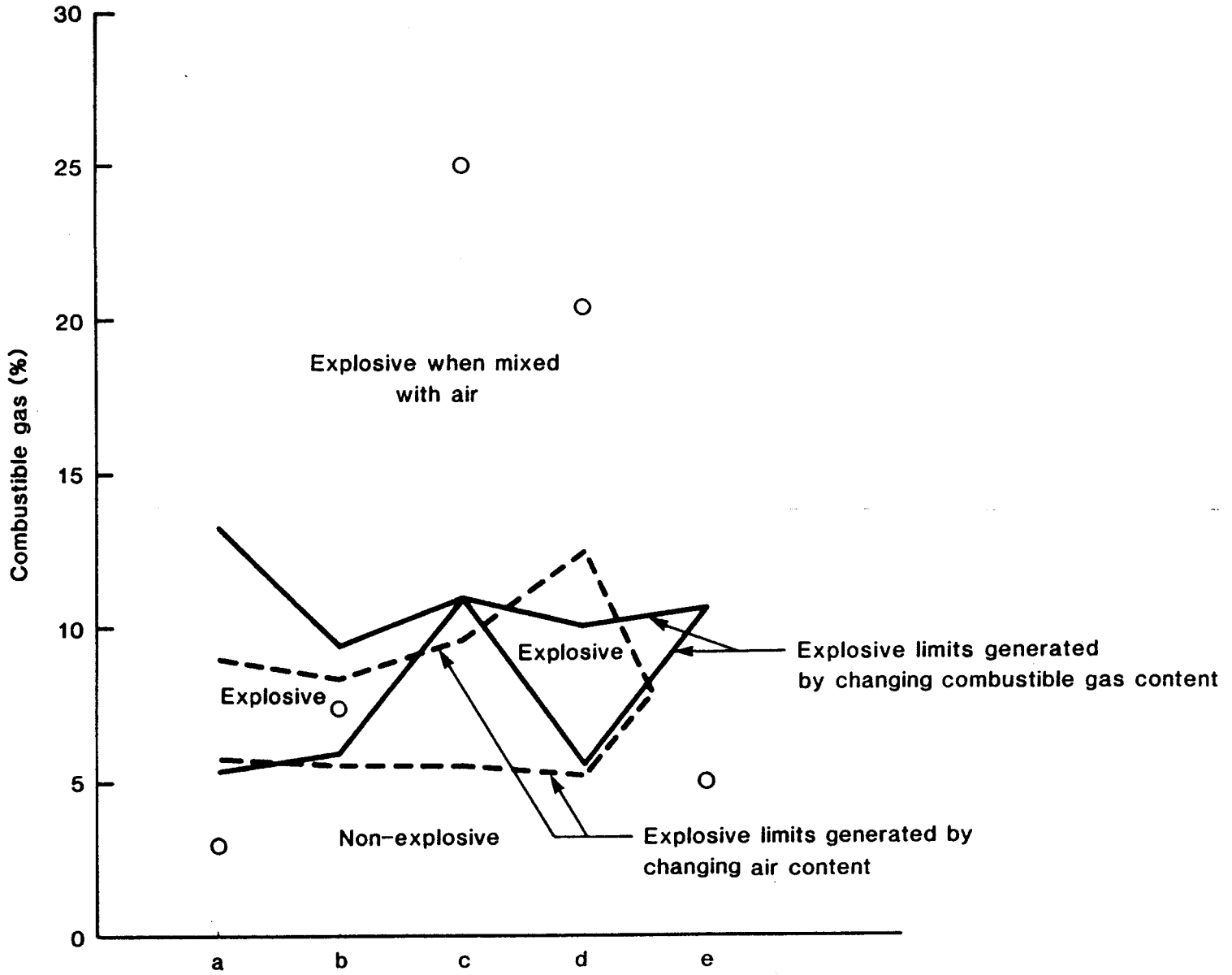


Figure II.6(f) Displaying the explosibility trend of samples 1-5 related to time

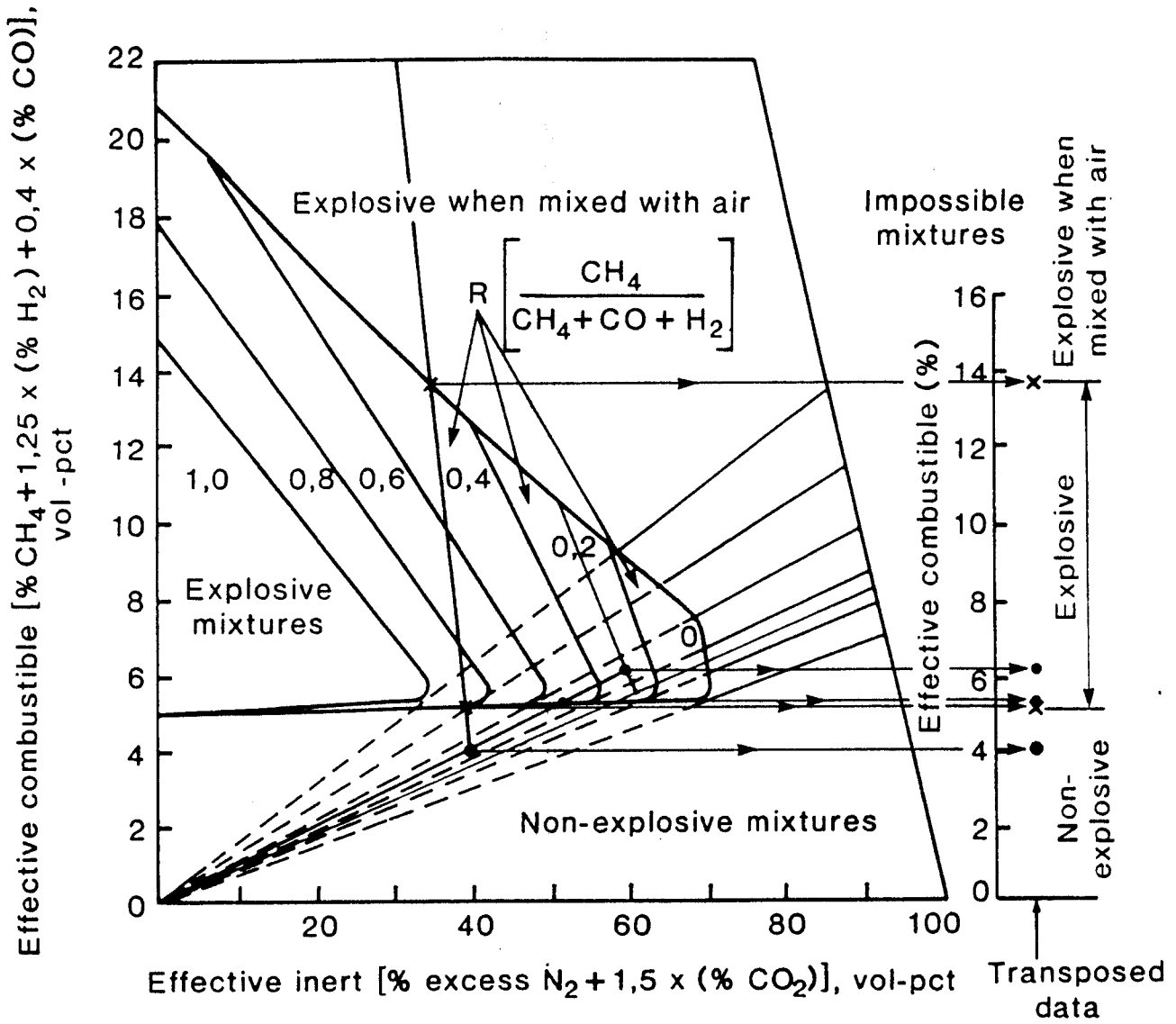


