Experience in longwall mining at Coalbrook Collieries

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SYNOPSIS

Coalbrook Collieries has produced approximately 2 million tons of coal by retreat longwalling since April 1976. Strata control, support, stone intrusions, coal conveyance, and the inflow of goaf and water between the face supports have been the major operational problems experienced to date. The progressive percentage availability of the total system (moves between panels excluded) for the first three panels mined was 40 per cent. Satisfactory solutions have been evolved for these problems, and the availability in the present panel (fourth panel) has increased to approximately 60 per cent. It is envisaged that longwalling will become the most economical method of ensuring optimum heat values, greater recovery, and reduced geographic expansion, as well as safer roof conditions, at Coalbrook.

SAMEVATTING

Coalbrook Collieries het sedert April 1976 ongeveer 2 miljoen ton steenkool met truwaartse strookafbou gemyn. Stratabeheer, bestutting, klipindringings, vervoer van steenkool en die invloei van puin en water tussen die frontstutte was die grootste bedryfsprobleme tot dusver ondervind. Die progressiewe beskikbare produksietyd van die totale stelsel vir die eerste drie panele (verskuiwings tussen panele uitgesluit) was 40 persent. Bevredigende oplossings vir die probleme is gevind en die beskikbaarheid in die huidige paneel (vierde paneel) het tot ongeveer 60 persent gestyg. Die verwagting is dat strookafbou die mees ekonomies mynboumetode sal word om optimum hittewaardes, groter herwinning en beperkte geografiese uitbreiding asook veiliger daktoestande op Coalbrook te verseker.

Introduction

Coalbrook Collieries, which is situated in the Vaal Basin coalfield approximately 20 km south-east of Sasolburg, supplies approximately 3,5 million tons of low-grade bituminous coal per annum to Escom's Taaibos and Highveld power stations. Some 30 per cent of the output is mined by retreat longwalling, while the balance is produced by continuous miner and conventional mechanized bord-and-pillar sections.

The longwall installation has been in operation since April 1976 and has mined approximately 2,0 million tons to date. Three panels have been completed, and production started in the fourth panel during August 1979. On average, the longwall face produced approximately 665 t per shift in the first three panels, with an average availability of 40 per cent excluding longwall moves. Mining difficulties associated with adverse geological conditions accounted for some 21 per cent of these delays, while engineering problems on the shearer, armoured-face conveyor, and outbye conveyor system accounted for approximately 9 per cent, 10 per cent, and 15 per cent respectively. The production from the present panel has improved to 1680 t per shift, with an average availability of approximately 60 per cent, as shown in Table I.

The best weekly output achieved is 34 880 t, while the best daily output over three shifts is 8000 t. Cost comparisons with the other mining methods applied at Coalbrook are being reviewed in the light of improved conditions, and modification and experience of the equipment. However, there are indications that the operational costs of longwalling compare favourably. Although the longwalling method results in reduced costs of geographic expansion, the relatively high capital expenditure involved in a longwall installation requires the consistent attainment of high rates of production. The recent performance of the installation gives rise to optimism that these rates can be achieved.

Reasons for Introduction of Longwalling

At Coalbrook Collieries, three coal seams (no. 1, no. 2, and no. 3 seams) are available to meet the quality specifications in terms of calorific value and content of volatile matter demanded by Taaibos and Highveld power stations.

Bord-and-pillar mining by means of hand-got, conventional mechanized, and continuous miner methods has been extensively applied. Owing to the poor mining conditions and the depth of the workings, the percentage extraction of the reserves is extremely low, while the mining costs are among the highest in the Transvaal and Orange Free State coalfields. Bord-and-pillar mining at 200 m with a safety factor of 1,6 and geological losses of approximately 10 per cent results in an effective extraction percentage in no. 1 or no. 2 seam of 35 per cent. Where these seams are separated by less than 3 m of parting, only one seam can be extracted by bord-and-pillar methods because the parting is laminated shale and sandstone of inferior nature. The effective extraction of the reserves in no. 1 and no 2 seam is therefore approximately 18 per cent. It is therefore possible by longwall mining to increase the percentage recovery of coal from something less than 30 per cent to approximately 70 per cent.

The roof conditions in no. 2 seam are extremely poor. There is a regional line of weakness that runs roughly in the same direction as the main east–west development of the mine. This weakness, together with fracturing due to differential compaction, breaks the roof up badly,

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TABLE I

LONGWALL REPORT, COALBROOK COLLIERIES

	Panel 4											
	Week ending											
	79-09-29		79-10-06		79-10-13		79-10-20		79-10-27		79-11-03	
1.0 *Production*												
1.1 Weekly budget	23 000		25 000		25 000		25 000		25 000		28 000	
1.2 Weekly actual	23 048		27 217		27 566		23 106		22 756		32 292	
1.3 Progressive weekly average	18 519		19 605		20 490		20 751		20 917		21 865	
1.4 Total progressive for panel	130 436		157 653		185 219		208 325		230 901		263 193	
2.0 *Delays*	**F*	**P*	*F*	*P*	*F*	*P*	*F*	*P*	*F*	*P*	*F*	*P*
2.1 Maintenance	12	9,1	11	6,8	10	7,8	10	7,1	9	5,8	12	7,7
2.2 Shearer	3	1,5	8	3,2	3	0,9	2	0,4	0	0,0	3	1,6
2.3 Face conveyor	6	3,9	3	1,9	4	2,9	1	8,4	2	4,0	4	1,3
2.4 Stage loader	2	2,8	0	0,0	0	0,0	2	1,1	0	0,0	1	0,2
2.5 Face supports	0	0,0	0	0,0	0	0,0	0	0,0	0	0,0	0	0,0
2.6 Section cables	0	0,0	3	1,3	1	0,4	1	0,3	0	0,0	0	0,0
2.7 Gate conveyor	1	0,4	3	1,2	1	1,6	5	2,6	2	1,6	3	1,3
2.8 Energy train	0	0,0	0	0,0	1	0,4	1	0,6	0	0,0	3	1,1
2.9 Pick change	0	0,0	0	0,0	0	0,0	0	0,0	0	0,0	0	0,0
2.10 Move stage loader	0	0,0	6	6,3	1	0,4	2	0,6	0	0,0	2	1,4
2.11 Face falls of roof	0	0,0	0	0,0	0	0,0	0	0,0	0	0,0	0	0,0
2.12 Face slabbing	0	0,0	3	2,0	3	4,1	6	2,6	4	3,4	3	0,6
2.13 Gate falls of roof	0	0,0	0	0,0	0	0,0	0	0,0	0	0,0	0	0,0
2.14 Inflow of groundwater	0	0,0	0	0,0	0	0,0	0	0,0	0	0,0	1	0,2
2.15 Gas	0	0,0	0	0,0	0	0,0	0	0,0	0	0,0	0	0,0
2.16 Water supply	3	2,1	2	0,4	3	0,6	0	0,0	0	0,0	2	0,5
2.17 Power supply	0	0,0	1	0,4	0	0,0	3	1,6	1	0,2	1	0,4
2.18 Main conveyors	15	10,0	1	2,3	21	8,6	19	8,5	35	25,5	39	12,6
2.19 Shaft conveyor system	3	3,4	23	8,4	9	3,1	26	8,0	10	2,6	5	1,3
2.20 Others	0	0,0	1	0,2	4	1,0	2	0,5	5	1,0	2	0,6
2.21 Total downtime		33,2		34,6		31,7		42,3		43,9		30,7
3.0 *Performance*												
3.1 Percentage availability	66,8		65,4		68,3		57,7		56,1		69,3	
3.2 Weekly available production time (hours)	88,3		86,3		90,2		75,8		74,0		91,4	
3.3 Progressive available production time (hours)	517,5		603,8		694,0		769,8		843,8		935,2	
3.4 Weekly tons per available production hour	261,0		315,4		305,6		304,8		305,1		353,3	
3.5 Progressive tons per available hour for panel	252,0		261,1		266,8		270,6		273,6		281,4	

**F* Frequency
**P* Percentage

especially on the northern side of the east–west roadways. In addition, the shales immediately above the no. 2 seam deteriorate very rapidly when exposed. At least 1 m of coal is therefore left in the roof of bord-and-pillar workings in the no. 2 seam, which obviously further decreases the percentage recovery, and coal with better heat values is often found in this roof coal in the no. 2 seam. Excessive pillar scaling often results in unsafe mining conditions. Although systematic roof-bolting is applied in the coal pillars, weathering of the coal and the midseam shale band (in the no. 2 seam) results in the scaling of large slabs. The longwall method of mining affords better protection against these sources of danger.

Although conventional mechanized and continuous mining methods are employed at Coalbrook, the operation has become rather labour-intensive. The rapid expansion of the mine as a result of the low percentage extraction and the adverse roof and pillar scaling conditions, necessitates the employment of large numbers of workmen in the backbye areas. It is estimated that the mine's total labour force could be reduced from the present complement of 2300 to 1800 if longwall mining were applied more extensively. Cost estimates and experience thus far indicate that a total longwall operation at Coalbrook will be the most economical mining method if the longwall units can produce at least 90 000 t/m consistently.

Another prime reason for the introduction of longwalling at Coalbrook is that it will provide firsthand production and cost information for the possible introduction of longwalling on a much larger scale at Matla Coal. The contract awarded by Escom to Trans-Natal Coal Corporation and Clydesdale Collieries for the supply of coal to the new 3600 MW Matla power station recognized the need for the investigation of high-extraction methods of mining. The Coalbrook installation also provides an excellent 'in-company' training ground for longwall mining personnel.

