

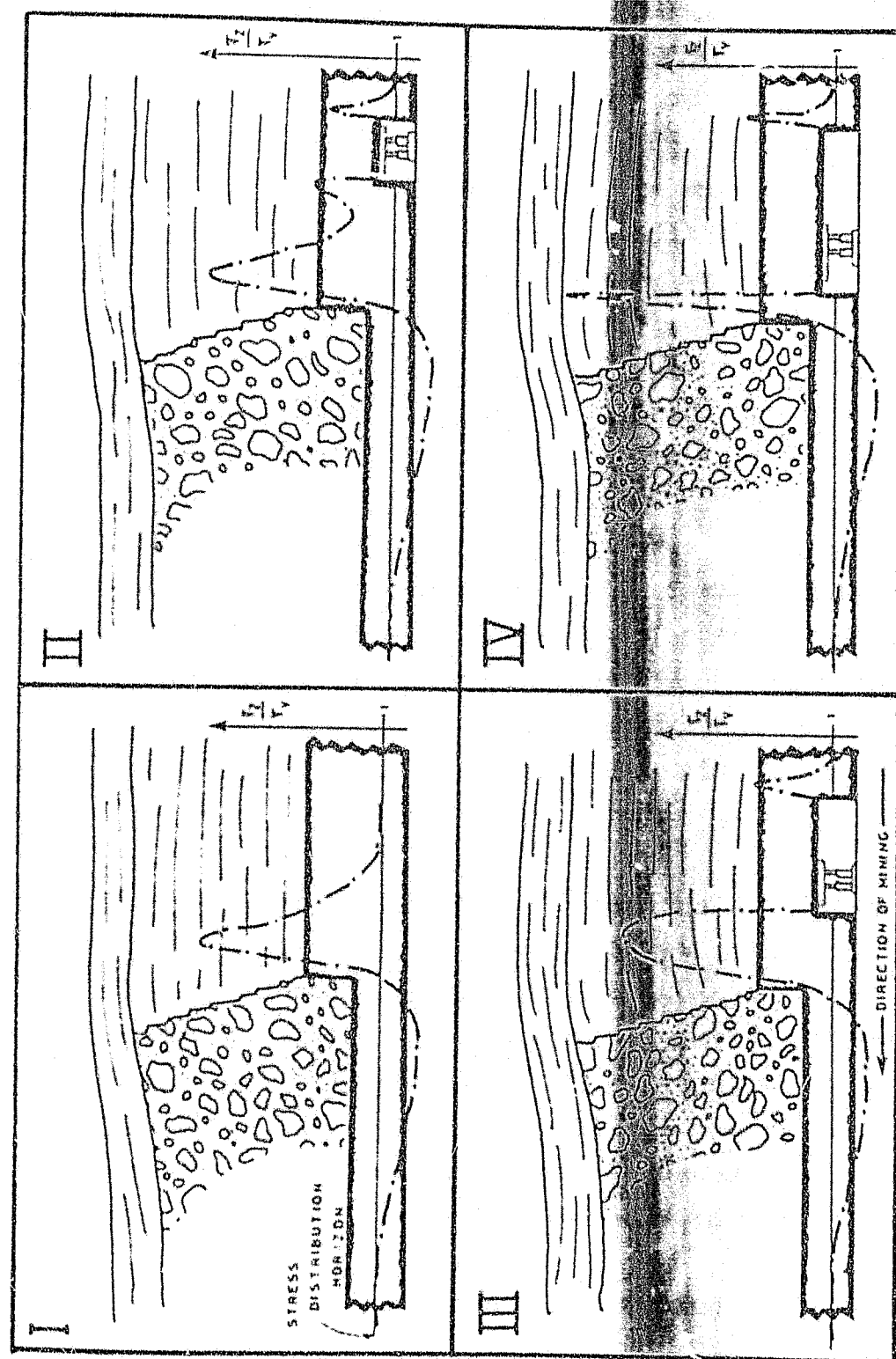


FIGURE 11 Formation of a highly stressed remnant pillar as a result of locating the lower slice starting face beneath the upper slice abutment.

4.3.3 Evaluative literature

Literature falling into this category is concerned primarily with evaluating the potential of particular thick seam mining methods under specific geologic and economic conditions. These publications usually give a very detailed description of the mining method being evaluated, and the geologic conditions to which it is best suited. Known applications of the method are reviewed, with emphasis being placed on the geologic conditions existing at each location as well as on the type of equipment in use, problems encountered with method, modifications required to the method, and the economic viability of the method. The geologic conditions of the site where it is proposed to employ this method are evaluated and, if the method is potentially suitable for use under these conditions, an economic evaluation is conducted. Thus, these publications usually give a comprehensive report on the method under review and are, therefore, often viewed as reference texts on that particular method.

The best known of such publications are those by Wilson et al. (1976) on multi-slice longwall mining, Callier (1972), Barron (1974), and Wallis et al. (1977), on longwall mining with sublevel caving and Grimley (1974), Parkes and Grimley (1975), Parkes and Grimley (1976) on hydraulic mining. In view of the comprehensiveness of these publications they are discussed under the sub-heading of the mining methods to which they refer.

4.3.3.1 Multi-slice longwall mining

Wilson et al. (1976) assessed the feasibility of multi-slice longwall mining methods for mining western United States thick seams, and, in particular, for the mining of a +7 m thick seam at Emery Mine.

The authors identified three tasks, which they approached in the following order:

- i) investigate multi-slice mining methods employed elsewhere in the world;
- ii) study the suitability of the Emery Mine site for multi-slice longwall mining;
- iii) perform an economic and marketing analysis of the proposed project to predict its competitive position within the industry.

When investigating multi-slice mining methods Wilson et al. categorised the many variations of this system according to whether slices were mined simultaneously or not, and to what was used to form a parting between the slices. In addition to studying these various methods, experience was also gained from visiting twelve thick seam operations in Britain, West Germany, Yugoslavia, Spain, France, Australia, South Africa and Japan.

After establishing and describing the various multi-slice mining methods the authors investigated the geology of the Emery Mine site. As a result of this study, several potential problem areas were identified and their significance briefly discussed.

In the next section of the report, concerned with mine design considerations, these problems were discussed further. In addition, considerable attention was given to entry layout designs which, by United States regulations, must be multiple entry systems. Three multiple entry layouts were selected as being the most suitable, and these were analysed from a rock mechanics viewpoint using finite element analysis. Wilson et al. used the results of the analysis to verify the location of the entries and to determine which of the three layouts was the most stable.

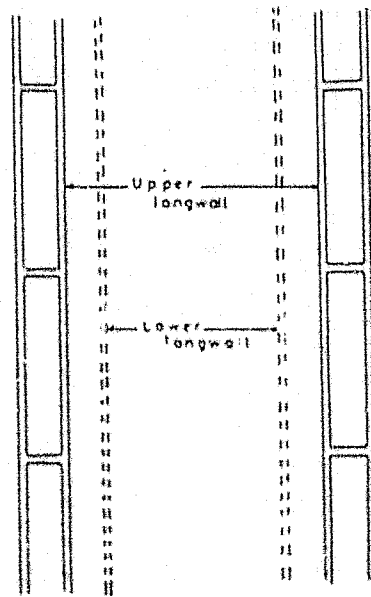
However, a number of anomalies appear in the stress distribution profiles derived from the finite element analysis. These have not been detected by the authors and, in fact, some of the authors' conclusions have been based upon these anomalies. The stress profiles associated with the proposed entry layouts are shown in Figs. 12, 13 and 14, with anomalous regions being labelled A to D. In these figures, which are taken from the authors' paper, the top slice interpanel pillars are shown, incorrectly, to be of different widths. In the following discussion of the anomalous regions it should be appreciated that the stress distribution profiles under consideration were computed for horizons through the mid-mining height of the lower longwall slices.

