BEHAVIOUR AND STABILITY OF DEEP COAL MINING TUNNELS

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ABSTRACT

This article gives an account of the parameters involved in predicting and analysing the support requirement and estimating the tunnel closure in main access tunnels in coal mining. A detail of in situ investigation of monitoring the extent of yield zone (zone of broken rock around the tunnel) and deformation survey of three main tunnels in Coal Measure Strata together with a comprehensive laboratory studies of mechanical properties of rocks associated with above site are presented. The influence of rock strength properties and their stratified nature upon the development of zone of broken rock is examined. The field data of the three case studies of tunnels driven in Carbonferrous rocks are analysed, presented and discussed in relation to overall stability as well as predictive design formulae.

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1. INTRODUCTION

The stability of mining tunnels in sedimentary Coal Measure Strata play an important role in the success of any major underground Coal Mining projects. The initial deformation which may lead to considerable closure and damage to the *unnel is caused by redistribution of high stresses around the excavation as well as attainment of equilibrium by the disturbed rock mass. The closure and damage of the tunnel reduces considerably the efficiency of the ventilation, speed and reliability of transportation system of men and material as well as the removal of the coal. The tunnels which have experienced high deformation require repair, entailing high direct and indirect cost.

Considerable studies are currently being conducted to investigate the design of such mining tunnels and special attention is being given to the stresses and deformation as well as mechanical properties and behaviour of surrounding rock mass. The tunnelling drivage technique and support system employed influence markedly the achieving of an effective strata control within such mine roadways. The full face tunnelling machines with cutting heads to profile the tunnel walls contribute not only to a favourable rock stress distribution but also result in a more effective and quicker installation of the support system. However tunnelling by means of drill and blast inhibits the immediate disturbance of rock mass surrounding the tunnel and contribute significantly to the formation of a yield zone (broken rock zone around the tunnel profile), thus promoting instability.

The shape and size of the roadway play a major role in achieving a long-term stability. Whilst most coal mining tunnels are arch shaped, there is new trend towards having the major access tunnels in circular shape. Virtually most of the coal mining tunnels are lined or supported with steel girders, predominantly as arched profiles although the future trend is more towards circular type (rings). Such support system is that of the 'H', rolled steel joint section constructed to fit the arched as well as circular shaped tunnel. The steel arch supports are made of 2-3 pieces and the steel ring support generally constructed from 4-5 piecer. Each steel arch or ring is spaced by means of appropriate number of tie members (strut). This type of support system is covered with steel sheeting or steel mash to prevent small roof falls. They are a passive support and activated by natural ground deformation leading to strata loading, caused by rock stress redistribution.

Some of the most important recent mining tunnels of drifts which require to possess a more permanent and greater stability have a circular profile together with concrete lining as the main support system. Due to high stresses at depth, the concrete lining is not necessarily the best form of support system. There has been a number of cases where concrete failure has been encountered causing considerable costly repair work to the lining. In the new mining projects such as the Selby in U.K. and Dunken Morrian (Nova Scotia) in Canada, the main access roadways are inclined circular tunnels (drifts) rather than shafts and are mainly supported by steel girder supports. This is mainly due to the decreased cost, speed of installation as well as their ability to accept considerable amount of structural deformation without appreciable loss of support strength.

The selection of support required or support resistance together with estimation of tunnel closure has led to development of number of design criteria which are examined and compared by the authors using data from monitoring three field sites and laboratory studies of physical and mechanical properties of rocks.