An added advantage of longwalling at Coalbrook is the possibility of improving the heat value of the coal produced. The 1 to 1,5 m of coal left with bord-and-pillar mining in the roof of the no. 2 seam can be extracted within the longwall panel. It is expected that the immediate forward support facility on the face supports will render adequate support if the shales above the no. 2 seam are exposed. This method is being tried at present with promising results.

Also, longwalling is probably the only feasible method of extracting the higher-quality no. 1 seam coal in a double-seam extraction operation where the parting is less than 3m between the no. 2 seam and the no. 1 seam.

Method of Longwalling

The longwall operates on the retreat system with a face length of 215 m and a shearing height of between 2,2 and 2,8 m, as shown in Fig. 1.

The faces lie in an east–west direction, this being the axis along which an apparent line of weakness exists in the roof strata throughout the coalfield. Panel lengths could vary between 800 and 4000 m, depending on layout, and geological and quality considerations. Three

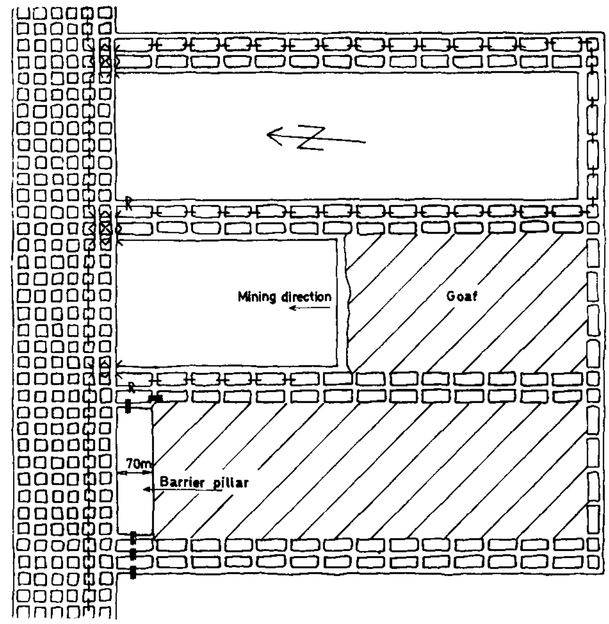

Fig. 1—Layout of the longwalling

roads are driven on either side of the face line by continuous miners. The tailgate road is 4 m wide and accommodates the energy train, which is railmounted. The maingate road is 5 m wide and carries the stage loader, crusher, and 1200 mm panel conveyor. At this stage, the maingate road and companion, as well as the tailgate road, serve as intake airways. The return air returns through the tailgate companions.

The installation consists of 135 Dowty 450 t four-leg chock shields, 11 Dowty 450 t four-leg shields, an Eickhoff E.D.W. 600L shearer, and a Halbach and Braun EKF4 armoured-face conveyor and stage loader with Auldsman & Beckshulte crusher. The energy train is made up of a mobile compressor and store, cable cart, hydraulic power pack, three transformers, and switchgear as illustrated in Fig. 2.

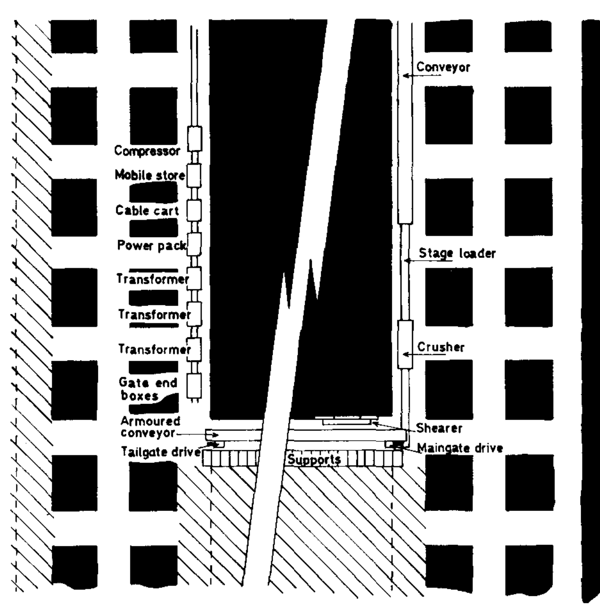

Fig. 2—Layout of the installation

A 34 mm single-strand flight chain powered by two 250 kW motors is in use. The coal is end-discharged onto the stage loader, which feeds it through a crusher set at 20 cm. The automatic tensioning device for the panel conveyor is situated at the drive end of the panel conveyor.

Conventional bi-directional shearing is applied at present, while the half-face method was used on the second panel, which had a face length of 120 m.

The longwall operates from Monday 6 a.m. to Saturday 6 p.m. on the basis of 3 shifts per day. Routine inspection and lubrication services are done on each shift. At this stage, the sub-units of the system (i.e., shearer, armoured-face conveyor, stage loader, etc.) receive a weekly routine maintenance on different days of the week, while major overhauls are done when the equipment is moved from one panel to the next. Information on the performance of the sub-units is being processed in order to relate maintenance to tonnage produced.

Operational Problems

Horizon Control

The distribution of average calorific values of the no. 1 and no. 2 seams is shown in Fig. 3.

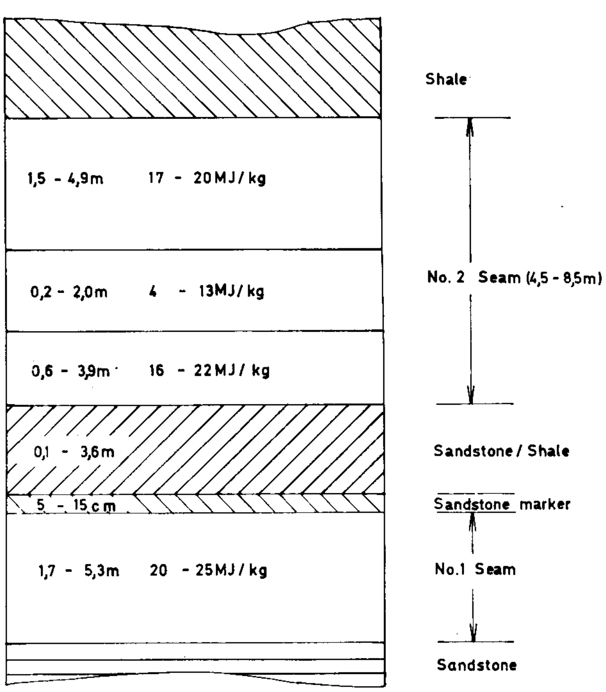

Fig. 3—Section through no. 1 and no. 2 seams

The no. 1 seam, which was longwalled in the previous panel (no. 3 panel), contained occasional sandstone intrusions. The stone caused excessive wear and damage to the armoured-face conveyor, cutting picks and surface mills. Horizon control in the no. 1 seam was achieved by staying clear of the sandstone band in the roof and by limiting the cutting width in order to minimize cutting of stone rolls on the floor. A limited application of horizontal drilling and surveying techniques to determine the presence of stone intrusions was fairly successful.

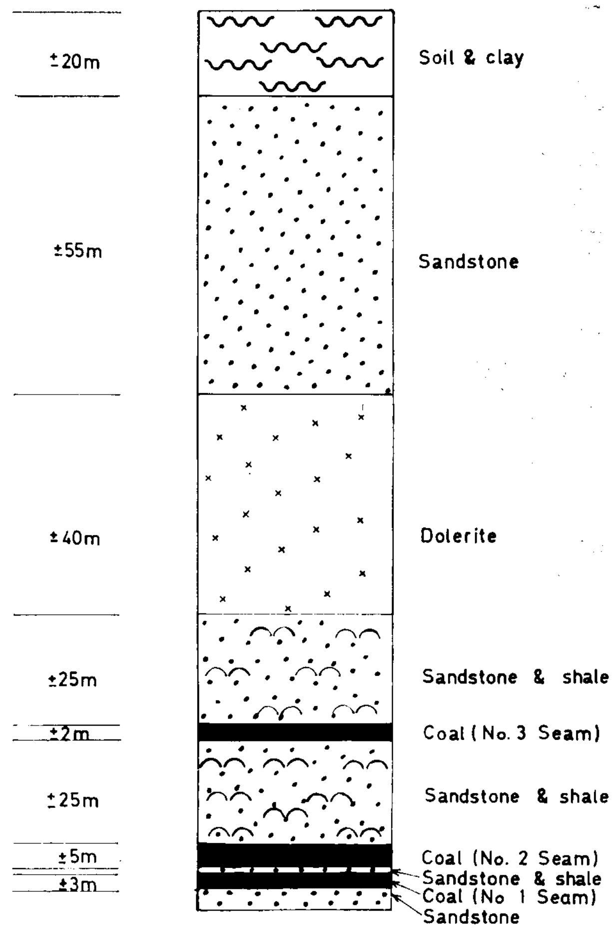

Fig. 4—A typical borehole section

Difficulties were experienced in controlling the cutting horizon in the no. 2 seam on the first two panels. Owing to the weak shale roof immediately above the coal in the no. 2 seam, the primary development is done leaving at least 1m of coal below the shale. This is achieved by relating the mining horizon to three bright coal bands or the shale band, which is generally present in the no. 2 seam. The same horizon is followed in the longwall face using the same indicators. Since some of the higher heat values appear to be in the coal that is being left against the shale roof, an attempt is being made in the present panel to cut as close as possible to the shale roof. Both ends of the face are slightly curved to align the roof horizon of the face with the roof in the gate roads, which, as explained, is at least 1 m below the shale roof. This method appears to be working well, and may be the only way to improve the calorific value of the coal produced from the no. 2 seam since bord-and-pillar mining cannot be applied within 1 to 1,5 m of the shale roof.