Region A - Figs. 12 and 14

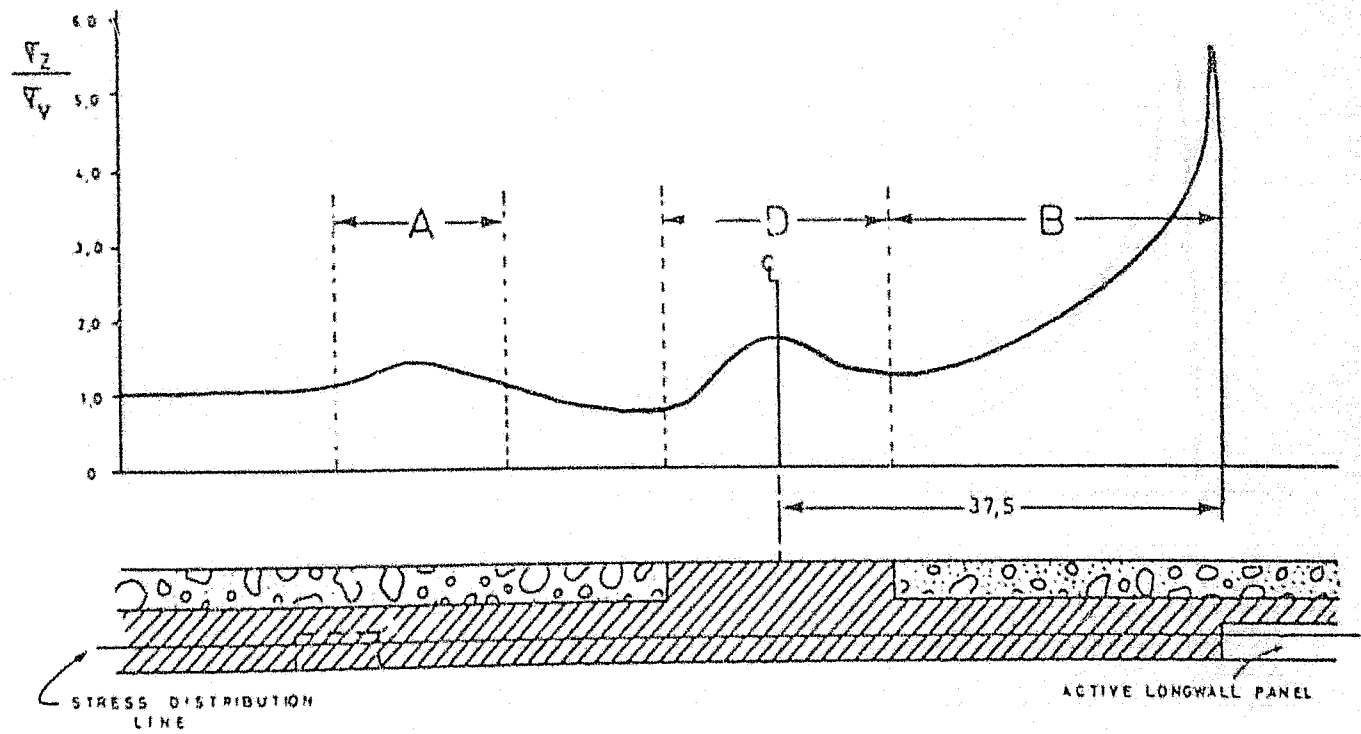
This region lies immediately below the goaf of an upper slice longwall panel. Thus, on the basis of the discussion presented in Appendix II, the maximum value of the stress concentration factor, σ_z/σ_v , should not exceed 1. However, the authors have shown this factor to be in the order of 1.5, and take cognisance of this fact by suggesting that the panel development entries for the lower slice should be located beyond this region.

Region B - Figs. 12 and 13

The stress concentration factor in this region increases rapidly to attain a magnitude of almost 6 as the lower slice panel entry is approached. However, the horizon for which the stress profile is constructed is located only a few metres below goaf. Thus, once again, stress concentration factors along this horizon should not exceed 1.

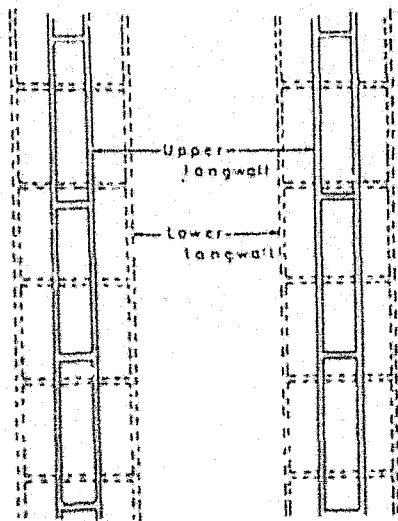


(a) Longwall panel layout with two single entries for lower slice and double entries for upper slice.

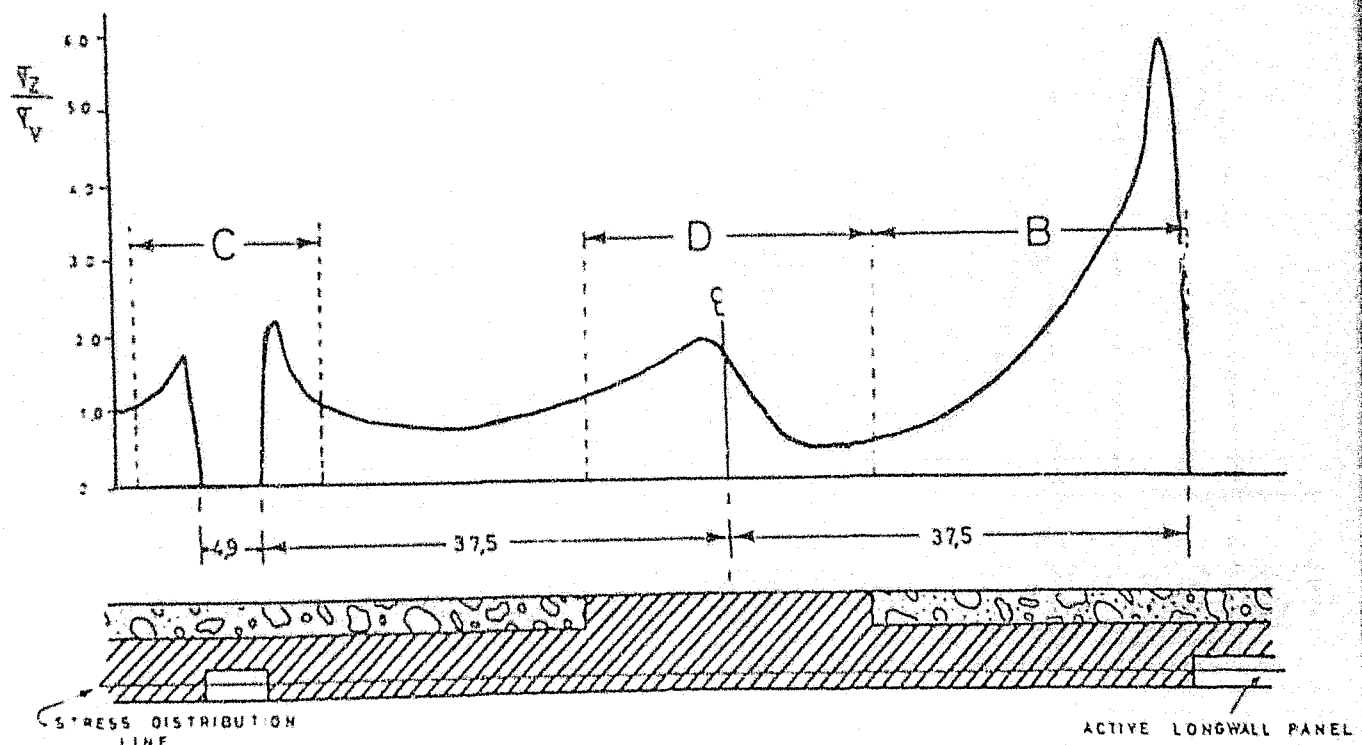


(b) Stress distribution profile through mid-height of lower slice interpanel pillar for panel entry layout shown in (a).