Figure II.7(a) Plotting sample 1 on the USBM Diagram

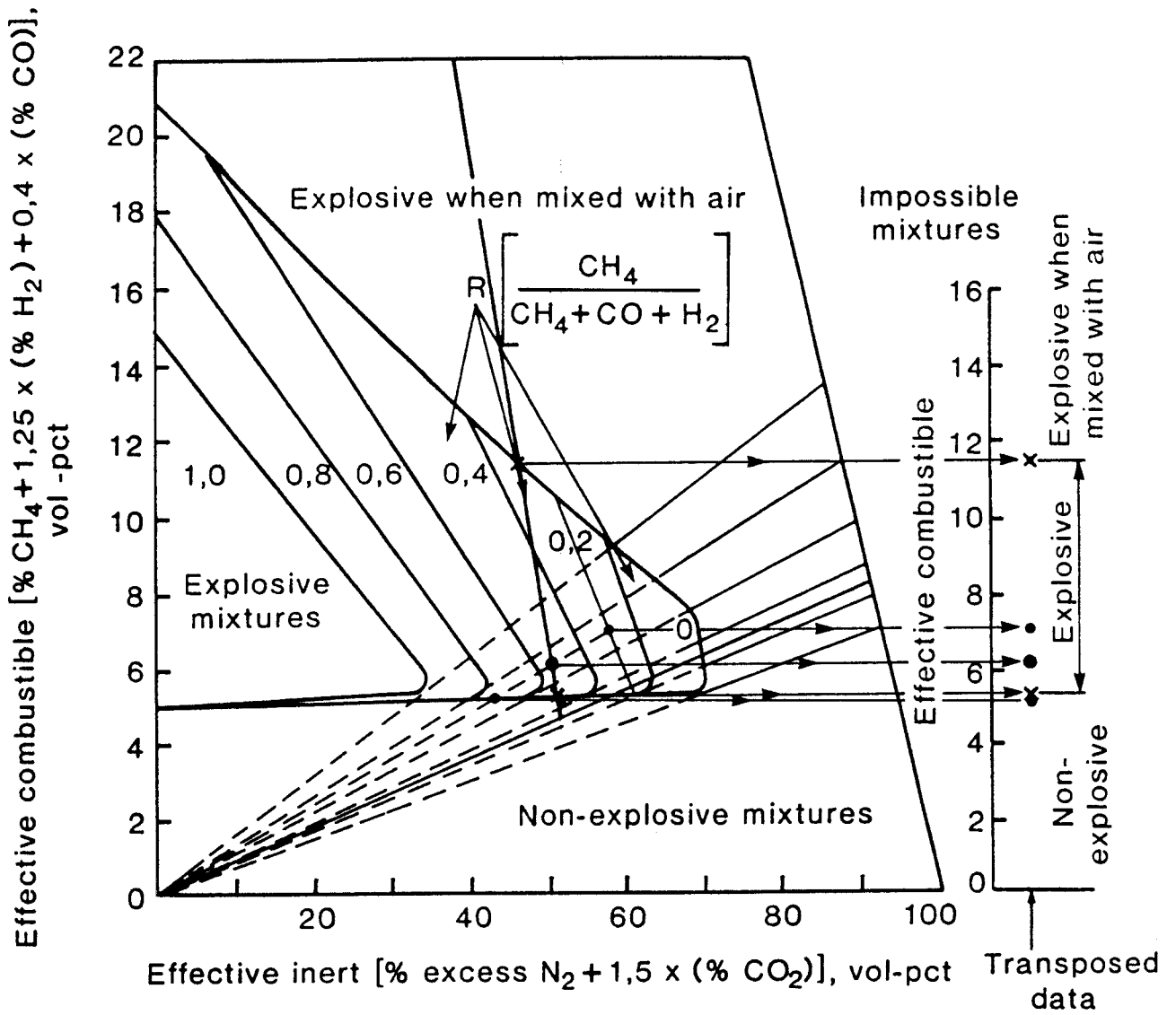


Figure II.7(b) Plotting sample 2 on the USBM Diagram

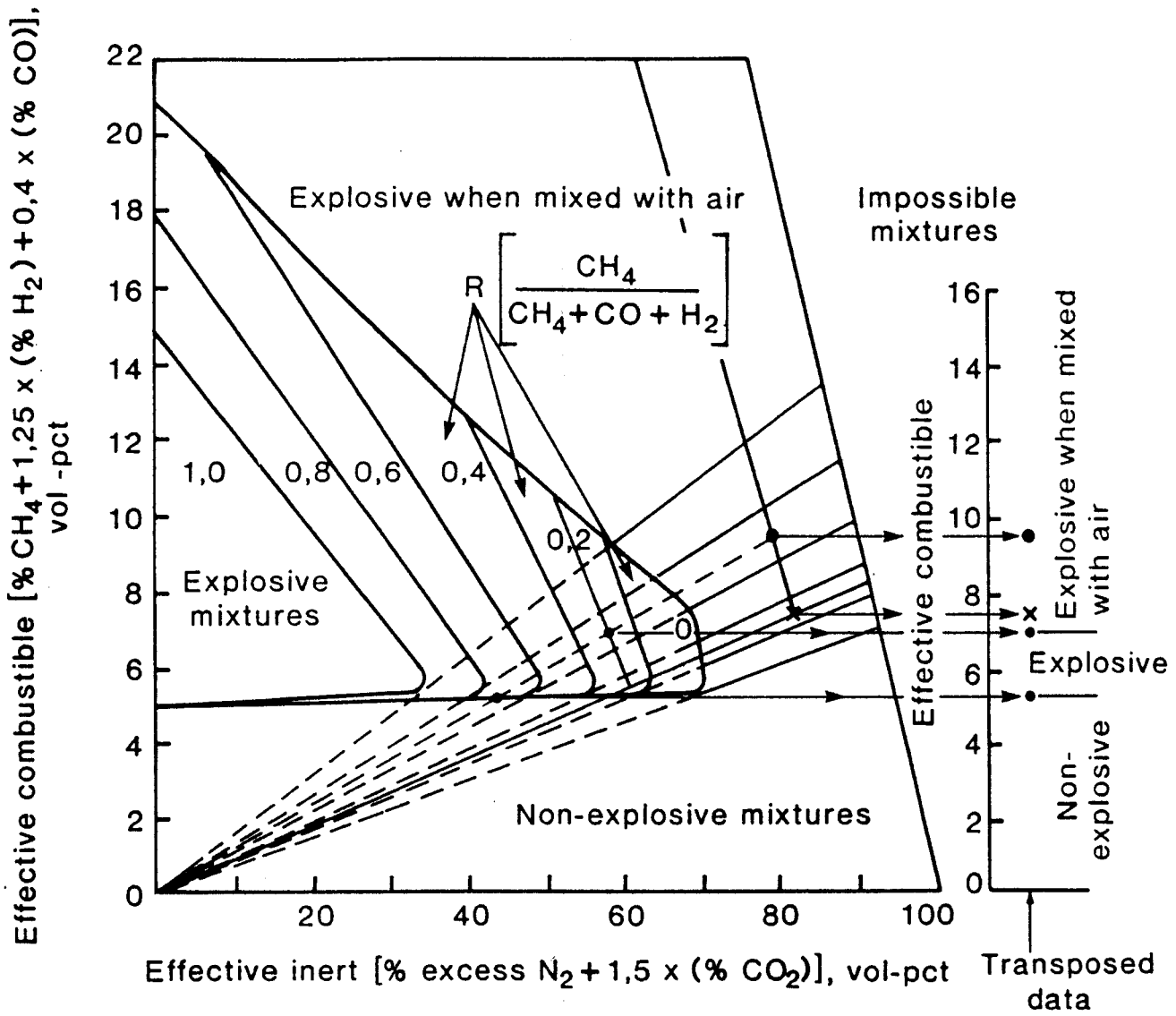