Face Alignment

Face alignment is controlled by observing the position of cyalume fluorescent tubes suspended from the supports, and through regular face surveys. The initial problems experienced in keeping the faces straight were overcome mainly by correcting the face line on day shift only when more senior supervision was available.

Face Support

Major roof control problems were experienced in the first panel. The roof supports were initially designed for immediate forward support in a 'one-web-back' method of working. The load that the cantilevers could support (i.e. 10 t) proved to be insufficient to carry the immediate roof in the face. Several falls of roof between the face and supports were experienced.

The borehole section displayed in Fig. 4 shows a 40 m thick competent dolerite sill that is present over most of the area to be longwalled.

On the first face, which was 200 m long, this dolerite sill collapsed only when the face had retreated approximately 250 m. After that, the surface elevations taken indicated that major breaks occurred fairly consistently at 50 to 70 m retreat intervals. The weight of the dolerite in cantilever caused severe pre-formed breaks in the roof and coal seam ahead of the retreating face. The pressure exerted by the dolerite in cantilever together with these pre-face breaks caused the falls of roof as illustrated in Figs. 5 and 6. The situation was aggravated when the heavy weight conditions coincided with geological disturbances or slow face advance due to technical problems.

The immediate problem after such a fall was to remove the rock and support the void created, as shown in Fig. 7. The cavity formed in the roof is basically bridged with steel girders resin-bolted to the face and supported on the main canopies. Heavy face dowelling is required to control spalling, and it is advisable to install resin roofbolts at an angle in the roof contact. Utmost care must be taken during the first webs after the fall to control the roof horizon and to bring the supports in behind the shearer as soon as possible.

It is often necessary to break up large lumps on the armoured face conveyor by means of blasting. During comparatively large falls when the armoured face conveyor was stuck, it became necessary to install a temporary conveyor on the armoured face conveyor and even over the shearer, as illustrated in Fig. 8.

Since the cantilever extensions on the supports were forced down under these heavy weight conditions, it

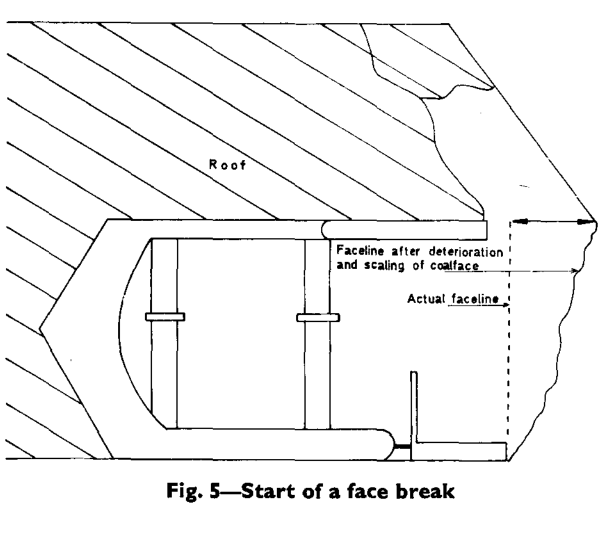

Fig. 5—Start of a face break

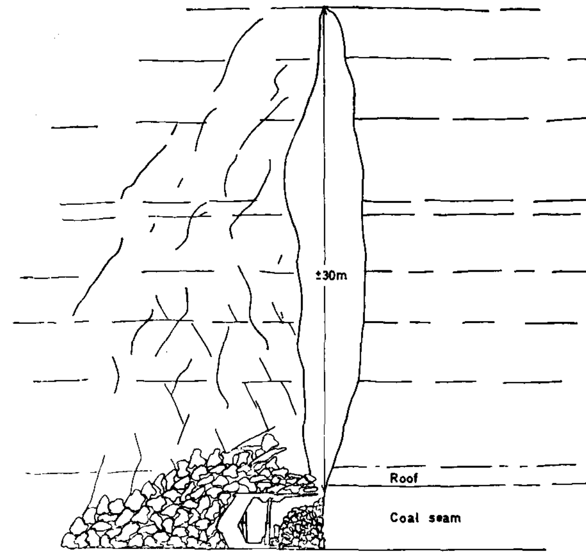

Fig. 6—A face break

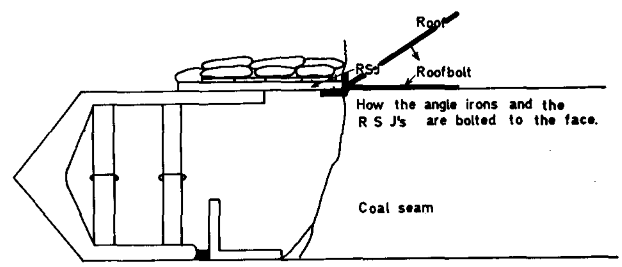

Fig. 7—Support of a face break

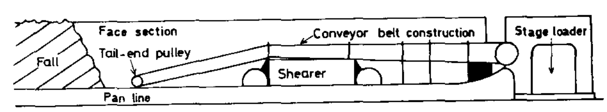

Fig. 8—Section showing the clearing of face break debris

was decided to remove the extensions and advance the supports and armoured face conveyor in the conventional way. Although this resulted in a higher tip load nearer the face, it was not possible to support the newly exposed roof over a distance of some 30 m behind the shearer until the armoured face conveyor had been fully snaked over. Further falls occurred in this snake area.

As a short-term solution, the face length of the second panel was shortened to 120 m where the span of dolerite was unlikely to fail during the panel life. This arrangement enabled the mine to continue longwalling while a start could be made on increasing the load characteristics of the chock shields, as described later.

Although the results from the short face panel were rather promising, it was decided to continue with the longer faces because the production potential of long faces is undoubtedly higher. In addition, the cost of geographical expansion for a short-face layout will be significantly higher since 200 m wide bord-and-pillar panels will have to separate the 120 m longwall panels to prevent possible uncontrolled failure of the dolerite over a large area.

Once all the chock shields had been modified, 11 additional four-leg shields were obtained to increase the face length on the third panel to 215 m. Improved roof support was experienced in this panel, although two falls occurred as a result of excessive slips in the face and extremely poor face retreat. It was also noticed that the dolerite sill over this longer face collapsed when the face had retreated only 200 m. Subsequent failures also occurred at shorter intervals, i.e. 40 to 50 m, even though the face retreat was very slow. The four-leg shields provide tip loads approximately 300 per cent higher than the comparative loads provided by the chock shields.

The shields were installed in the centre section of the face, and resulted in improved roof conditions and excellent flushing protection there. However, it appears that their support resistance varies if the rear legs are wrongly operated. This occurs because the front and rear legs have to be operated independently when a roof cavity exists between the front legs and the hinge point of the canopy and goaf shield. If all four legs are pressurized simultaneously under this condition, the canopy tip tends to drop away from the immediate roof, leaving the roof between the face and the front legs unsupported.

In order to ensure safe longwalling conditions, the theory has been put forward by the rock mechanics personnel of the Chamber of Mines that either of the two following procedures must be applied:

- ensure that the dolerite breaks in a controlled manner, or
- ensure that the dolerite does not break.

Investigations carried out by the rock mechanics personnel have indicated that, to ensure the above conditions, the following longwall face lengths must be adhered to:

- to ensure a controlled dolerite break, the face length should be at least 200 m;
- to ensure that the dolerite does not break, the face length should not exceed 120 m with longwall panels being separated by bord-and-pillar panels with an approximate span of 200 m and pillars of approximately 30 m by 30 m.

In connection with the first condition, Coalbrook was advised to increase the face length from 200 m (first panel) to 215 m (third panel) to ensure earlier initial and subsequent regular failure of the dolerite.

Production potential, cost of geographic expansion, ease of supervision, and the implications and handling of the inflow of groundwater are being scrutinized at present so that the optimum layout can be selected.

Gate Road Support

Systematic roof support in the development roads consists of 1,8 m by 19 mm (diameter) full column resin roofbolts with headboards installed at 1,5 m centres. Roof falls occasionally occur in these primary development roads when slips are present, or if the shale above no. 2 seam is exposed. Experience has shown that these falls require immediate re-bolting and support using steel sets and cribbing to improve stability when the longwall retreats through these fall areas. Research and experimentation are continuing in the bord-and-pillar sections at Coalbrook and elsewhere to improve roof bolting support and so prevent localized falls of roof.