FIGURE 12 Stress distribution profile associated with a longwall panel layout comprising two single entries for lower slice and double entries for upper slice as presented by Wilson et al. (1976).

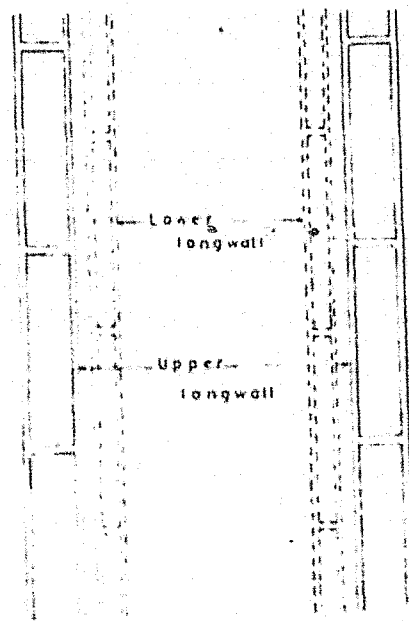


(a) Longwall panel layout with double entry gateroads for upper and lower slices.

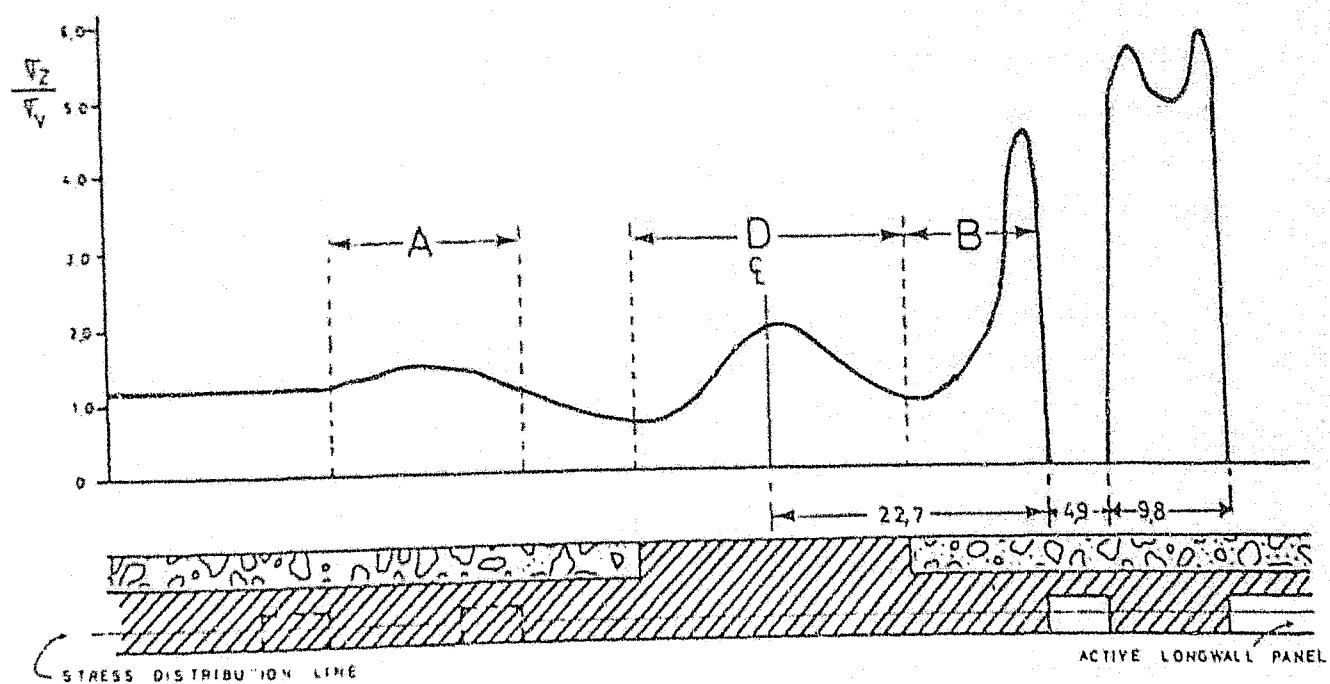


(b) Stress distribution profile through mid-height of lower slice interpanel pillar for panel entry layout shown in (a).

FIGURE 13 Stress distribution profile associated with a longwall panel layout comprising double entry gateroads for the upper and lower slices as presented by Wilson *et al.* (1976).



(a) Longwall panel layout with two double entries for the lower slice and double entries for the top slice.



(b) Stress distribution profile through mid-height of lower slice interpanel pillar for panel entry layout shown in (a).

FIGURE 14 Stress distribution profile associated with a longwall panel layout comprising two double entries for the lower slice and double entries for the top slice as presented by Wilson *et al.* (1976).

Region C - Fig. 13

In this region the stress distribution profile on either side of the lower slice panel entry is shown to be non-symmetrical, with a higher stress concentration existing on the interpanel pillar side of the entry than on the proposed mining side of the entry. But, as reference to Fig. II.5 or II.6* highlights, if the entry is sufficiently far removed from the interpanel pillar a lower stress concentration should exist on the interpanel pillar side of the entry. It is interesting to note that this later statement is supported by the stress distribution profile over region C, Fig. 14, presented by the authors.

Region D - Figs. 12, 13 and 14

This region comprises that portion of the lower slice interpanel pillar which is located immediately below the upper slice interpanel pillar. On the basis of previous considerations (refer Fig. II.6 and Fig. 9) the stress distribution profile through this region should be almost symmetrical. This is clearly not the case in Figs. 12, 13 and 14. Furthermore, a considerable portion of the interpanel pillar associated with the entry layout illustrated in Fig. 13 is actually destressed. The stress concentration factor over above one-third of this pillar is only of the order of 0.5. The authors accept this stress distribution profile and state that the destressed area will be subjected to tension and buckling, and that roadways located in the area can be expected to be unstable as the lower slice longwall approaches.

* Figs., Tables, Eqns., Plates etc. which are captioned in Roman numerals in this main text are contained in the Appendix captioned by the same Roman numerals.

The anomalies in the stress profiles derived from the finite element analysis may possibly be accounted for by the unrealistic assumption which the authors made concerning the development of the goaf in the upper slice. This assumption states that the goaf from the longwall in the top slice settles in such a way that triangular openings of dimension 20 m (64 ft) by 2,4 m (8 ft) are left beside the interpanel pillar, Fig. 15. The presence of these openings could influence significantly the distribution of stresses in the immediate vicinity.

After considering the state of stress associated with the various panel entry outlays, and the support of the lower slice entries, the authors discuss the selection of the mining equipment. Since the most important decision to be made in the area of equipment selection for a multi-slice operation is that of face support, considerable attention is given to the various types of support systems available. Good roof support, particularly when mining the lower slices in multi-slice mining, is essential.

Based on the preceding discussions concerning the many variations of multi-slice longwall mining, the geology of the proposed mine site, the design of the panel entries, and the longwall mining equipment available, the authors decide on the actual mining method, mine design, and mining equipment to be used. In order to ensure that the proposed mining method would be economically viable and acceptable to the coal mining industry the number of radical changes encompassed within the method have been kept to a minimum. This has been achieved at the cost of percentage extraction.