Figure II.7(c) Plotting sample 3 on the USBM Diagram

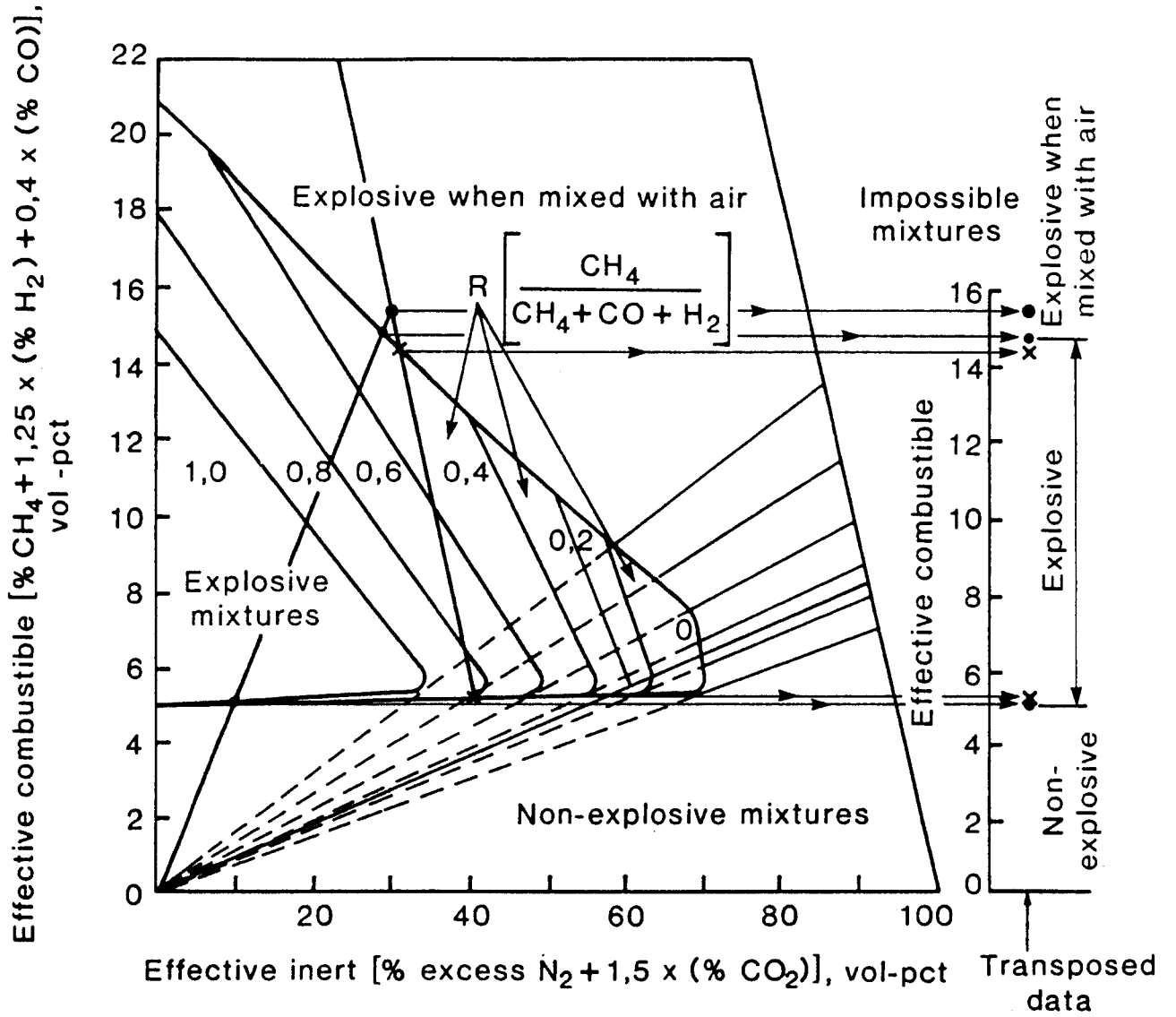


Figure II.7(d) Plotting sample 4 on the USBM Diagram

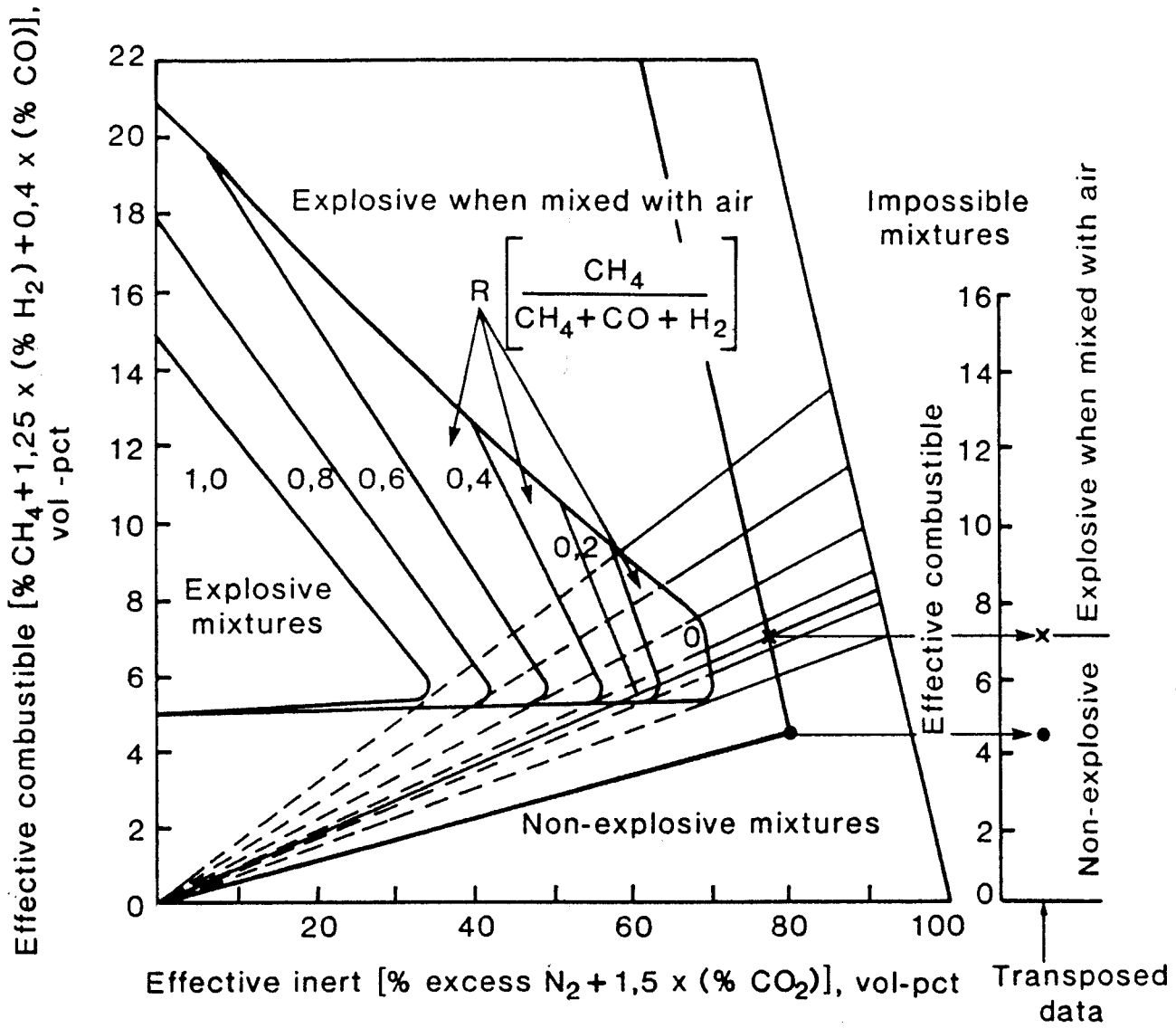