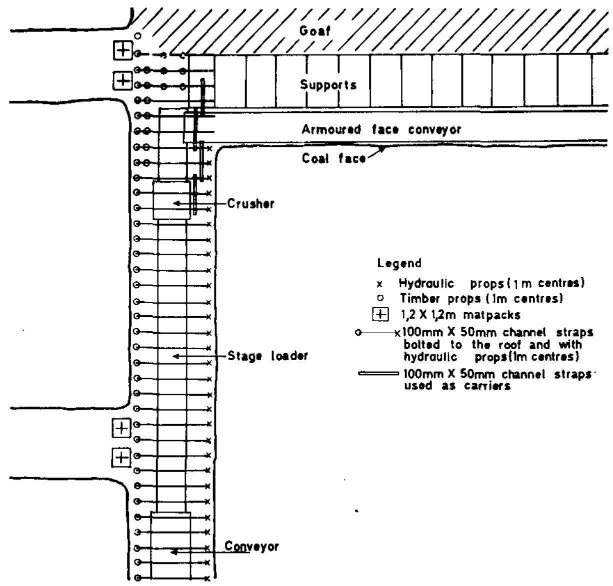

Fig. 9—Additional roof support in maingate

During retreat of the longwall face, several falls of ground were experienced in the maingate road. Since the stage loader is accommodated in this road, it is not possible to achieve the same support density with timbers as in the tailgate, where the timbers are placed at 1 m centres. At present the maingate road is supported as illustrated in Fig. 9.

Channels 100 mm by 50 mm by 5,0 m are bolted to the roof at 1 m centres. A single row of hydraulic props is set under the channels between the stage loader and the face side of the roadway, while timber props are used on the opposite side. The hydraulic props are systematically replaced with bearer sets on the face side. In the splits between the maingate and the maingate companion, the roof is supported with two mat packs between the pillar corners.

On the tailgate side, hydraulic props are set between the energy train and the face side of the roadway with a double row of timber props at 1 m intervals between the energy train and the other side. Timber props are also concentrated at 1 m spacing in the splits and between the energy train and goaf, as shown in Fig. 10.

Investigations are being conducted together with support manufacturers to provide a self-advancing support system to cover the area ahead of the face supports in the maingate road. It was considered that a significant improvement could be made if the face supports were extended across the gate to the rib side. These supports could have extended canopies to cover the armoured face conveyor–stage loader intersection. Not only would the support density be an improvement over the hand-set supports, but the labour requirement would be significantly reduced. However, because of the particular intersection used at Coalbrook, where the stage loader is advanced only every 3 webs, such a support could be as much as 2,25 m behind the rest of the face, thus largely removing any benefit it might have.

To overcome this problem and still extend powered

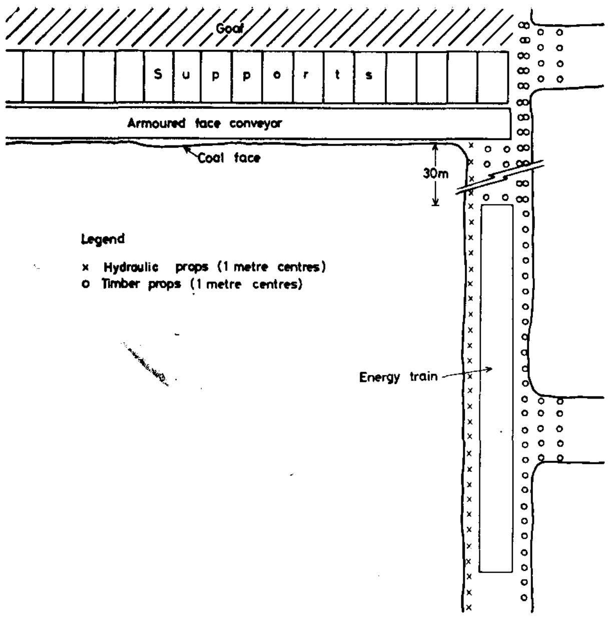

Fig. 10—Additional roof support in tailgate

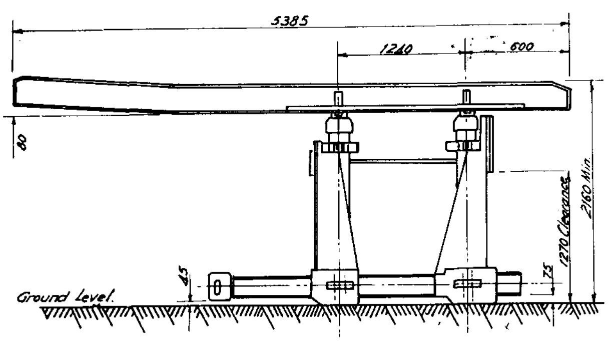

Fig. 11—Elevation of a straddle chock

support to the rib side of the main gate, a straddle chock (Figs. 11 and 12) is to be installed. As far as is practicable, its hydraulic components are identical to the run of face supports. It consists of two base pontoons linked together across the stage loader so as not to interfere with the advance every 3 webs. Two hydraulic legs are resiliently mounted on each of these pontoons and support two canopy beams. The canopy beams have extended cantilevers that stretch across the armoured face conveyor, and are capable of being set under the channels that support the gate before the hydraulic prop has to be removed to allow the conveyor to be advanced. The chock is advanced by a ram mounted in the left-hand pontoon and attached to the delivery end of the conveyor, thus allowing the chock to be advanced with the conveyor and not to be affected by the position of the stage loader. As this means it is pulled forward on 'the skew', it is stabilized by a guide rail running on the base of no. 1 chock shield to its left. Should it be necessary, a second advancing ram can be installed in the right-hand pontoon and attached to the stage loader by an adjustable chain.

Flushing

Flushing of the goaf material between the chock shields restricts the travelling way between the legs and causes damage to electric cables and hoses. After unsuccessful trials with conveyor belting fitted between the supports, steel plates designed and manufactured locally were bolted on the sides of the chock shields as described later. These side plates proved to be successful in keeping approximately 80 per cent of the flushing back. The four-leg shields in use are fitted with hydraulically operated flushing shields, which appear to be very effective in retaining the flushing.

Spalling

Excessive spalling of the face in very large slabs presents a major conveying problem on the armoured face conveyor. These slabs cause blockages on the conveyor or at the transfer point onto the stage loader. Their conveyance is often also restricted by the tunnel opening underneath the shearer.

Paving breakers are at present in use to break up large slabs on the armoured face conveyor. A small crusher attached to the tailgate end of the shearer for this purpose proved to be ineffective. The performance of larger crushers for this purpose is being investigated.

Ventilation

The longwall face is ventilated as shown in Fig. 13. Concentrations of methane in the goaf are drained at the balloon stopping in the tailgate companion. Intake air is passed over the energy train in the tailgate to keep the temperature of the transformers down and to prevent a build-up of methane at the tailgate drive.

Consideration is being given to the adoption of a bleeder system of ventilation as illustrated in Fig. 14. This method of ventilation would enable accumulations of groundwater in the goaf to be pumped from the lower elevations alongside the goafed area.

Cutting

Three sets of drums are being used on a rotational basis, i.e. one set on the shearer, a spare set, and one set for major overhauling. One set, i.e. three spiral vane drums, has been specially designed to cope with the stone intrusions experienced in the no. 1 seam. Pick performance has varied from approximately 15 t per pick in the no. 1 seam to 350 t per pick in the no. 2 seam. Slug-insert picks were found to be more suitable than the parrot-beak type in the tougher cutting conditions in the no. 1 seam. The coal in the no. 1 seam has a compressive strength of 47,6 MPa, compared with

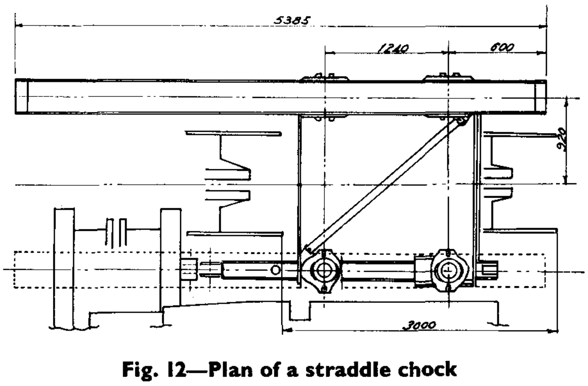

Fig. 12—Plan of a straddle chock

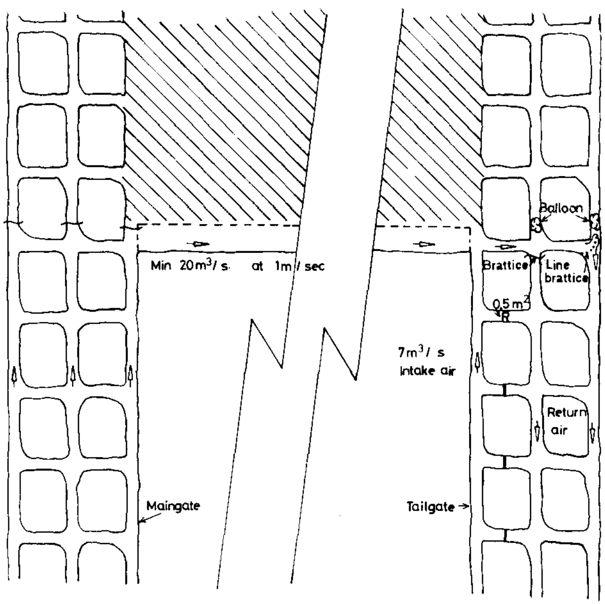

Fig. 13—Layout of the ventilation

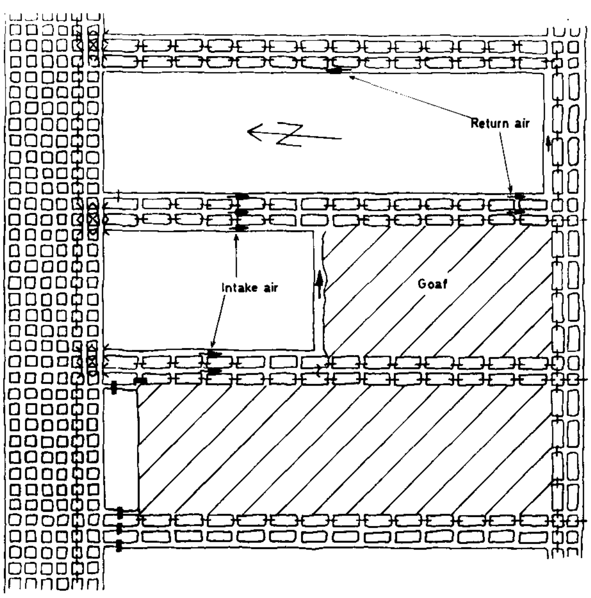

Fig. 14—Bleeder system of ventilation

40,8 MPa in the no. 2 seam. The Hardgrove index for both seams is approximately 60. The difficult cutting conditions experienced when stone intrusions occurred in the no. 1 seam make this seam less attractive for future longwalling.