Finally, the health, safety and environmental impacts of the mining method are considered and the paper concludes with an assessment of the economic feasibility of the

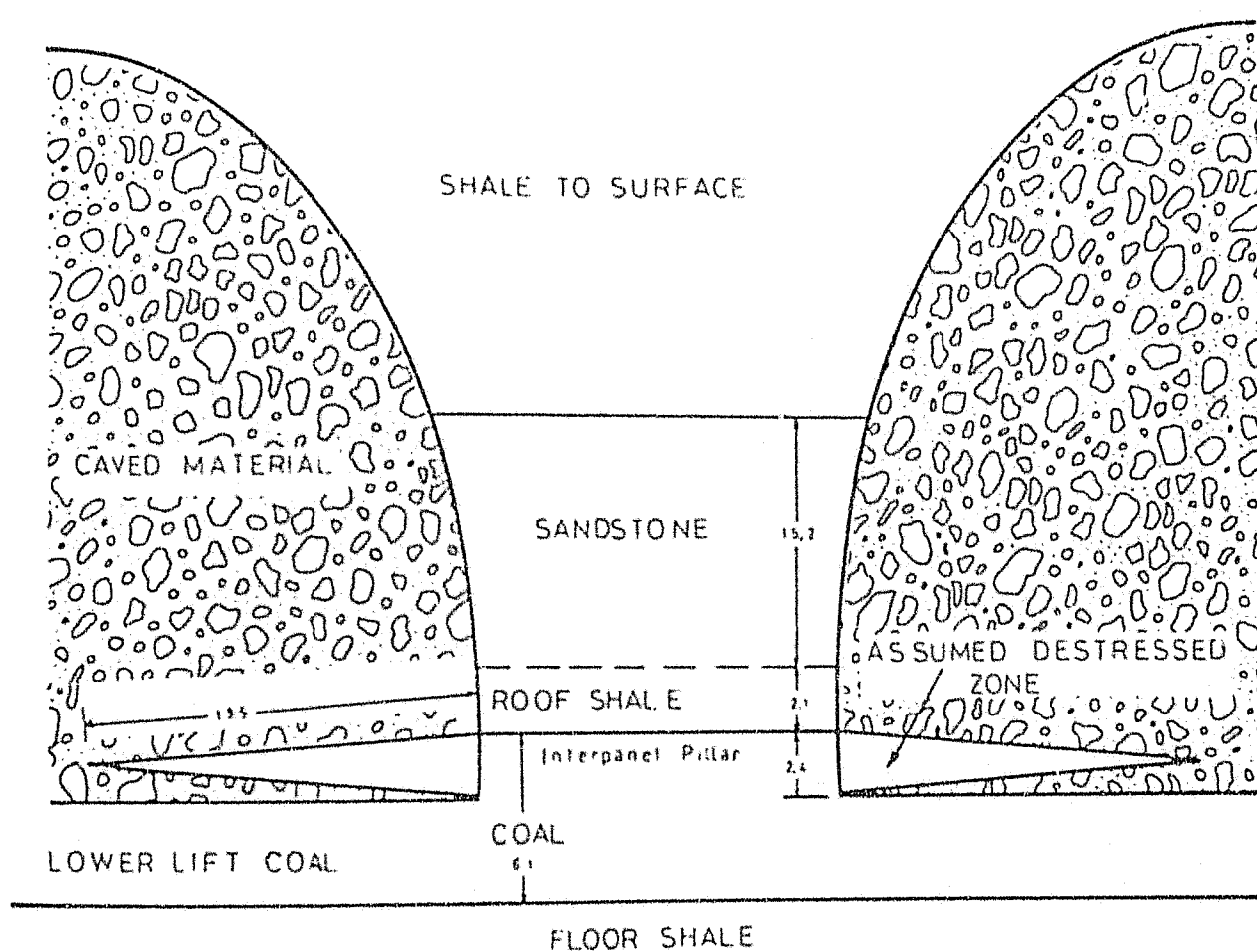


FIGURE 15 Goaf development in the top slice panels as assumed by Wilson *et al.* (1976).

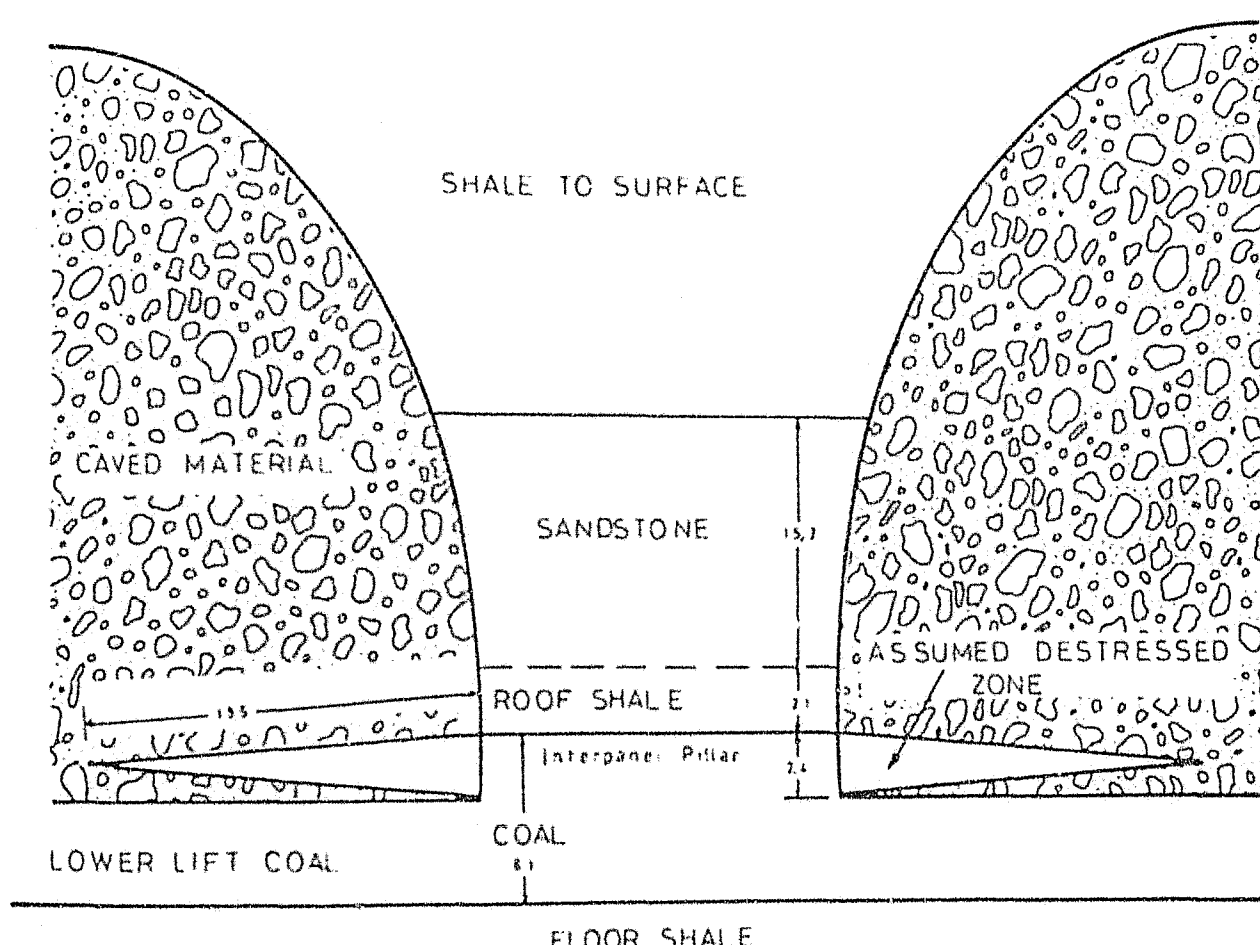


FIGURE 15 Goaf development in the top slice panels as assumed by Wilson *et al.* (1976).

whole operation. In the appendices of their report, the authors have presented informative descriptions of the various thick seam mining operations which they visited outside the United States in the course of composing the publication. Overall, the publication constitutes a good reference manual on most aspects of non-simultaneous multi-slice longwall mining.

4.3.3.2 Longwall mining with sublevel caving

Although longwall mining with sublevel caving has been practiced in many countries throughout the world, including France, Yugoslavia, Romania, the U.S.S.R, India and Great Britain, only the French operations have been discussed in detail in the English literature. One of the first papers on this method was by Callier (1972). This paper briefly discusses the development of the method in France before going on to describe the operation of the method, the face equipment, and the difficulties experienced with the method. Whilst prevailing geologic conditions have not been recorded nor discussed quantitatively, a good qualitative idea of the competence of the coal seam is obtained from the author's discussion of the support requirements of the method and the difficulties experienced with the method due to 'friable' and 'seldom hard' coal.