Figure II.7(e) Plotting sample 5 on the USBM Diagram

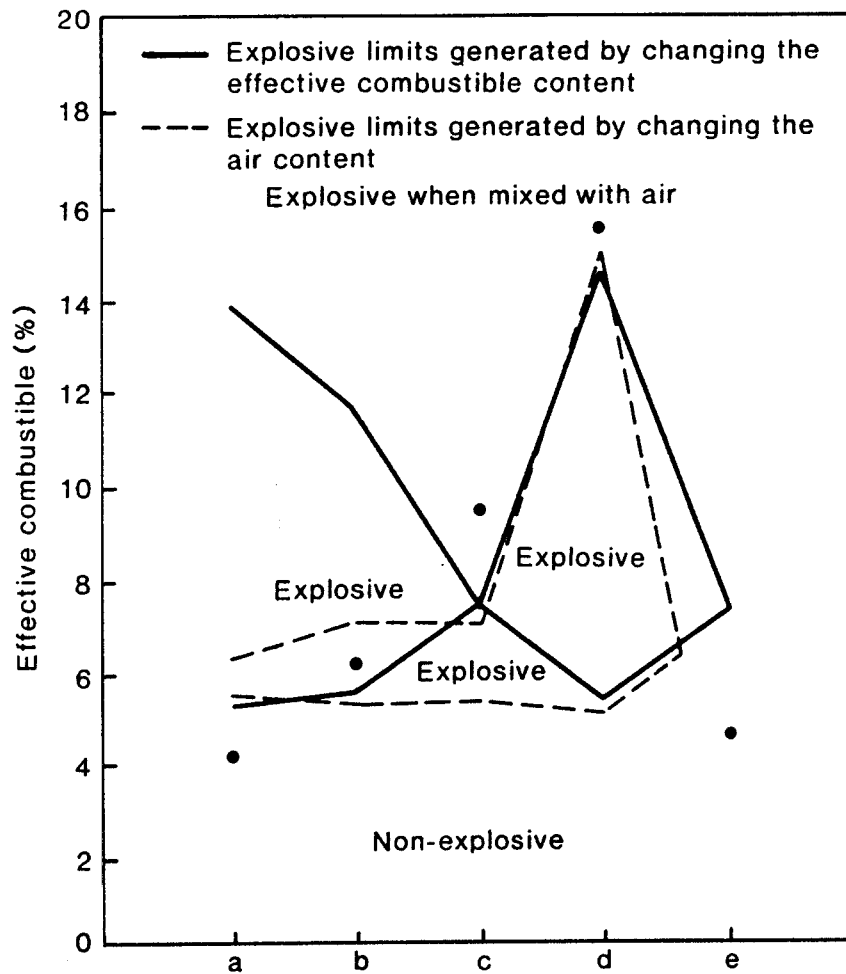


Figure II.7(f) Displaying the explosibility trends of samples 1-5 in relation to the sample times