Dust Control

Dust readings indicate a concentration of approximately 160 PER at the shearer operator, compared with 100 PER at the continuous miner and 80 PER at the conventional coal cutter. The prime source of dust is the cutting action of the shearer. The first set of drums purchased was fitted with a perforated pipe on the disc scroll. The disadvantage of this system is that the scroll, when cutting, is covered with coal, which prevents the water from getting to the picks. Drums subsequently purchased were of the pick-point flushing type with improved dust-suppression characteristics. Research is being conducted on an industry-wide basis to minimize the production of dust and to allay the dust produced in continuous miner sections. Developments in this respect could assist in reducing the comparatively high concentrations of dust experienced on the longwall face.

Handling of Water Inflow

The overlying strata consist of layers of Karoo shales and sandstones intruded by a dolerite sill with an average thickness of 40 m, as illustrated in Figs. 4 and 15.

Very few primary sources of groundwater are normally encountered on the horizon at which the coal is being mined. The permeability of the overlying and underlying rocks is very low, and there is no leakage of groundwater into the mine from sources other than isolated cracks.

However, groundwater does occur in the strata overlying the mine workings. This water is normally encountered in surface boreholes at depths between 15 and 70 m. The water is contained mainly in cracks in the sandstone, and high yields can sometimes be expected from the contact zone between the sandstone and the dolerite sill.

When the overlying strata collapse, an inflow of groundwater is experienced in the longwall. The water mainly accumulates in the goaf, from where it flows into the face if the gradient permits. This inflow into the face causes severe difficulties in the operation of the longwall. The inflow into the third panel was handled in the following way: the face profile was cut with the lowest point near the tailgate but with sufficient distance from the tailgate drive to prevent flooding of the motor. The inflow was pumped from this low point. Where the gradient permits, the inflow water was pumped from the gate roads behind the face line. When a constant rate of shearing was maintained, most of the inflow was removed via the armoured face conveyor before it could accumulate at the low point for pumping. The water presented considerable problems on the outbye conveyor system and on the surface crushing and screening plant. A slot was made in the tail-end section

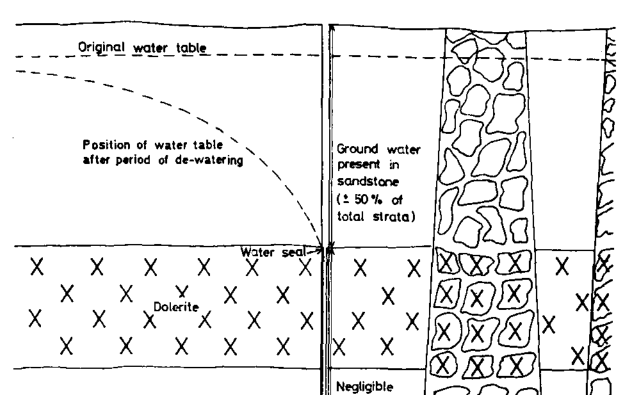

Fig. 15—Section through a dewatering borehole and panel

of the armoured face conveyor to discharge some of the water collected by the bottom flight chain into a sump in the tailgate road. The fine duff expelled with this water caused a build-up in the tailgate area. An attempt was made to remove this duff with a conventional dam scraper powered by an air hoist. Consideration is now being given to the following aspects in order to improve the handling of water inflow on the face:

— the installation of pumping facilities at the lower elevations of the panel alongside the goafed area when the bleeder ventilation system is implemented.
— the use of perforated sections in the conveyor system and or cyclones to collect the water.

To limit the inflow of water to the longwall faces, a dewatering system as shown in Figs. 15 and 16 is envisaged. Dewatering boreholes will be drilled from surface through the water-bearing strata into the workings. These holes will be spaced around four or more panels at intervals to collect the inflow from the water-bearing strata. The holes will be equipped and connected to an underground piping system to allow controlled drainage and usage of the water.

Five holes were recently drilled on the eastern side of the present panel to intercept groundwater entering from that side. Unfortunately, none of these holes intersected cracks in the water-bearing strata. By the drilling of inclined holes or deflection techniques an attempt will be made to intersect the water-bearing fissures. A limited inflow of groundwater is being experienced on the present panel, but with no detrimental effect on production.

Transport of Coal

Longwalling is a method of mining capable of producing high tonnages, and necessitates a high-capacity conveying system to ensure that operations are carried out on a continuous basis.

The installed power of the armoured face conveyor (i.e., two 250kW motors) has proved inadequate since the face length was increased from the original 200 to 215 m. Overload situations occur when the production rate approaches 1000 t/h. The present motors will be replaced by two 350 kW motors with a 30 mm double-strand flight chain.

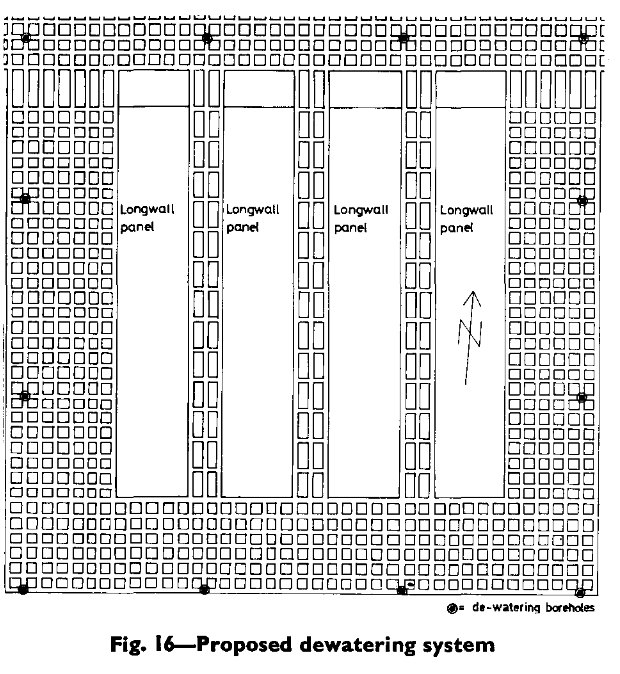

Fig. 16—Proposed dewatering system

The present panel conveyor is 1200 mm wide. It is equipped with an automatic loop take-up with storage for 150 m of belting, i.e. 75 m of retreat. In the method currently in use, the armoured face conveyor can be moved up three webs before the stage loader and conveyor tail-end must be retreated. The steering of the conveyor tail-end presents a problem, as described later.

Surges due to the spasmodic production characteristic of bord-and-pillar mining created trunk conveyor problems. Computer simulation was employed to determine the optimum size of a central collecting bunker from which production surges and the flow of coal onto the trunk system could be regulated. A 1000 t bunker and 1350 mm inbye main conveyor system were completed recently, with encouraging results. The main trunk conveyor system outbye of the underground bunker and the incline shaft conveyor were uprated to carry 1400 t/h. A 500 000 t open stockpile was constructed at the pit-head to accommodate a strategic stock of one month's consumption (i.e., 300 000 t), with the balance to cater for the loss of production when the longwall is moved, as well as for seasonal fluctuations in demand.

Longwall Move

The longwall move can basically be described as preparation of the face for moving before the end of the panel, the withdrawal of the equipment, the execution of the necessary overhauling and repair of equipment, and the transport and installation of the equipment in the next panel. The supports are drawn off under cover of a wire-mesh and rope protection reinforced by strategically placed girders and timber supports. All the face equipment is transported by load–haul–dump units Longwall moves are scheduled to be completed within a period of 45 days, including a draw-off preparation of 15 days.

Each move so far has presented its own unique problems, i.e. the modification of supports during the first move, and the repair of basecracks and fitting of flushing shields during the second move.

Poor roof conditions are experienced in the maingate and on the face during the preparation phase as a result of slow face retreat. Difficulties are experienced in removing the supports under these poor roof conditions. Potential ventilation problems were eliminated by the use of pre-developed airways through the barrier pillars to short-circuit air when required. Investigations are proceeding to reduce the moving time, especially since the mine has no duplicate sets of equipment (e.g. a complete armoured face conveyor) that could be installed beforehand.