This mining method was discussed in much greater detail by Barron (1974) when he considered the potential of the system for use in, or adaption to, Canadian conditions. The author studied the mining method in operation at two French collieries, namely Darcy Mine and Rozelay Mine, and recorded most of the geologic, mining, environmental, and economic features of the method. These features formed the basis on which to assess the potential of the method under typical Canadian conditions.

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The applicability of longwall mining with sublevel caving to a given set of mining conditions depends, to a very large extent, on the caveability of the coal. A strong coal with very few structural discontinuities and weaknesses must often be induced to cave, and tends to cave in large blocks which are difficult to draw. On the other hand, a very weak friable coal may result in very poor face conditions, and face and roof stability ahead of the face supports could be very difficult to maintain, particularly at depth. Therefore, when considering the applicability of this method to a given set of conditions, it is important to assess the seam geology on a quantitative basis. Barron acknowledges that he fails to do this, both for the French thick seams in which he observed the method in operation, and for the Canadian thick seams in which he is considering applying the method.

When presenting the results of ground control studies conducted at the two French collieries visited, Barron attaches particular significance to face convergence. In particular the author notes that the measured face convergence is much more than that predicted from the empirical derived formula -

$$C_v = 0,2 (pH)^{0,75} D^{-0,25} , \quad (1)$$

where C_v is the convergence per metre of face advance (m/m),

p is the subsidence factor (for caving, $p = 1$),

H is the face height (m), and

D is the depth below surface (m).

This formula was based on the statistical analysis of 140 European longwall faces in thin seams ($0,8 \text{ m} \leq H \leq 3 \text{ m}$). However, the author deduced for the Darcy Mine that if H is taken as the total seam thickness, rather than as the

face height, then the measured value of face convergence and the predicted value of face convergence based on Eqn. (1) are in close correspondence. Therefore, he concludes:

that for thick seams, the total face convergence depends not on the face height but on the seam thickness between the true roof and true floor, i.e. the general statistically derived formula for thin seams, given above, is also applicable to thick seams, provided that H in both cases is taken as the seam thickness and not as the face height'

This is a very important conclusion which, if accepted, would have a significant influence on the design of thick seam mining methods. However, this conclusion is only based on a single set of measurements on one longwall face, although a second set of measurements from another longwall face are also in agreement if the seam height is reduced due to the presence of a sandstone parting within the seam. No consideration was given as to whether the application of the empirically derived thin seam formula to thick seam mining methods was, in fact, valid. Furthermore, no attempt has been made to explain this important conclusion.

Convergence on the longwall face is determined by the response of both the coal strata ahead of the longwall face and the face support system to the applied load. As such, convergence is a function of many parameters, including the elastic modulus of the coal, the nature of the immediate roof strata above the seam, the characteristics of the support system, and the behaviour of the immediate roof above the longwall face. None of these parameters are reflected in the empirically derived thin seam convergence formula, Eqn. (1), but their influence on convergence can be illustrated simply by the following elementary equalities relating convergence to load and stress -

$$\Delta \ell = \frac{L}{k_s} = \frac{L \ell}{EA} = \frac{\sigma \ell}{E}, \quad (2)$$

where $\Delta \ell$ is the convergence (analagous to face convergence),

L is the load (analagous to strata load),

ℓ is the length (analagous to face height),

k_s is the stiffness constant,

E is the elastic modulus (of seam),

A is the area, and

σ is the stress (analagous to abutment stress).

For example:

- i) Elastic modulus of coal - Eqn. (2) clearly illustrates that, for a fixed set of conditions, convergence (seam compression) is a function of elastic modulus. Thus a longwall face in a seam having a low elastic modulus will undergo a much higher convergence than a longwall face in a seam having a high elastic modulus.
- ii) Roof strata properties - The discussion of the stress distribution around an isolated longwall panel, Appendix II.2, highlighted that the caving angle, β , has a significant influence on the weight of the undermined roof strata which has to be carried by the coal strata ahead of the face. In turn, this parameter is a function of the composition and structure of the immediate roof strata. For example, a high strength, massive strata would be expected to cave at a greater caving angle than a low strength highly laminated strata. The influence of the caving angle, β , is analagous to the parameter, L , in Eqn. (2), since the caving angle determines the volume and thus the weight of overhanging strata to be supported by the face abutment.

Special consideration must also be given to very competent immediate roof strata. This strata may fail to cave at the rear of the face supports and cantilever over the supports into the goaf. Unless a well designed support system which generates a high support resistance at the rear of the support is used, this strata can induce very high face abutment stresses. This situation is represented in Eqn. (2) by the stress parameter, σ .

- iii) Support system characteristics - The stiffness and load carrying capacity of the support system determine to what extent the face is protected from high abutment stresses, and to what extent convergence increases with distance from the face. That is, the support system also influences the values of L and σ in Eqn. (2). In a later publication by Joscen and Gouilloux (1978), the empirical convergence formula has in it a term to account for the load-bearing capacity of the support system. The formula now reads -

$$C_vT = \left(\frac{6800}{PM} + 66 \right) (pH)^{0,75} D^{-0,25} \quad (3)$$

where C_vT is the convergence per metre
of face advance (mm/m),
and PM is the load bearing capacity
of the support
(tonnes/linear metre
of face).

Solving this formula for the support system employed on the longwall faces under consideration by Barron, produces the following formula -

$$C_vT = 0,1(pH)^{0,75} D^{-0,25} \quad (4)$$

The face convergence calculated with this new formula is only half of that calculated by the formula used by Barron, so that an even greater discrepancy exists between predicted and measured face convergence.

- iv) Behaviour of the top coal above the longwall face -
In conventional mechanised longwall mining very little lateral movement of the roof occurs until after it has been undermined. Thus, this roof provides lateral confinement to the coal ahead of the longwall face, thereby increasing its load carrying capacity and limiting fracturing, spalling and slabbing. However, it can be seen clearly in Fig. 16 that in longwall mining with sublevel caving, the immediate roof above the longwall face undergoes considerable migration towards the goaf long before it has been undermined. This is because the roof consists of a considerable thickness of weak and friable coal which is being caved and drawn from behind the longwall supports. Therefore, instead of confining the coal ahead of the face, the immediate coal roof strata induces lateral tension in face coal, causing it to fracture and form a number of slender slabs running parallel to the face. The formation of these slabs significantly reduces the load carrying capacity of the coal, and effectively reduces the area of coal capable of carrying full load. As can be seen again from Eqn. (2), a reduction in the area, A , results in increased convergence.

The preceding discussion illustrates that convergence is a complex function of many parameters. Few of these parameters are contained in Eqn. (1), upon which Barron bases his important conclusion concerning convergence in thick coal seams. The range of conditions for which this equation was empirically derived need to be appreciated,