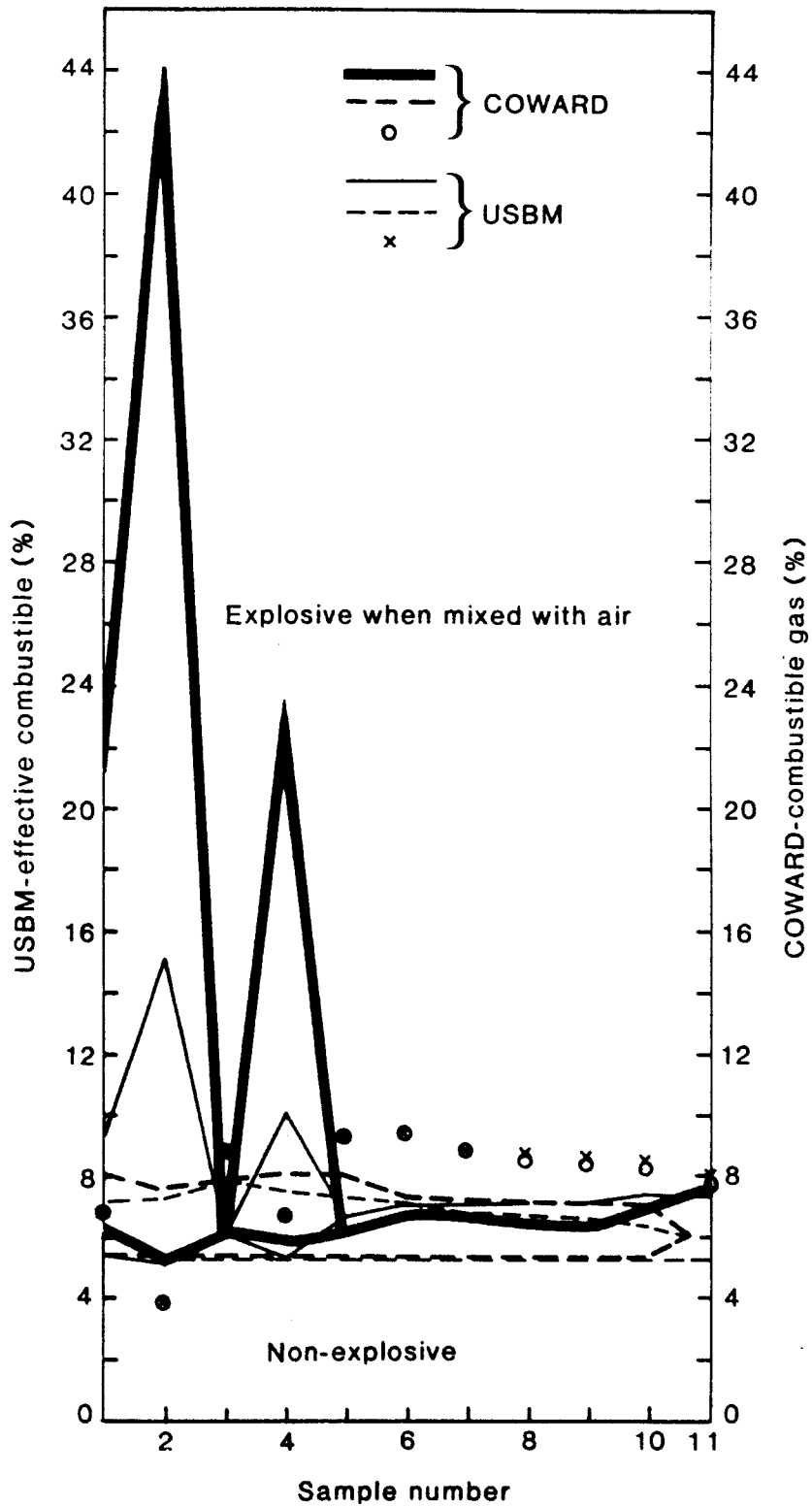


Figure II.8 Comparing the USBM diagram and Coward triangle using samples from an actual fire

APPENDIX III

PRESSURE BALANCING

The pressure difference required across whole mines and sections of mines in order to create ventilation flow can also be instrumental in creating leakage currents which can maintain a fire even after it has been sealed off. This is particularly true in the case of spontaneous combustion, in which smouldering can be maintained by very low concentrations of oxygen.

Pressure balancing is a means of minimising or even nullifying these unwanted pressure differences while maintaining normal ventilation flow in the rest of the mine.

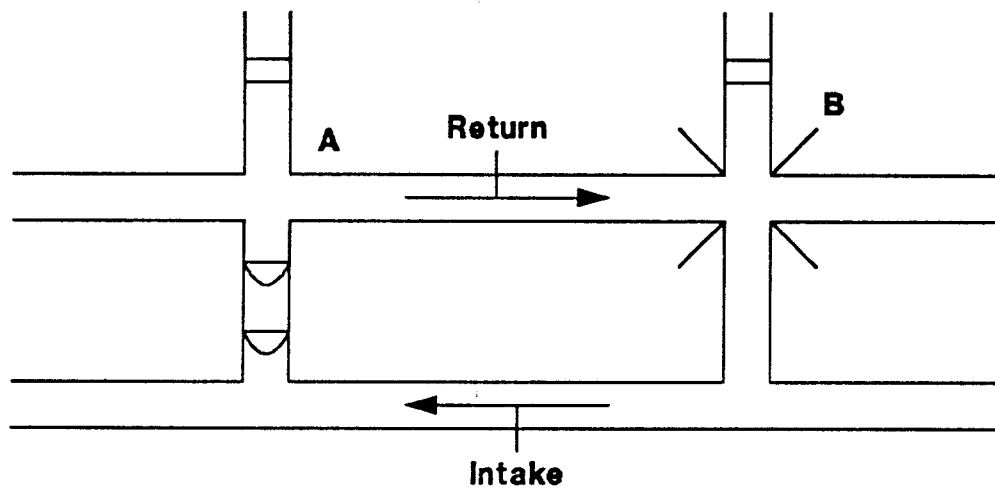


Figure III. 1 Simple pressure balancing using normal working pressure difference

Normal working pressure difference across the sealed section. The face of the stopping at B is at intake pressure while the face of the stopping at A is at return pressure, so there is a large pressure difference across the sealed area.

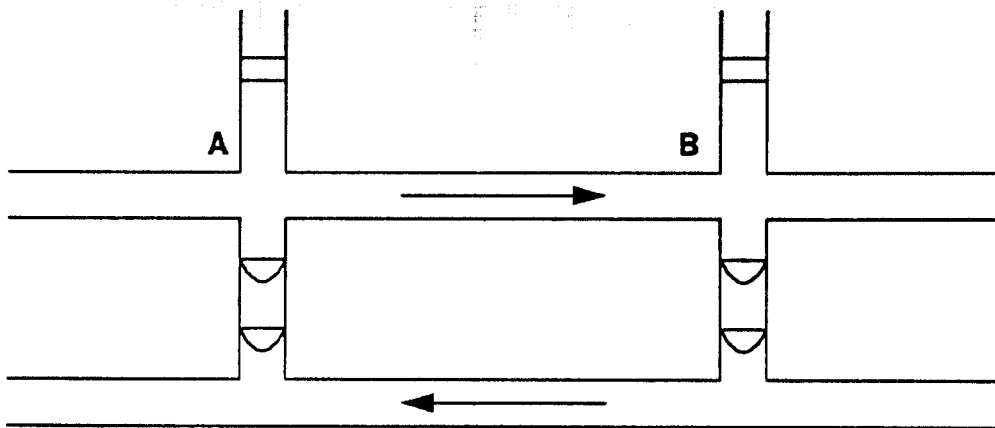


Figure III. 2

Pressure difference across the sealed section now equals the pressure loss from A to B and is less than it was originally.

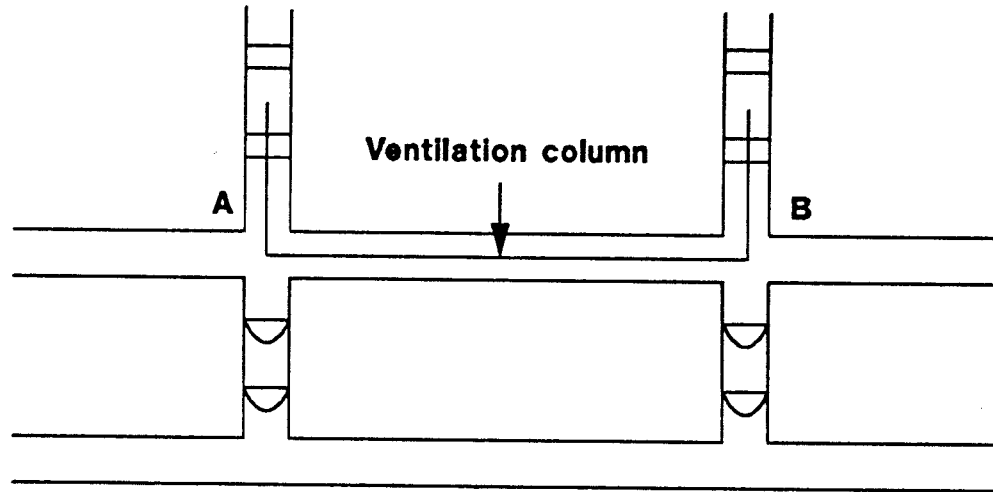


Figure III. 3

There is little or no flow along the ventilation column, therefore there is little or no pressure difference between the faces of the inner seals at A and B.