Technical Problems

Roof Supports

Relay Bars and Rams. The rams of the supports are connected by means of relay bars to the armoured face conveyor. The tongue attachment pieces of the relay

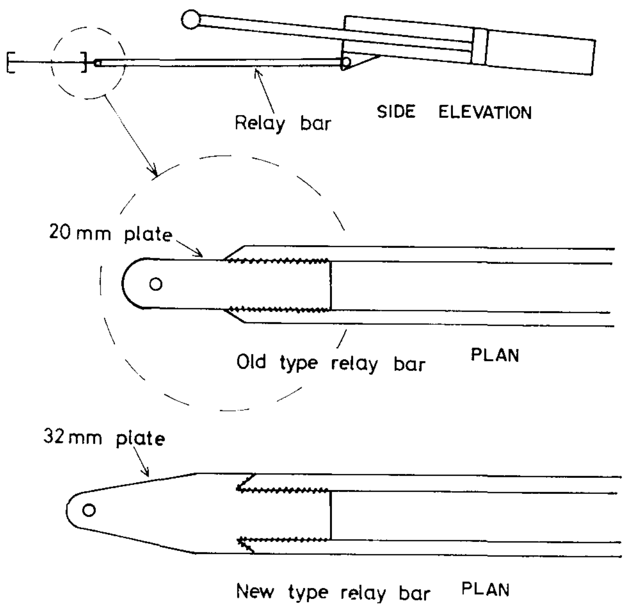

Fig. 17—Modifications to relay bar

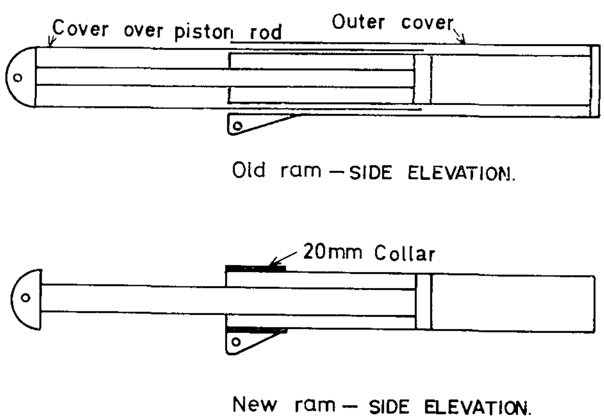

Fig. 18—Modifications to ram

bars were originally underdesigned and broke when the conveyor was pushed forward. The attachment pieces were redesigned and made strong enough to withstand the forces applied by the ram (Fig. 17). When this problem had been resolved, some of the piston rods of the rams were bent. The ram was redesigned with a piston rod that is stiff enough to suffice on its own, and a separate cover that is attached to the support base was made to protect the piston rod and to guard against workmen being struck during operation. A 29 mm collar has been welded round the end of the cylinder to strengthen this area, which has also failed in the past (Fig. 18).

Capsules. The chock shield supports have an articulated canopy that facilitates good roof contact, the cantilever being operated by a hydraulic cylinder called a capsule. When the cantilever is taking load, the capsule is placed under compression. The piston end of the capsule is rounded and fits into a socket on the cantilever. When the capsule yields, the rounded end of the piston slips in the socket. Because of the materials used on the end of the piston rod and the socket, a stick–slip situation developed when the capsule yielded under pressure. This caused very high peak pressures and flowrates through the yield valve, and contributed towards the wearing of flats on the piston ends, which aggravated the situation. This was overcome by the use of a different metal for the rounded ends of the capsule. The poppets in the yield valves were also redesigned to reduce the rate of flow of the hydraulic fluid through the valve on yield.

Valve Gear. Another problem encountered was a drop in pilot pressure when too many operations were being carried out on the face at once, which caused the valves to chatter instead of opening cleanly. This in turn caused damage to the valve seats. This problem has been overcome by means of restrictors placed in the hydraulic circuit.

Bases of Chock Shields. The most serious defect of the supports was the original underdesign of the welding where the base portions join the heavy plate that supports the lemniscate links for the shield. This problem was aggravated by steps cut in the floor and adverse roof conditions, which occasionally caused the back legs to become steelbound. The two plates now welded onto the bases causes the lines of stress to flow more evenly from the 50 mm plate into the formed 20 mm plate, and thereby relieve the high stress point. This has proved to be successful (Figs. 19 and 20).

Flushing Protection. The supports are designed so

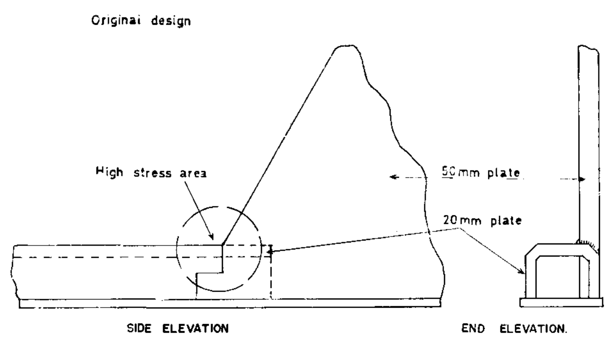

Fig. 19—Problem area on base of chock shields

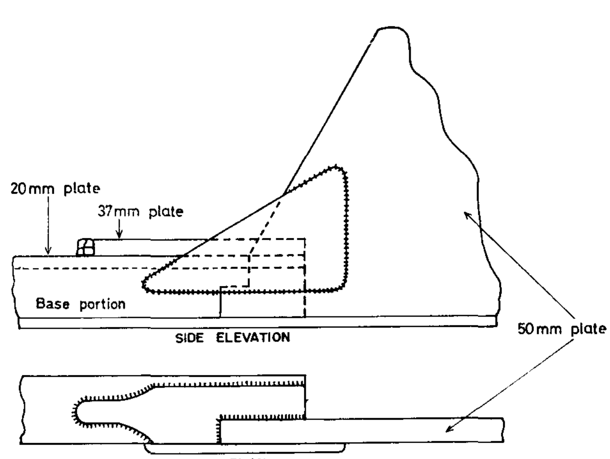

Fig. 20—Modifications to base of chock shields

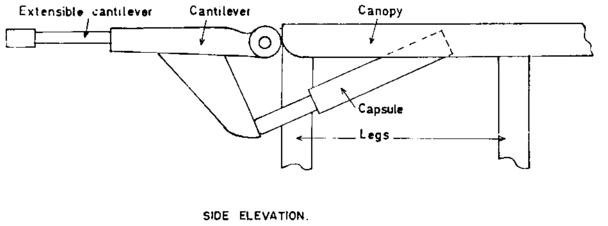

Fig. 21—Side elevation of modified canopy and cantilever

that there is a gap of 50 mm between adjacent support canopies and a gap of 200 mm between the goaf shields. Flushing as described earlier enters the walk way through these gaps. Plates were bent and fixed to the sides of the goaf shield to reduce the gap to 50 mm.

Cantilever Tip Loads. As explained earlier, the load that the cantilevers could support (i.e., 10 t) was not sufficient. The supports were modified by use of a bigger cantilever capsule and a greater leverage to increase the cantilever resistance. Also, the cantilevers were replaced with new ones that were provided with extensible cantilever extensions (Fig. 21). The tip load on the extended cantilever was increased to 30 t, and in the retracted position, to 46 t. The cantilever is extended to support the roof immediately behind the top shearer drum. After the armoured face conveyor has been snaked in behind the shearer, the extensions are retracted when the supports move up to the conveyor.

Shearer

Crusher. As mentioned earlier, a small crusher was attached to the tailgate end of the shearer to break big lumps of coal that occur on that side of the shearer so that they can pass under the shearer. Initially problems were encountered with water getting into the motor, and, once the motor had been suitably sealed, it was found that the crusher was ineffective.

Bearing Failure on Ranging Arms. The cylindrical roller bearings in the ranging arm failed. The manufacturer redesigned the planetary gearing in the ranging arm so that bigger bearings could be installed to alleviate this problem.

Tilting Mechanism. The shearer is fitted with a tilting underframe to negotiate sudden changes in the cutting horizon. The tilting mechanism and underframe had to be redesigned to prevent failure of the tilting cams.

Cowls. The mechanism used to swivel the cowls over when the direction of shearing is changed caused many delays owing to duff getting into the chain mechanism. The first set of cowls wore very badly and had to be rebuilt frequently. Cowls of a heavier section were then designed and manufactured locally. The cowls are swivelled by being pushed against the floor while the shearer is moving along.

Cutting Drums and Picks. During the first two panels, which were in the no. 2 seam, the shearer was fitted with two-start drums with a pick spacing of 50 mm. The drums were fitted with radial type slug-insert picks. These drums and picks were very successful in the no. 2 seam. Cutting in the no. 1 seam was much more difficult, causing excessive wear to the pick boxes and picks. Three-start drums with a 30 mm pick spacing were purchased, and these, together with radial-type slug-insert picks, proved to be most successful in the tough cutting conditions, especially where stone intrusions occurred. The parrot-beak type picks used in the no. 1 seam proved unsuccessful. Pick performance was about 15 t per pick.

Armoured Face Conveyor

Ramp Plates. The armoured face conveyor is designed to take the weight of the shearer on the ramp plates in front of the conveyor and the tubular trapping behind it. The original design of the ramp plates was poor, and the path on which the shearer travelled broke off. The ramp plates were redesigned with sufficient strength (Fig. 22).