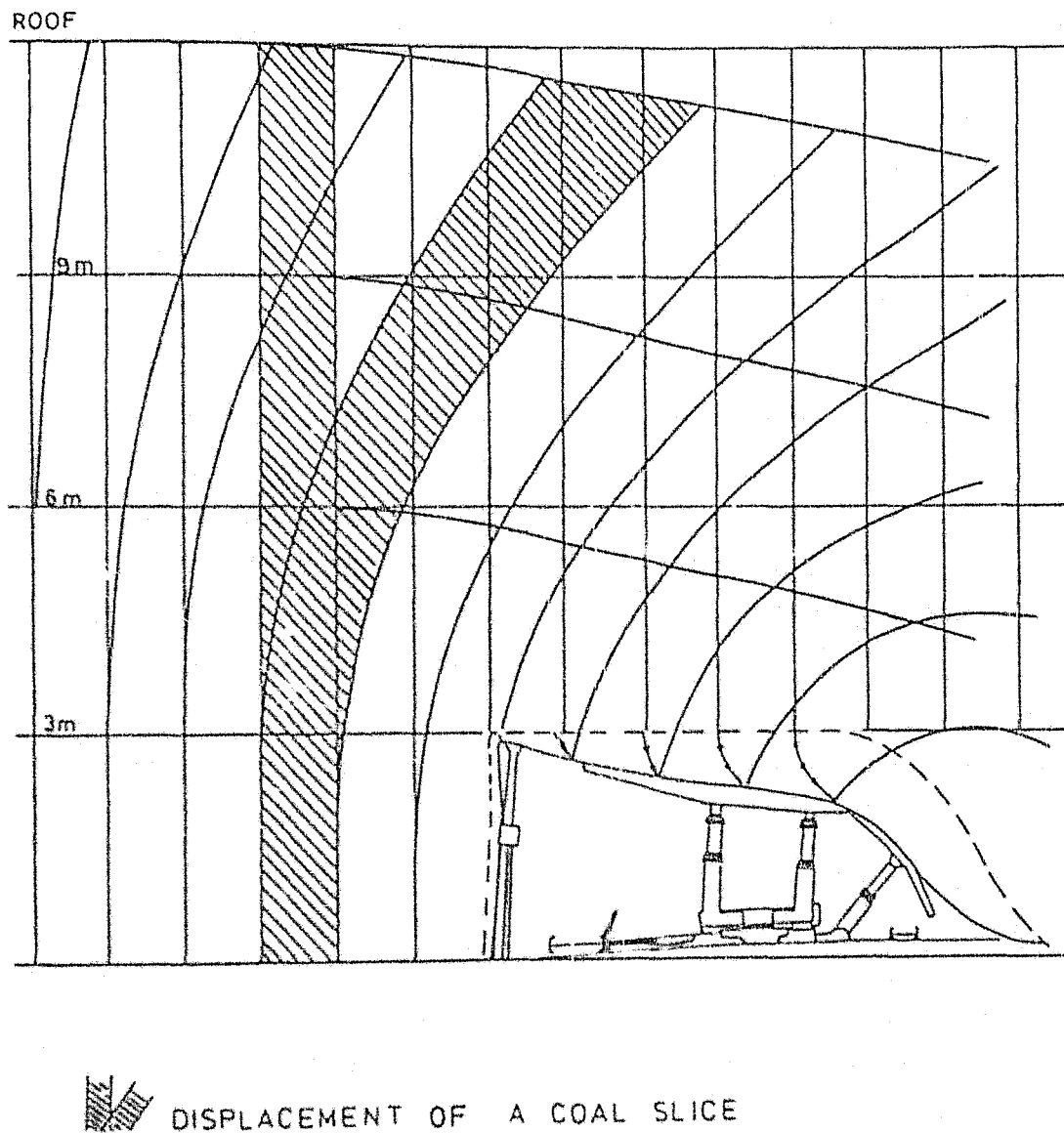


FIGURE 16 Displacement of immediate coal roof strata associated with longwall mining with sublevel caving (Barron, 1974).

and the equation should not be employed outside this range of conditions. In particular, this equation was based on strata behaviour in a conventional longwall face. It cannot be used to calculate the convergence associated with longwall mining with sublevel caving because this mining system causes the coal strata to behave in a significantly different manner to that upon which the formula is based.

In summary, Barron presents a well detailed study of longwall mining with sublevel caving as conducted in two French collieries, and briefly considers the potential of the method to the mining of Canadian thick coal seams. However, his conclusion that 'for thick seams, the total face convergence depends not on the face height, but on the seam thickness between the true roof and the true floor', is unacceptable.

Wallis et al. (1977) considered the design and feasibility of longwall mining with sublevel caving for mining western United States thick coal seams, and adopted a slightly different approach to those of Wilson et al. (1976) and Barron (1974). The authors recognised that the success of longwall mining with caving depended to a very large extent on the use of a well designed support. Therefore, the authors actually designed a longwall support system. The support was designed 'primarily for very friable coals which cave readily and need essentially no size reduction to be compatible with available haulage systems'. The support designed is notably different from other longwall caving supports in that it is over 5,5 m long and 2 m wide, and generates a load of almost 750 tonnes per metre run of face. The support systems used on the French longwall face described by Barron (1974) only generate a load of about 190 tonnes per metre run of face, whilst those used in Bogovina Mine in Yugoslavia generate a load of 260 tonnes per metre run of face (Galvin, 1978).

The authors have based their support design on the philosophy that the longwall caving system 'shall be at least as safe, the productivity as high, and the production cost per ton of clean coal as low' in seams exceeding 6,7 m (22 ft) in thickness as that presently achieved by standard bord and pillar mining or conventional longwall mining in 2,1 m (7 ft) thick seams. This philosophy necessitates the use of a very large rear conveyor, which is independent of the support system and only needs to be advanced every second face cycle.

One disturbing point in the authors' calculation of the required size of the rear conveyor and their final prediction of productivity is that they have assumed a shearer cycle time of 34 minutes over a 125 m long face. This is an extremely fast cycle time. Barron (1974) calculates that for a longwall caving installation similar to those operating in France, a shearer operating 100 per cent of the time will have a cycle time of 195 minutes over a 100 m long face. Typical cycle times on 200 m long conventional longwall faces in South Africa are in the order of 105 minutes. Correspondingly, the authors predict a face productivity of 181 to 229 tonnes per manshift, as compared to about 17 tonnes per manshift achieved from installations in France (Barron, 1974) and 11 to 35 tonnes per manshift achieved in Yugoslavia (Lang, 1980, personal communication). Even conventional longwall installations in South Africa only achieve about 65 tonnes per manshift (Esterhuysen, 1981). Thus, the authors' prediction of face productivity appears unrealistic.

After presenting their longwall support design and discussing the selections of the other face equipment, Wallis *et al.* (1977) then go on to consider the following points in the order listed:

- i) ground control,
- ii) health and safety requirements and hazards evaluation,
- iii) panel design and ventilation,
- iv) analysis of operating cycles,
- v) economic analysis,
- vi) application of longwall mining with caving.

Overall, the content matter of this paper is not presented in a logical sequence and one has to frequently refer forward in order to learn the reasoning behind an assumption.

In discussing face convergence the authors have presented a sketch of the stress distribution around a longwall panel, Fig. 17. This figure incorrectly shows a uniform stress distribution along the longwall face and along the interpanel pillars. These stress profiles are shown to meet at the corners of the longwall face and interact to produce a stress peak, rather than a stress low as discussed in Appendix II.2 and illustrated in Fig. II.4. The incorrect stress distribution probably originated from a publication by Whittaker (1974) which has been referenced in a number of publications (refer e.g. Peng (1978), p215). The profile has since been corrected in a publication co-authored by Whittaker (refer Whittaker and Singh, 1979).

Wallis et al. also makes use of the previously discussed empirical formula presented by Barron (1974) for calculating face convergence, Eqn (1). However, the authors recognise that this formula has many shortcomings and only use it 'in the absence of sufficient time and information in which to perform (their own) rigorous analysis of face convergence'.