Building a second seal in front of the original one, as shown in Figure III. 3, creates a pressure chamber between the two. It is sometimes necessary to raise the pressure in this pressure chamber to oppose a tendency for the original seal to 'breath out', or to lower the pressure in the pressure chamber to oppose a tendency for the original seal to 'breath in'. Figure III. 4 shows how this can sometimes be done using normal mine pressure difference.

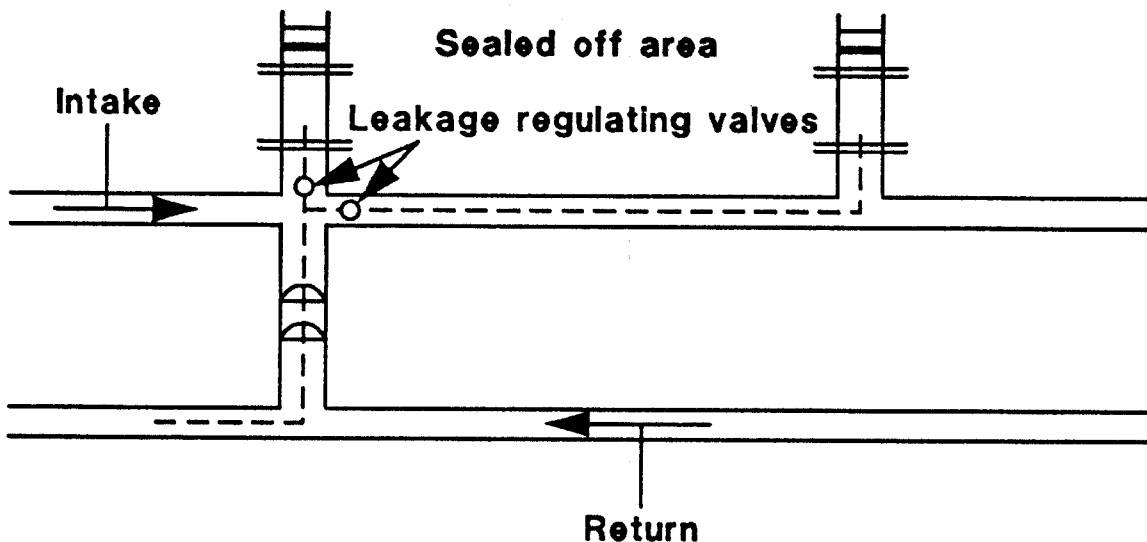


Figure III. 4

If it is not possible to achieve an adequate pressure balance using the normal mine pressure differences, it is then necessary to resort to local devices, usually fans and regulators. It

must be borne in mind that when such devices are used to achieve a balance, the failure of any of these devices will upset the balance. To this extent the use of normal mine pressure differences is preferred as the surface fan is less likely to fail than local devices, and even if it does, pressure balances between different parts of the mine will remain largely unchanged, all circuits in the mine being affected more or less equally.

Creating a pressure chamber using compressed air injectors or ejectors

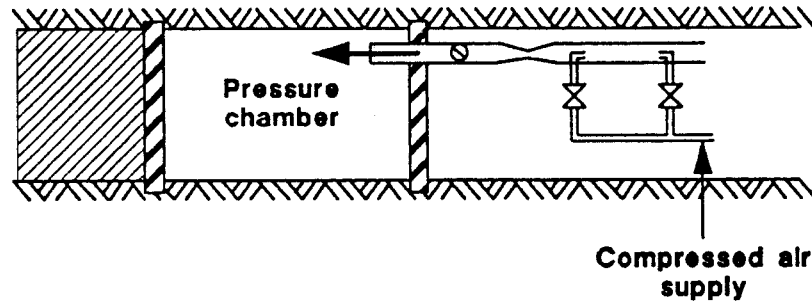


Figure III. 5

The pressure in the pressure chamber can be raised or lowered according to the direction in which the compressed air is injected. It is necessary to monitor the pressure in the chamber as over-pressurising a return side seal or under-pressurising an intake side seal can cause polluted air from inside the sealed area to be expelled into the mine airways. Pressure should be adjusted to create a slight outflow from the return seal. Continuous monitoring of pressure is required as the effects of barometric pressure changes must be counteracted as well as imbalances in mine pressures.

Creating a pressure chamber using auxiliary fans

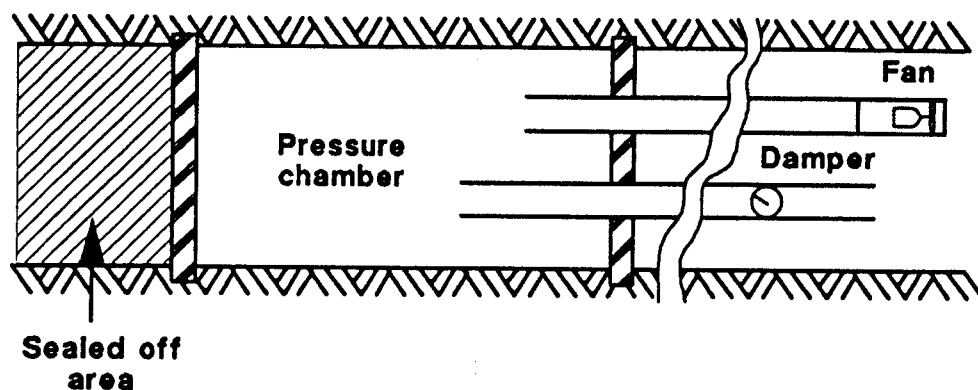


Figure III. 6

Again, a forcing fan can be used to raise the pressure in the pressure chamber, while an exhaust fan can be used to lower the pressure in the pressure chamber. Such an exhaust fan must be bifurcated and must be switched off automatically if the exhaust air contains 1.4% flammable gas or more. The purpose of the damper at the seal is to control the pressure in the pressure chamber, while the damper at the fan is to prevent the fan from going into the stall zone. While the fan must be situated in an adequate through ventilation current, the fan duct must be as short as practicable so as to minimise duct pressure losses, thus maximising the range of pressurisation in the pressure chamber.

Control of pressure on a longwall retreat panel

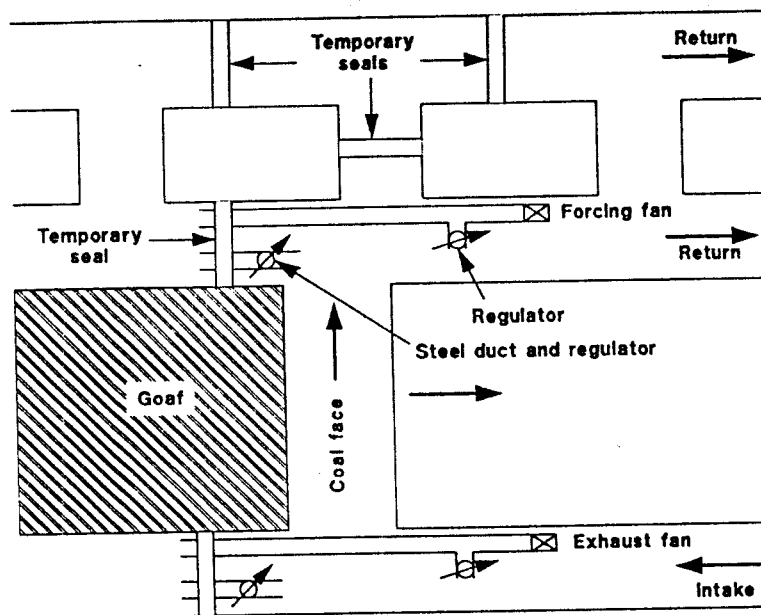


Figure III. 7

The principle is to apply a slight negative pressure to the goaf at the intake end of the face and slight positive pressure at the return end of the face, thus minimising the pressure difference across the goaf.