Pan Connectors. The other major problem with the armoured face conveyor concerns the pan connectors. These are dogbone-shaped pins that are secured in the pans by clips, which are designed to break before the pins break. These clips and pins were not strong enough to withstand the full force of the advancing rams of the supports. Stronger clips are being tested.

Stage Loader. The head-end of the stage loader is attached to the tail-end of the gate conveyor, which is retracted by an automatic take-up device situated at the drive-end of the conveyor. The purpose of this arrangement is to facilitate retreat without stoppages for any significant time for removal of sections of conveyor structure and belting. Owing to the mass of the crusher, the stage loader tends to move off line when it is retreated where the floor is sloping, and by so doing it drags the gate conveyor tail-end off line as well. Proper alignment is necessary to avoid damage to the edges of the conveyor belting.

Organizational Problems

Longwall Organization

The longwall operation and the longwall development

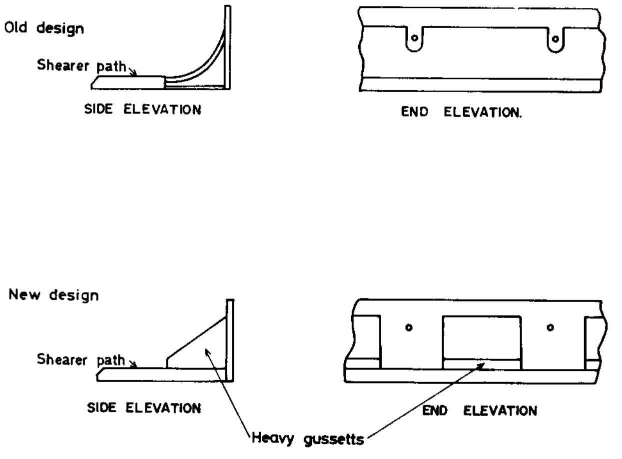

Fig. 22—Modification to ramp plate

operation (usually two continuous miner sections) are supervised by an underground manager, section engineer, mine overseer, and senior foreman. Both the underground manager and the section engineer report to the production manager of the mine. Two foremen, one mechanical and one electrical, exercise day-shift supervision on the longwall only, while a shiftboss is available on each shift for the longwall only. In addition, supervision is exercised by the miner and shift supervisor.

Since the implementation of the method in 1976, it has been found necessary to increase the fitters and electricians from 3 each to 6 each. In addition, 7 operators and 10 assistants are employed on each shift.

Training

Training of longwall personnel is regarded as an important function of line management. Training manuals were compiled, and the training is carried out by the mining and engineering training officers and supervisors.

As the need arose to acquire the necessary expertise in a specific problem area, technical consultants were brought to South Africa. An extensive training programme is being conducted to improve engineering expertise in conjunction with the manufacturers. Two senior foremen, two foremen, and the engineering training officer were sent to Germany for an extensive 12-week training programme. The operators on the shearer, supports, and stage loader acquainted themselves with the operation in a relatively short period and perform well.

Back-up Service

Since the original installation, great difficulties have at times been encountered in regard to the provision of expertise and spares for the equipment obtained from Germany. Delays in the provision of parts have resulted in some of the more urgent spares having to be flown in at premium charges. There has been a significant improvement in back-up service since the recent change of agency for this equipment, which has resolved many of the problems that existed previously.

Local Manufacture

Some progress has been made in this respect in certain areas, although other areas, e.g. the manufacture of crown and pinion gears for the drives and chain of the armoured face conveyor, U-bolts, and shearer picks, are still a cause of concern.

Future of Longwalling at Coalbrook

The performance of the longwall at Coalbrook has been less than spectacular to date, but it has proved a safe method capable of coping with the most arduous mining conditions. The cost has proved to be more than competitive with those of conventional mechanization and continuous mining.

In spite of all the teething troubles experienced in the first panel and the serious problems experienced with the supports, the average production for that panel was approximately 55 000 t/m (moves excluded), with a best output of 115 000 t over a 30-day period.

On the short face (120 m long), the average production was 72 000 t/m, with a best month of 105 000 t. Although the theoretical production potential of the short face (120 m) is less, and the cost of geographic expansion greater as a result of having to leave 200 m wide bord-and-pillar panels between longwall panels, consideration will be given to the mining of short faces if the long faces do not come up to expectation.

The cutting difficulties and inflow of groundwater experienced in the no. 1 seam panel unfortunately clouded the potential of longwalling in this seam. On average, the longwall produced approximately 50 000 t/m. Investigations are in progress in an attempt to solve these problems to the extent that longwalling can later be resumed in the higher-quality no. 1 seam coal. Present exploration techniques are being studied in order to accurately establish the extent of stone intrusions in this seam.

So that the production and economic viability of longwalling can be assessed on a long face, the fourth panel has a face length of 215 m and is mined in the no. 2 seam. The results, given in Table I, are very encouraging.

Should longwalling in the fourth panel prove successful and future coal requirements by Escom warrant it, consideration will be given to the acquisition of a second installation to replace the remaining conventional mechanized sections. Consideration will also be given to double-seam longwalling to improve the heat value of the coal recovered and the percentage recovery.

In spite of the existing problem areas, the management of Coalbrook Collieries is confident that the longwall mining method affords the opportunity to increase the percentage extraction and the heat value of the coal produced, to improve productivity, and to produce coal safely at a competitive cost.

Acknowledgements

The author expresses his gratitude to the management of the Coal Division of General Mining and Finance Corporation for permission to publish this paper. He thanks the staff of Coalbrook Collieries for their assistance in its preparation, and, on behalf of the Coal Division of General Mining & Finance Corporation Limited, he expresses appreciation to the management of Escom for their sustained support in finding a mining method to extract the seams in the Vaal Basin safely and economically, as well as optimizing the recovery of valuable coal reserves in South Africa.

Contribution to preceding paper

by C. J. CLOETE*, Pr. Eng., B.Sc. Eng. (Min.) (Member)

Introduction

To evaluate the performance of the longwall operation at Sigma, it must be noted that conditions and equipment there differ considerably from those at Coalbrook.

As far as geological conditions are concerned, it must be noted that longwalling is carried out at Sigma in the no. 3 coal seam, whilst at Coalbrook it is practised in the no. 2 and no. 1 coal seams. The floor of the no. 3 coal seam at Sigma consists of a micaceous shale that grades into a fine-grained sandstone. The roof consists of a mudstone, followed by a thin coal seam known as the coal marker seam no. 1. This is followed by a fine- to medium-grained sandstone and the coal marker seam no. 2. The rest of the parting between the no. 3 coal seam and the overlying dolerite sill consists mainly of sandstone and shale. The sediments above the dolerite sill are mainly sandstone. The average thickness of the dolerite sill is approximately 40 m, and the top of the sill can be found between outcrop and a depth of 20 m. The parting distance between the roof of the no. 3 coal seam and the bottom of the dolerite varies between 50 and 80 m. The floor depth of the no. 3 coal seam in the area where longwalling is practised is approximately 115 m.

The equipment being used at Sigma consists of Klöckner Becorit two-legged shields with hydraulic components, Klöckner Becorit twin-chain armoured face conveyor with carrier frame, and Klöckner Becorit stage loader and crusher. The shearer is an AM500 mark II 750 kW double-ended ranging drum type with tilting facility and lump breaker.

Background

One longwall unit has been in operation on the no. 3 seam horizon since August 1975. The extraction of the fourth panel by this method is now nearing completion.

The width of the panels has been the same, namely 200 m, but the length has varied between 990 and 1270 m. Where conditions are favourable, longer panels will be developed in future. A total of $4\frac{1}{2}$ million tons has been derived from longwall mining to date, and this represents 19 per cent of the total mine production over this period.

Present Longwall Panel

The fourth panel was commenced with a more powerful shearer. To cope with the increased production from the shearer and to eliminate known shortcomings in the equipment, suitable modifications were made at the same time to the panzer face conveyor and crusher system. Calculations for the revised system showed that the possible production potential could consequently be raised by 41 per cent from 27 500 to 38 500 tons per week. This objective is high and to our knowledge not being obtained elsewhere in the world on a regular basis.

Production Performance

Realizing that much still had to be learnt about the system, management provisionally set the production target at 35 400 tons per week or 92 per cent of potential. In the light of subsequent experience this target is attainable. The average weekly output for panels 1 to 3 was approximately 17 000, 20 000, and 18 000 tons respectively, that of panel 4 to date being approximately 25 000 tons. With the improved equipment in panel 4, the average weekly production rose by 38 per cent (against the expected 41 per cent) over that for the first three panels. The highest weekly output from the fourth panel has been 44 000 tons per week. During the calendar month of June, a world record of 164 557 tons was obtained from the longwall face, which was 6815 tons more than the Japanese record set up in January 1979. Table I shows the frequency distribution of production levels achieved in the panels worked. In the conventional operation, daily production is much more steady than in longwalling.

Comparison of Longwall and Conventional Mining

Longwalling tends to be an inflexible system owing to the high degree of interdependence of personnel performance, geological conditions, and equipment capability. In conventional operations, short-term failure or less-than-adequate performance of man or machine, or somewhat adverse geological conditions, will not necessarily bring production to a standstill, as is fairly often the case with longwalling.