2. TUNNEL DESIGN CRITERIA

The theoretical elastic solution for a circular opening in anisotropic, homogeneous and linearly elastic rock is well defined in various rock mechanics text books Jeager and Cook(1), and Obert and Duvall(2). The predicted deformation using such elastic analysis is far too insignificant in comparison with the in situ measured data from tunnels in Coal Measure Strata. This is mainly due to the development of zone of broken rock material (yield zone) surrounding the excavation. This concept was originally examined by Fenner(3). It was shown that by redistribution of stresses around the circular opening, the peak tangential stresses is to be displaced from the tunnel sides to an interface separating the yield zone and elastic zones. A straight line (linear) Mohr-Coulomb relationship was used by Fenner as the rock failure criteria. This original work formed the basis for estimating support pressure due to yielding of rocks in immediate surrounding of the tunnel excavation. The Fenner's work has been extended by many other investigators and a number of theoretical and empirical solutions for obtaining a more accurate prediction of the state of stress and strain around circular opening in rocks have been proposed. Bray(4), Ladanyi(5) and Wilson(6) have developed theoretical design criteria to aid in the design of support requirement and anticipated tunnel closure. A detailed discussion of these design principles is beyond the scope of this paper, however, a brief discussion is presented in order to highlight the parameters investigated by the author.

There are strong similarities linking the different solutions suggested by the investigators. There are some common assumptions which apply generally to these analyses and are listed below.

a. Virgin stress field is hydrostatic.
b. Plane strain conditions exist and the distance from the tunnel face has no significance
c. The rock mass surrounding the tunnel excavation is assumed to be homogeneous and isotropic.
d. The drivage has a circular cross-section.

Considering the above assumption together with theoretical stress conditions and rock expansion in the yield zone there are number of parameters with the predictive design formulae; full details of their derivations are given in the original papers listed in the references. These parameters together with their associated symbols are listed below.

p' = residence of yielded rock
e = expansion factor
C_0 = cohesion of intact rock
C(j = diametric closure
d = driven diameter (r = driven radius)

v = Poissons ratio
E = Modulus of Elasticity
k = triaxial stress factor
q = cover load
o = laboratory-determined, unconfined compressive strength
p = lining resistance
f = a factor relating laboratory strength of rock
e = average plastic dilation
0 — angle of internal friction

The Bray(4) predective formula was specifically proposed for finely jointed rock mass to introduce the problem of stress change at joint surfaces where slip failure could occure. The intact rock has been considered as competent, but the cohesion of the broken rock material is considered zero.

Bray Formula:

$$\frac{C_d}{2} = \frac{1-\nu}{E} \{(p-q)\,r\} + \frac{1-\nu\,.\,R^2}{r\,.\,E} [q - p\,\{\frac{2q-\sigma+(k+1)\,.\,C_o}{(k+1)\,(p+C_o)}\}]$$

$$+\frac{1+\nu}{E}\{\frac{(k+1)\,q+\sigma}{K+1}\}R^2$$

where

$$R = [\frac{2q-\sigma+(k+1)}{(k-1)\;(p+C_o)}\,.\,C_o]^{\frac{1}{k-1}}$$

Ladanyi(S) in the development of a design criteria considered a long term strength concept of the rock surrounding the tunnel excavation. The post-failure reduction of strength and associated volumetric dilution are examined in depth. It is a well defined formula which is applicable to high stress rock environment and various rock types including creep-sensitive rock material.

Lavandi Formula:

$$C_d = r\,(1-((1-e)/(1+A))^{1/2})$$

where $A = \{2.\frac{1+\nu}{E}[\frac{\sigma+(k-1)\,q}{K+1}] - e\}\{q+p' - [\frac{\sigma+(k-1)\,q}{(k+1)\,(p+p')}]\}^{\frac{2}{k-1}}$

Wilson's(6) formula was originally derived on the theorotical considerations which have been modified to consider well defined empirical observations from British Coal Mining environment. It is supposed to be applicable to deep tunnels in various range of rock strength. It is very popular with British coal mining tunnel designers.

Wilson Formula:

$$C_d = d.\frac{1+\nu}{E}\left\{\frac{(k-1)\,q+\sigma/f}{(k+1)}\right\}\left\{\frac{2q-\sigma/f}{(p+p')\,(k+1)}\right\}^{\frac{2+\varepsilon}{k-1}}$$

The above formula together with the in situ data from three tunnelling sites (which will be discribed later) and used to back analyse and examine the sensitivity and predictive accuracy of these design equations.