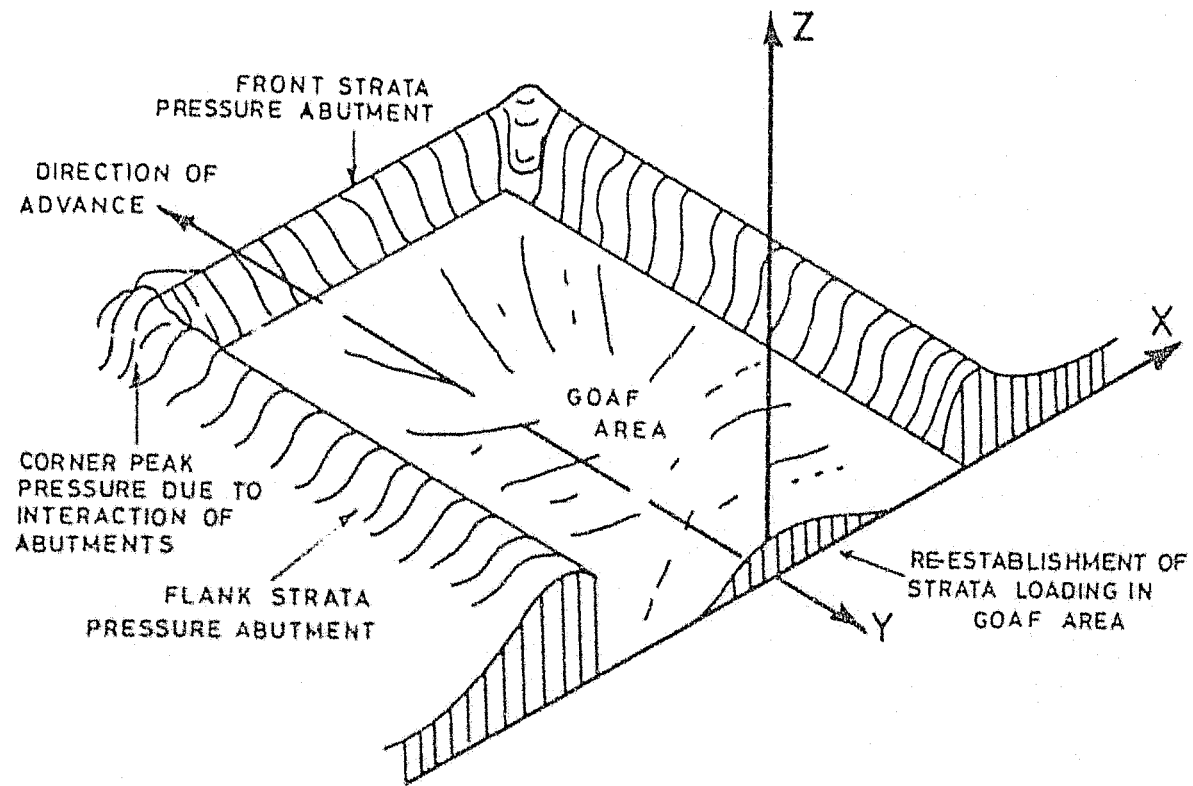


FIGURE 17 Stress distribution profile around an isolated longwall panel, as presented by Wallis *et al.* (1977).

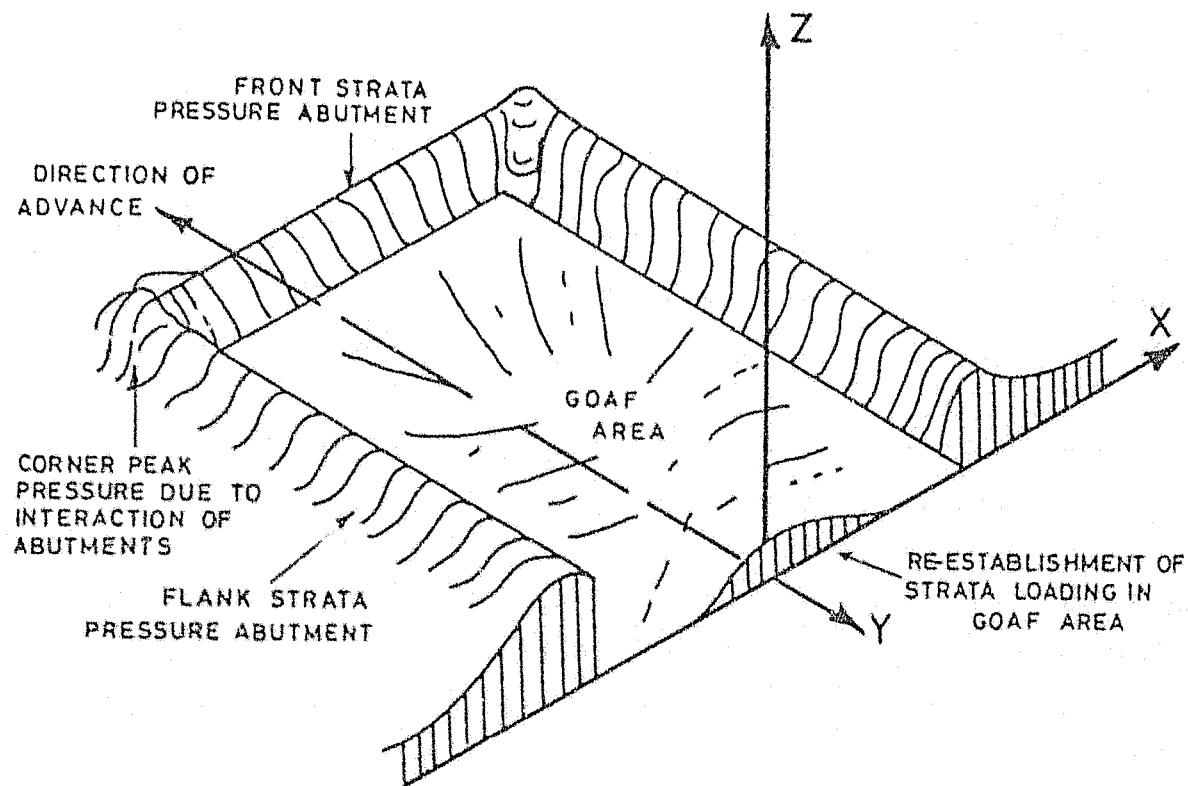


FIGURE 17 Stress distribution profile around an isolated longwall panel, as presented by Wallis *et al.* (1977).

4.3.3.3 Hydraulic mining

Hydraulic mining has many advantages over most other thick seam mining methods and these have been well documented in literature by Grimley (1974), Parkes and Grimley (1975), Parkes and Grimley (1976) and Shaw (1977). In addition, Grimley (1974) and Parkes and Grimley (1975) have also noted the physical requirements necessary for the successful application of the method for thick seam mining.

Most publications on hydraulic mining are concerned with the application of the method at Michel Colliery in Canada, and English literature concerning other hydraulic mining operations is scarce. Merret (1978) has described briefly a hydraulic mining operation in West Germany whilst Mills (1978) reviewed the state of hydraulic mining in the U.S.S.R. and listed the prerequisites for the successful operation of the method. The author noted that the strength, friability and cleavage of the coal influence the success of the method to a large extent. However, no publications are known of which record the approach adopted to quantitatively assessing the potential of hydraulic mining under a given set of conditions.

4.4 Conclusions

The literature review highlights that the problem of selecting the optimum thick seam mining method for use under a specific set of conditions has not been approached systematically in the past. Occasionally, the potential of one particular mining method has been evaluated in the light of specific site conditions but this potential has not been compared to that of other methods under the same site conditions. Therefore, this thesis represents a first known attempt at systematically evaluating the potential of established thick seam mining methods for use under specific

sets of conditions, although these conditions tend to be 'region' specific rather than 'site' specific.

Much of the literature reviewed only contains descriptions of thick seam mining operations and gives no consideration to the selection of the mining methods nor to the specific conditions under which the methods operate successfully. Furthermore, a number of publications contain proposals and conclusions which are unacceptable. In instances, these proposal and conclusions reflect a lack of understanding of not only the thick seam mining method (or methods) under discussion, but also fundamental mining principles. Overall, publications contain only limited quantitative information relating to the characteristics and requirements of established thick seam mining methods. Thus, for the purpose of evaluating the potential of these methods for use under local conditions, considerable reliance has to be placed on qualitative information contained in publications.

CHAPTER 5

GEOLOGIC AND ECONOMIC CHARACTERISTICS AND REQUIREMENTS OF ESTABLISHED THICK SEAM MINING METHODS

5.1 Introduction

An initial step in selecting a mining method for use at a specific location is to compare geologic conditions associated with established mining methods with geologic conditions existing at that location. In this manner mining methods technically feasible under these non-variable conditions can be identified. The final selection of a mining method is then based on a consideration of prevailing practical and economic constraints. As these constraints change with time consideration can be given to other technically feasible mining methods.

Such an approach has been adopted in this thesis to identify established thick seam mining methods which, firstly, are technically suitable to the mining of South African thick coal seams, and secondly, which may be introduced at the present moment in time. Therefore, in this chapter established thick seam mining methods are identified, classified and described, and the geologic and economic characteristics and requirements of these methods are tabulated. Unless stated otherwise, all further references in this thesis to thick seam mining methods imply 'established' thick seam mining methods.