Furthermore, in conventional mining, the replacement of equipment or the augmentation of labour can often help to maintain the required production level under adverse conditions. This is not the case with longwalling. However, longwalling has a production potential of three to four times that of an average conventional unit.

Problems Encountered

Each of the four longwall panels worked had characteristic problems, which had to be overcome by training, or by modifications or additions to equipment. In panel one, inexperience with the system, together with the falling of large lumps of coal from the face (for which there was no adequate breaking facility), presented major problems. Furthermore, it soon became evident that the 400 kW shearer was underpowered; this remained a problem until the shearer was replaced at the commencement of panel 4.

In the second panel, the problems were similar to those experienced in the no. 1 panel, because a suitable shearer and crusher were not available. The dolerite sill started to collapse at greater intervals, and caused roof control difficulties as a result of the excessive pressure that was exerted on the face at these times. The reasons for this lack of failure of the dolerite sill might be the

*Sigma Colliery, Sasol One (Pty) Ltd, Sasolburg, Orange Free State.

TABLE I

FREQUENCY DISTRIBUTION OF PRODUCTION LEVELS (EXCLUDING MOVES)

Tonnage per week	Under 10 000	10 000–20 000	20 000–30 000	30 000–40 000	Over 40 000	Average weekly tonnage per panel
Panel 1	8	63	27	2	0	17 151
Panel 2	9	29	62	0	0	19 598
Panel 3	24	25	47	4	0	17 787
Panel 4	10	12	50	25	3	24 924
Panel 1 to 4	13	33	46	7	1	19 494

increase of the parting thickness between the sill and the coal seam, and the dome structure of the sill.

In the third panel, geological dislocations led to roof falls and difficulty in following the seam horizon. (Horizon control is important in the avoidance of excessive stress on equipment and of the inclusion of inert roof and floor material.) The inflow of groundwater also became a problem. This panel experienced even more difficulties, which arose from the lack of failure and the well-developed dome of the dolerite sill. This, together with a slow moving face, caused the tailgate to deteriorate and also resulted in excessive scaling of the barrier pillar. This required an abnormal amount of support in the tailgate using RSJs (rolled-steel joists) and wire netting in addition to trusses and hydraulic props.

In the fourth panel, experience had to be gained with the new and modified equipment in the first place. Secondly, the initial availability of the shearer was much lower than expected due to breakdowns arising from deficiencies in design and operation. The availability of the shearer has improved recently (see Table II). Thirdly, the influx of groundwater, together with the adverse seam gradient, at times resulted in flooding of equipment and hampered production. Lastly, the pans of the armoured face conveyor have been failing more frequently recently because of wear (see Table III). These pans were used in previous panels and have given good service, viz 1,9 to 2,8 million tons compared with suggested standards of 1 million tons per set.

Additional pumping facilities consisting of a compressed-air ring feed with CP pumps helped to ease the water problem. This problem will not be solved adequately unless some form of groundwater drainage is installed before the overlying strata are disturbed by mining. Such a drainage system is under consideration.

Availability of Equipment

In the fourth panel, the availability of the overall face production system to date has been 42 per cent, against a desirable and attainable objective of 60 per cent. The average availability of the better faces overseas is quoted as approximately 40 per cent. Table IV gives a breakdown of the Sigma figures. With the expected percentage availability of 60 per cent, weekly outputs of 40 000 tons and higher could be obtained.

Face Moves

The layout of the longwalling equipment at Sigma is shown in Fig. 1.

When long-term weekly production targets are set for the longwall face, the non-productive time while the equipment is being moved into a new panel must be taken into account. The transfer of conventional equipment from panel to panel is less time-consuming because of its mobility, and can usually be done between working shifts or during weekends. However, the time taken to move from one longwall panel to the next depends on items such as distance between panels, and the extent of the major repairs or modifications required to equipment. At Sigma the face move is done in two phases. A dummy road is cut parallel to the gate ends and one-third of the face width from the tailgate. The first phase involves the taking out of 33 per cent of the armoured face conveyor and shields on the tailgate side up to the dummy road. Thereafter, the face advances normally by the cutting of another 30m before the second phase is commenced, whereby the rest of the face equipment is extracted. The advantage of

TABLE II — SHEARER DOWNTIME (% of production time)*; TABLE III — ARMOURED FACE CONVEYOR DOWNTIME (% of production time)

	Table II (% of production time)*	Table III (% of production time)
Sep. 1978	15	16
Oct.	29	4
Nov.	21	4
Dec.	16	3
Jan 1979	22	3
Feb.	15	9
Mar.	19	14
Apr.	12	25
May	30†	13
Jun.	6	15
Average	19	11

*This percentage includes electric supply cable and haulage chain failures.

†Shearer out of commission for week owing to major part failure.

TABLE IV

AVAILABILITY ACHIEVED IN PANEL 4 AND EXPECTED AVAILABILITY IN PANEL 5 EXPRESSED AS A PERCENTAGE OF TOTAL PRODUCTION TIME

	Percentage actual	Percentage expected
Time available	42	60
Maintenance and extensions	5*	7
Loss for:		
— shearer	19	12
— armoured face conveyor	11	5
— hydraulic supports	3	3
— geological conditions	6	4
— conveyors	6	3
— pick changing	6	5
— supervision	2	1
	100	100

*This is less than expected because certain scheduled maintenance work is undertaken while other repair work is being done.

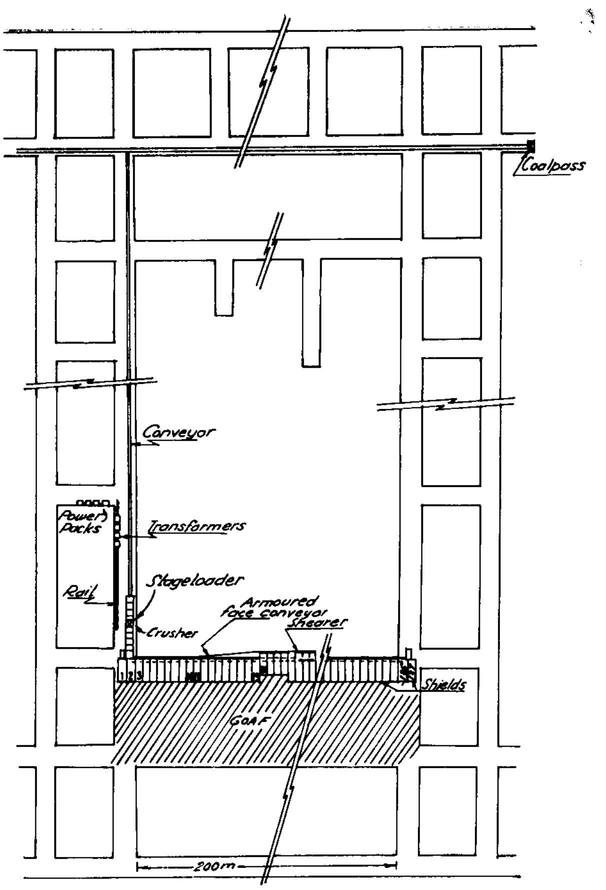

Fig. 1—Layout of longwall equipment at Sigma Colliery (not to scale)

this method is mainly the reduced pressure exerted on the face while moving, and less congestion of equipment underground and in the workshop. The total transfer period to the new face takes six weeks, of which the second phase (no production) lasts four weeks. The longer panel length envisaged in the future will reduce production loss due to transfers.

Face Productivity

The average face productivity in panel 4 (over a period of 43 weeks) is 65 tons per man shift, with a maximum of 113 tons per man shift over one week. For the future, improvements in longwall manpower productivity can be expected with improved knowledge of the system and equipment availability. Improved equipment availability can be achieved through better understanding of equipment limitations, the elimination of weaknesses, and improved operational skills.

Future Problems

The major problems foreseen at present, for which solutions are being sought, are related to the disturbance of the overburden through subsidence. These problems involve, firstly, surface damage and the possible loss to users of groundwater and, secondly, the flow of water into the workings, which hampers the operation. These problems will probably be compounded if the seams below the worked-out longwall panels are removed by the same method.

Agglomeration

The 3rd International Symposium of Agglomeration, which will be held from 6th to 9th May, 1981, in Nürnberg, West Germany, will continue the series of international Symposia on Agglomeration. The first symposium was held in Philadelphia in 1961, and the second in Atlanta in 1977.

It is the object of the 3rd International Symposium to introduce and discuss the latest works in various areas from fundamentals, through methods of agglomeration, up to operating plant practice. Among others, questions of raw materials, and process and mechanical engineering will be discussed.

The following are the topics to be discussed.

Aspects of Agglomeration

Fundamentals — binding mechanism, adhesion forces, kinetics modelling, reaction mechanism, compaction behaviour, role of binders, methods of measuring. Laboratory tests, plant and process design and automatic control, use of agglomerates.

Methods of Agglomeration

Granulating, balling, pelletizing in drums, pans, mixers, spray fluidized bed, etc., liquid phase agglomeration, instantization, tabletting, briquetting, compaction, extrusion in tabletting machines, roll presses, pellet presses, etc., sintering, heat hardening, and various novel methods.

Agglomeration of Products

Ores, minerals, ceramics, coal, fertilizers, cement, food, pharmaceuticals, chemical products, waste products, etc.

Further information is obtainable from NMA Nürnberger Messe- und Ausstellungsgesellschaft mbH, Messezentrum, D-8500 Nürnberg, West Germany.