3. DESIGN AND INSTALLATION OF BOREHOLE EXTENSOMETER

Borehole extensometers have been extensively used to measure strata deformation associated with underground excavations in varying geological conditions. Although the basic design and use of extensometers is well established, the final design may be modified to meet the particular requirement of individual investigations.

In this investigation it was required to measure the development of rock yield zones surrounding various mine tunnels in relation to tunnel advance and time. The following design conditions were considered in adipting this particular type of extensometer:

1. **to be installed with relative ease and short duration of time to avoid delaying tunnel d rivage operations;**
2. **to be installed as close to the tunnel face as possible (1 - 2 m);**
3. **to incorporate multi-anchor points, in order to allow the evaluation of relative rock movement within the extent of the yield zone;**
4. **to have sufficient length to allow a firm anchorage beyond the extent of the yield zone; and**
5. **to be intrinsically safe for use in British underground coal mines.**

The multi-wire extensometer used in this investigation is shown diagramatically in Figure 1. The extensometer consists of 10 steel anchor points each separated at 1 m intervals by flexible P.V.C. tubing. Attached to each anchor point is a 1 mm diameter stainless steel strain wire which is passed through the anchor bore and contained by the P.V.C. tube. The tube is fastened securely to each anchor point by Jubilee adjustable hose clips. Attached along the whole length of this assembly is a 'breather' tube which is incorporated for use in the grouting operation. The extensometer system was designed in this instance to be installed and fully grouted in a 43 mm diameter borehole of 10 m overall length.

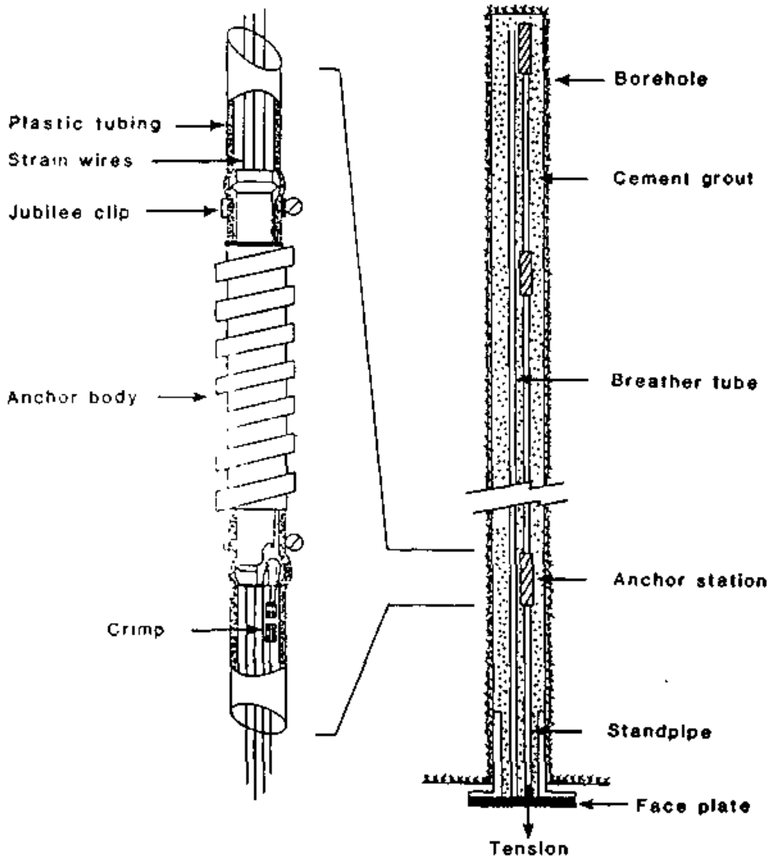

Figure 1— Borehole extensometer anchoring system.