5.2 The Classification of Thick Seam Mining Methods

In order to record in a logical sequence the large number of thick seam mining methods identified in the course of conducting the literature review a system for classifying these methods is required. One such system is that briefly

discussed in Chapter 4, namely, to classify methods as either full face, slicing, or caving and drawing mining systems, depending on the manner in which the total mining height is extracted. However, this single criterion still results in too broad a classification of the many thick seam mining methods identified. Therefore, a second criterion, based on the effects of a mining method on the roof strata, has been devised. Methods may:

- i) 'maintain' the integrity of the roof strata;
- ii) result in 'limited subsidence' of the roof strata;
- iii) 'cave' the roof strata.

Roof strata control is a convenient criterion to select since this criterion, alone, often determines whether a mining method is suited to a particular set of conditions. This is particularly so if a coal seam is very thick, and/or underlies important surface structures, other mineable seams or very competent strata.

By combining the three types of mining systems and the three types of roof strata control in the form of a matrix, nine classes of thick seam mining methods can be identified, Table 3, although no thick seam mining methods fall within two of these nine classes. It is interesting to note that all methods which maintain the roof strata are based on some form of bord and pillar mining, whilst all methods which control roof strata displacement are based on some form of longwall mining with stowing.

5.3 Description of Thick Seam Mining Methods

Reference to Table 3 shows that although a considerable number of thick seam mining methods are employed throughout the world, many of these methods are variants of conventional coal mining methods. In fact, all methods recorded in the Table are based on one of the following six basic mining methods:

TABLE 2 Classification of thick seam mining methods according to the type of mining system and roof strata control.

MINING SYSTEM			
	FULL FACE	SLICING	CAVING AND DRAWING
ROOF STRATA CONTROL	MAINTAIN	• Bord and pillar: - in a number of slices - with top- or bottom-coaling - with top- or bottom-coaling followed by stowing - with repeated cycles of stowing and top-coaling	
	LIMITED SUBSIDENCE	• Non-simultaneous multi-slice longwall mining in: - descending slices with stowing - ascending slices with stowing • Simultaneous multi-slice longwall mining in: - ascending slices with stowing	
	CAVE	• Bord and pillar: - multi-slice with pillar extraction - with pillar extraction and top- or bottom-coaling • Non-simultaneous multi-slice longwall mining in: - descending slices • Simultaneous multi-slice longwall mining in: - descending slices • Non-integrated longwall mining with sublevel caving	• Integrated longwall mining with sublevel caving • Hydraulic mining • Stope mining: - open stoping - sublevel caving and drawing

- i) Bord and pillar mining.
- ii) Conventional longwall mining.
- iii) Multi-slice longwall mining.
- iv) Longwall mining with sublevel caving.
- v) Hydraulic mining.
- vi) Stope mining.

These six basic mining methods provide a convenient and concise basis on which to describe, in general terms, thick seam mining methods. A brief description of each specific thick seam mining method is contained in Appendix III. The more relevant publications describing these methods are listed in Appendix IV.

5.3.1 Bord and pillar mining

Primary development consists of driving tunnels through the coal seam in such a manner that the seam is divided into blocks, or pillars. These pillars are usually square or rectangular in shape and typically range in size from 5 x 5 m to 30 x 30 m, depending on depth. Secondary mining operations consist of either top- or bottom-coaling with or without stowing, or pillar extraction. In multi-slice operations it is usual to leave a 2 to 5 m coal parting between slices and to superimpose both panel and interpanel pillars.

5.3.2 Conventional longwall mining

Reference to Table 3 reveals that only two thick seam mining methods are based on conventional longwall mining, namely, full face longwall mining and full face longwall mining with stowing. Panel development and secondary extraction operations are similar to those associated with conventional longwall mining operations which extract 2 to 4 m thick coal seams, although face length may be limited in order to maintain a rapid rate of

advance thereby reducing strata control problems. However, face support systems may differ significantly, with the use of shield supports to maintain lateral support stability, face sprags to limit slabbing of the face, and elevated travelling ways for inspection and maintenance purposes, Plates 1 and 2. In addition, the shearer must usually traverse the face twice to extract the full mining height.

5.3.3 Multi-slice longwall mining

Multi-slice longwall mining, also known as multi-lift, multi-pass, and top-slice longwall mining, extracts a thick coal seam in a number of slices by longwall mining, Fig. 4. If slices are extracted concurrently, the operation is referred to as simultaneous multi-slice longwall mining, Fig. 4(a). If a considerable time interval, in the order of several months or years, exists between the mining of each slice, the operation is referred to as non-simultaneous multi-slice longwall mining, Fig. 4(b). Stowing can be incorporated into both these operations.

5.3.3.1 Simultaneous multi-slice longwall mining

When stowing is incorporated into this method, slices are extracted in ascending order, whilst with no stowing, slices are extracted in descending order. On rare occasions slices have been extracted in descending order and stowed. The mining height employed in each slice is typically 2 to 3 m. Face length is usually constant for each slice and of the order of 80 to 100 m. A distance of 30 to 50 m is usually maintained between faces to minimise strata control problems. When slices are extracted in a descending order, the roof of lower slices consists of either:

- i) an artificial roof, such as wire mesh, which may be placed on the floor during upper slice mining operations or against the roof during lower slice operations;
- ii) a stone parting;
- iii) a 0,5 to 1,5 m coal band.

Significant economies are often achieved in entry development costs by utilizing panel entries for more than one longwall panel. In particular, where single entries are employed in two slice operations, each entry can serve as a maingate for upper and lower slice mining in one panel, and as a tailgate for upper and lower slice mining in the adjacent panel.

5.3.3.2 Non-simultaneous multi-slice longwall mining

With this method, slices have been extracted in descending or ascending order with or without stowing. However, it is most common to extract slices in a descending order and, in fact, mining slices in an ascending order without stowing has fallen into disuse. The need to locate panel entries for each successive slice in distressed ground necessitates placing these entries within the confines of preceding longwall panels (section 4.3.2). This results in a reduction of face length and a corresponding decrease in percentage extraction for each slice mined. Mining height is typically 2 to 3 m in each slice. A time interval of 6 to 24 months usually exists between the mining of successive descending slices to allow sufficient time for consolidation of the goaf. In these instances, the roof of lower slices may take one of the three forms noted when discussing simultaneous multi-slice longwall mining. However, with the advent of self-advancing longwall supports, which provide good roof coverage and support close to the face, it has become increasingly popular to mine directly beneath consolidated goaf.

5.3.4 Longwall mining with sublevel caving

The lower 2 to 2,5 m of a thick coal seam is extracted by longwall mining. However, as the face is advanced, the remaining top-coal in the seam caves into the excavation at the rear of the supports from where it is recovered, Fig. 5(a). This is achieved by using specially designed self-advancing powered supports which allow two conveyors to be operated along the longwall face. A conventional front conveyor removes coal mined from the longwall face, whilst a second conveyor located at the rear of the support recovers caved top-coal from the goaf. The most common type of support system used for this operation is a chock support incorporating a hydraulically activated shield plate at the rear of the support. This shield protects the conveyor from the goaf and assists in controlling the drawing of the coal. Face lengths are typically 80 to 100 m and panel lengths 600 to 800 m. Advances in longwall support technology have almost eliminated the need for non-integrated longwall mining with sublevel caving whereby an upper coal slice was extracted, usually by longwall mining, to destress the coal seam prior to commencing longwall caving and drawing operations in the base of the seam. Seams of up to 10 m in thickness are now mined in a single operation.

5.3.5 Hydraulic mining

Typically, main developments are located in the base of the seam and secondary panel developments are driven at a minimum of 5 degrees to the rise from the main developments. Secondary development may divide a panel into a number of long rib-pillars or, if cross-cuts are driven, into a bord and pillar layout.

Secondary extraction is achieved with high pressure water monitors usually mounted on feeder breakers, Fig. 5(c).

Author Galvin J M

Name of thesis The mining of South African thick coal seams - rock mechanics and mining considerations 1981

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