Continuous monitoring of pressures as well as carbon monoxide and methane concentrations is required. The system would be used where there is a severe liability to spontaneous combustion and moderate methane emission, since the use of any gas bleeder system would have to be suspended while the pressure balancing system was in operation.

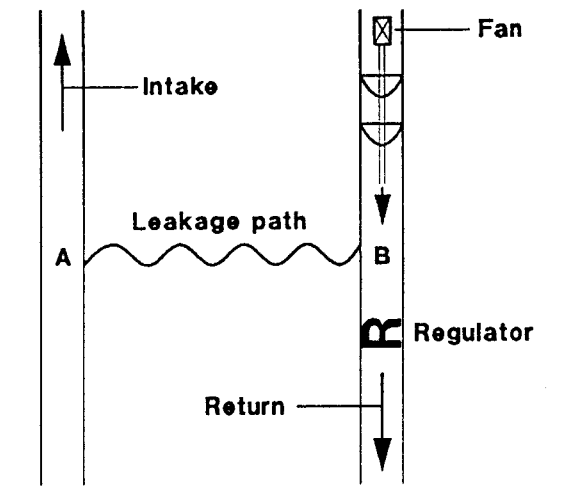
Control of pressure across the leakage path

Figure III.8

The fan must be capable of handling the normal air quantity flowing around the circuit. The regulator and fan combination can be adjusted to raise the pressure at B so as to equalise the pressure difference between A and B. A close watch would have to be kept on the flammable gas content in the return, and under present South African restrictions special permission would be required to operate the fan in the position shown.

The system can be used 'in reverse' order to lower the pressure at A rather than raise it at B.

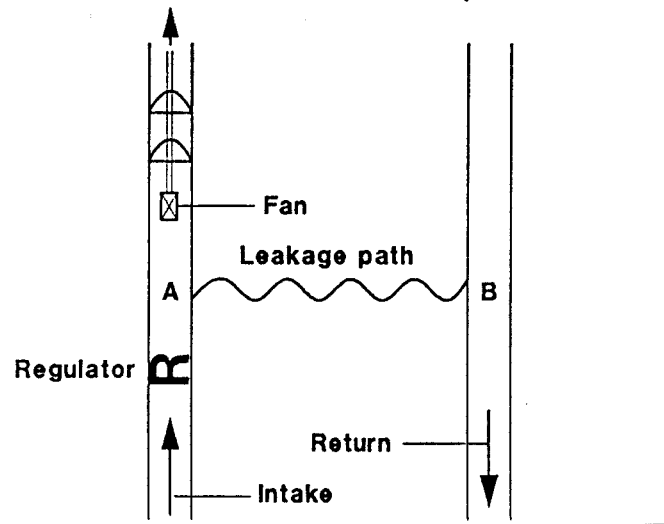


Figure III.9

In this case the fan would be in intake air and therefore in fresh air. However, as the intake airway is likely to contain other equipment such as a conveyor, it may not be convenient to operate the fan and regulator in this position.

In order to design pressure balancing systems of any type, a detailed knowledge of the pressure and flow distribution in all parts of the mine is essential. Systematic measurements

of pressure differences should therefore be made as part of the routine duties of the ventilation department.

Efforts at pressure balancing can be affected by gradients. Where there is a significant gradient there may be a buoyancy effect due to differences in the composition or temperature of the gas. If the buoyancy effect arises from the make of methane rather than hot gases from a fire the effect may not be apparent until some time after the stoppings are complete.

A close watch must be kept at the higher stoppings to ensure that any appearance of noxious or flammable gas is noted. This could be corrected by the drainage of gases or the use of a pressure chamber. If the buoyancy effect is caused by hot gases it will reduce as the gases cool.

APPENDIX IV

THE PROPENSITY OF COAL TO IGNITE SPONTANEOUSLY

Tests of various types designed to predict the propensity of coal or associated strata to ignite spontaneously, using samples of the material under test and laboratory techniques, have been carried out for many years. Some test methods which have been used with some success overseas have not enjoyed the same success when used on South African coals. It was concluded that there were significant differences between the characteristics of South African coals and coals from overseas.

A new index has been devised at the University of the Witwatersrand. This is known as the WITS-EHAC Index, the research having been funded by EHAC (Explosion Hazards Advisory Committee). The test method has been used fairly widely elsewhere and the tests can be conducted in a laboratory. A sample of the test material is placed in a container and subjected to external heating. An inert reference medium is subjected to the same external heating. The temperature of the test material and that of the inert reference medium are measured. The difference in temperature between the two is plotted against the temperature of the inert reference medium to form a diagram known as a differential thermal analysis thermogram. The thermogram shows three distinct stages as illustrated in Figure IV. 1.