On completion of drilling the borehole, the extensometer is immediately inserted to prevent the loss of hole due to collapse, especially in weak mudstones. The inclination and position of the borehole is then logged. A standpipe is then passed over the end of the extensometer and securely fixed into the mouth of the borehole using a fast setting grout. A face plate is then positioned around the extensometer and secured to the standpipe. This results in the borehole being sealed to prevent grout leakage. Grout is then pumped into the borehole via a grout inlet which is incorporated in the face plate. The operation is continued until borehole is completely filled. It is then sealed off and the borehole left for approximately 24 hours to allow the grout to set. A typical installation is achieved within a 1/2 hour time limit. The grout used in the installation is a cement/water mixture by the ratios of 3:1 by weight. Cores taken from underground casting have shown that the strength properties of the grout possess a compressive strength of 10 MN/m^2 and a tensile strength of 2.2 MN/m^2. This strength is below that normally encountered in the rock mass and consequently does not have any significant strengthening effect. After 24 hours the face plate is removed and the extensometer head unit is cleaned up and made ready for the measurement of readings. Readings are obtained by attaching a measuring head unit to the standpipe. Each wire is successively strained by passing the wire over the pulley wheel and attaching a 5 kg dead weight. All measurements are taken relative to the face of the standpipe. The movement of each anchor point is gained by measuring the relative displacement of a copper crimp which is attached to the strain wire. The displacement of the copper crimp is measured on a calibrated steel tape which is fixed to the measuring head unit.

4. IN SITU AND LABORATORY INVESTIGATIONS

Site investigations were conducted in three major mining tunnels driven in the Coal Measures rocks. At each of these sites a detailed deformation survey was undertaken, monitoring rock movements with the extensometer patterns shown in Figure 2. It was not always possible to incorporate a floor extensometer. In these cases a floor plate was used to monitor the amount of overall floor lift. A detailed section of the geology was taken at each site and the driven horizon marked. Details of drivage technique and support method were recorded for the evaluation of roadway stability.

The mechanical characteristics of the rocks encountered in the underground excavation contributes substantially to their structural stability. An extensive research programme into the pre- and post-failure characteristics of the Coal Measures rocks has been conducted by Hassani(7), (8) and (9), this is to evaluate the overall characteristics of the compressive, tensile and trixial strengths and volumetric expansion of such rock types associated with underground roadway stability assessment. The rocks from these three underground sites were incorporated within this laboratory testing programme. An electro-hydraulic servo-controlled testing system was used to perform the rock strength tests. Some of the mechanical properties of the rocks (mainly uniaxial compressive strength, UCS, and triaxial stress factor, $\sigma_1/\sigma_3 = k$) associated with the three underground sites are given in Figure 2. These results have been used in the prediction of the tunnel deformation at these three sites.

5. ANALYSIS OF THE DATA FROM IN SITU INVESTIGATION

The data obtained from the underground installations were analysed to determine the rock displacements associated with yield zone development. These data were obtained with reference to the measurements of tunnel wall and extensometer anchor point movements. Figure 3 shows results of movement of the tunnel wall and individual anchor points with respect to time. Figure 4 shows the actual displacement of the strata layers along that particular extensometer length. This analysis was performed on all 16 extensometer stations associated with the three underground sites. These results enable an interpretation to be made of the extent of the yield zone with respect to each individual site, as shown in Figure 2 and Table 1. Such an analysis indicates that displacements within the rock mass surrounding tunnel drivages can be successfully monitored and allow an interpretation of the parameters effecting yield zone development and tunnel closure to be assessed.