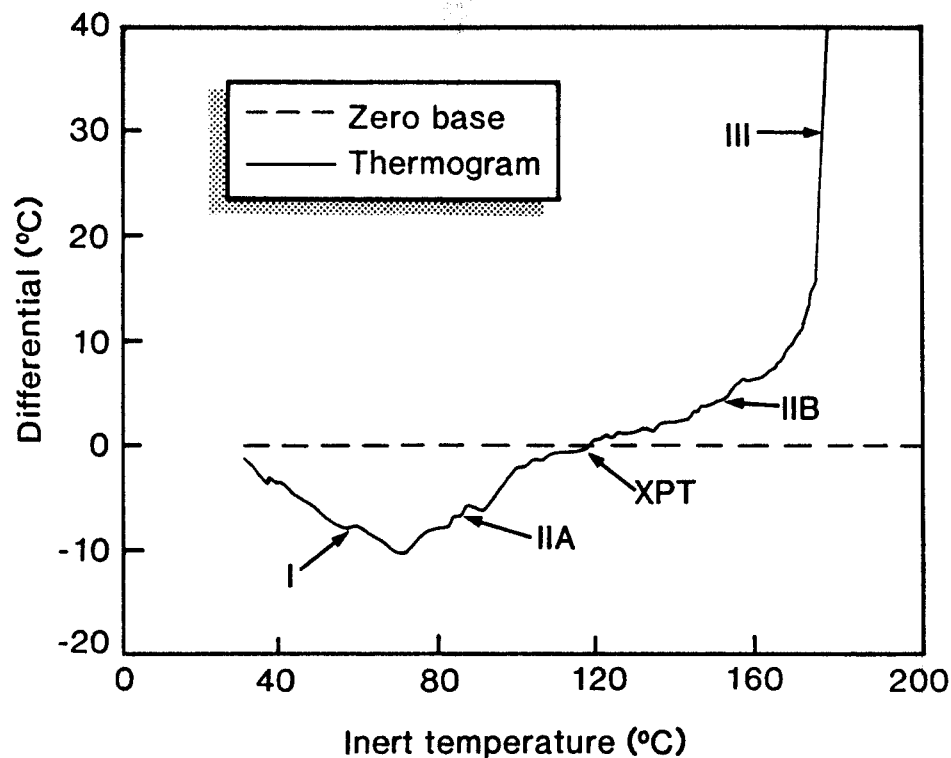


Figure IV. 1 TYPICAL DIFFERENTIAL THERMAL ANALYSIS THERMOGRAM

In Stage I the temperature of the test material falls below that of the inert reference medium because of the latent heat required to evaporate the moisture in the test material. When all the moisture is evaporated Stage II (a) commences and the temperature of the test material starts to gain on that of the inert reference medium. The two temperatures eventually become equal and the temperature at which this occurs is known as the Crossing Point Temperature (XPT). Stage II (b) then begins and as the heating process of the test material becomes exothermic the temperature of the test material rises above that of the inert reference medium. As the process becomes even more exothermic the temperature of the test material 'takes off'. This is Stage III of the thermogram. It should be noted that the Stages on the thermogram are not always as clear cut as those shown in Figure IV. 1.

The basis of the WITS-EHAC Index is that a coal with a propensity for self heating will have a low XPT (or a high value of $\frac{1}{XPT}$) and a high Stage II slope. (If a point on the $\frac{1}{XPT}$ axis is joined with a point on the Stage II slope, as shown in Figure IV. 2, by a straight line, a triangle is formed.) The greater the propensity for self-heating, the greater will be the area of the triangle. The WITS-EHAC Index is simply a measure of the area of the triangle, and is given by:

$$\text{WITS - EHAC Index} = 0,5 \times \text{Stage II slope} \times \frac{1}{XPT} \times 1000$$

Correlation between WITS-EHAC Index values in specific mining areas and known incidence of spontaneous combustion has so far been reasonable.

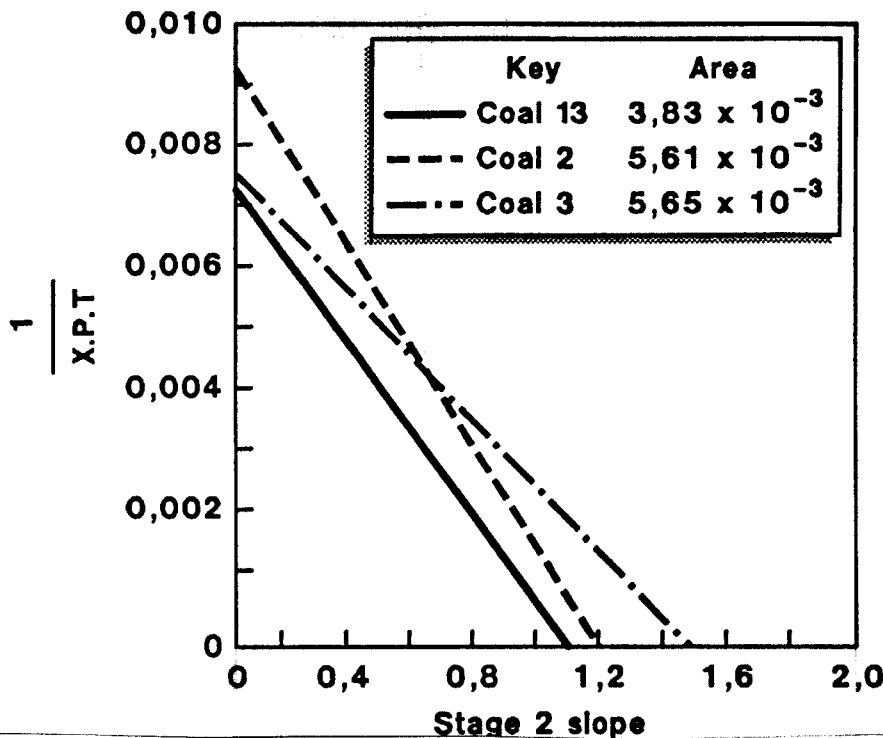


Figure IV. 2

APPENDIX V

THE DEFICIENCY OF THE MOGAL IN RESPECT OF HYDROCARBONS

In 1992 there was an occurrence in which detector tubes were used to test for carbon monoxide where there was a suspected heating. The reading was 3 500 ppm. Samples were taken and analyzed by a Mobile Gas Analysis Laboratory (MOGAL) which indicated a low concentration of carbon monoxide. Hand held electronic instruments confirmed the MOGAL reading. Only after samples were analysed by a gas chromatograph was it revealed that there was a high concentration of ethylene (C₂H₄) present.

Two samples were analysed, with the following results.

	Sample 1	Sample 2
O ₂ %	6,10	5,80
CO%	Trace	Trace
CO ₂ %	2,40	2,50
CH ₄ %	31,10	31,10
C ₂ H ₄ %	2,70	2,70
H ₂ %	0,22	0,23
N ₂ %	57,50	57,70

It is well known that detector tubes designed to detect carbon monoxide are cross-sensitive to ethylene, so the presence of ethylene was the reason for the high carbon monoxide indication. Since carbon monoxide and ethylene are both indicators of a heating (See Appendix I) the practical steps to be taken by management would be the same whether the high carbon monoxide indication is due to carbon monoxide or ethylene. In this case samples were taken and submitted for fuller analysis. The reason the matter was not cleared up at this point is that the MOGAL is not designed to measure the ethylene concentration. The detector tubes 'found' the carbon monoxide or ethylene but could not distinguish between the two, while the MOGAL could not detect the ethylene. The MOGAL appears to have been specified with a view to assessing explosibility by measuring or calculating all the parameters needed to use the USBM Explosibility Diagram, or the Coward Diagram. This is a deficiency which should be remedied as a knowledge of the presence of hydrocarbons is a useful refinement in the detection of spontaneous combustion in its early stages. A gas chromatograph would fill the gap and could be used either in conjunction with an existing MOGAL or in place of one. Most mining countries seem to have adopted the gas chromatograph as standard equipment for gas analysis, and some versions can be used underground.

The significance of ethylene

Ethylene is a product of a heating at a temperature of 150°C - 180°C. It is not a product of a fire because it would be consumed by the fire. However, the atmosphere represented by the samples quoted earlier does not seem to be typical of that produced by a heating. A heating always produces more carbon monoxide than hydrogen and more hydrogen than ethylene or propylene. The Trickett Index for the samples is 0,4, which denotes a low level of activity and would be considered by some authorities as denoting residual gases.

It would seem that there are two possible explanations for the extremely high concentration of ethylene in the sample:

- a) The analysis is incorrect.
- b) The mixture of gases is a residue from a past heating, and the original gases have undergone some sort of change. This change has resulted in a reduction in carbon monoxide and hydrogen, leaving the ethylene intact.

This is obviously a very unusual, if not unique, set of circumstances.