It is clearly shown that the shape and extent of a yield zone is dependent upon the lithological nature of the rock mass and their associated mechanical properties, as detail-

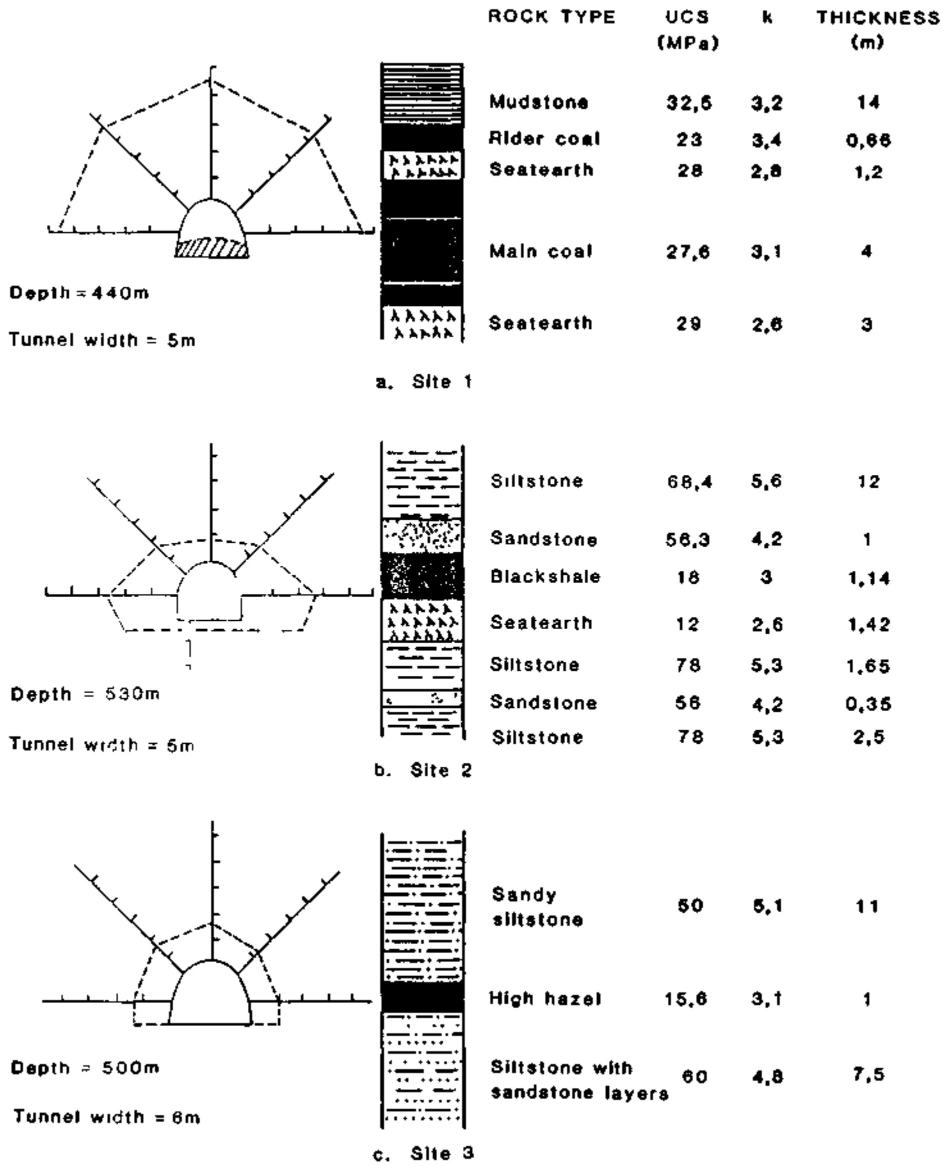

Figure 2— SITE installation geotechnical data and borehole configuration with yield zone.

ed in Figure 2a, b, and c. Figure 2b shows the shape of the yield zone at Site 2 where a tunnel has been driven in a weak rock formation with higher strengths in the tunnel roof and floor. This produced a maximum yield extent of 2.5 m in the side walls with the minimum in the roof and floor. A similar feature was also observed to be the case at Site 3. However, where weak stratified layers of sedimentary rocks are encountered, as the case of Site 1, the overall yield zone was observed to demonstrate uniformity in an ap-

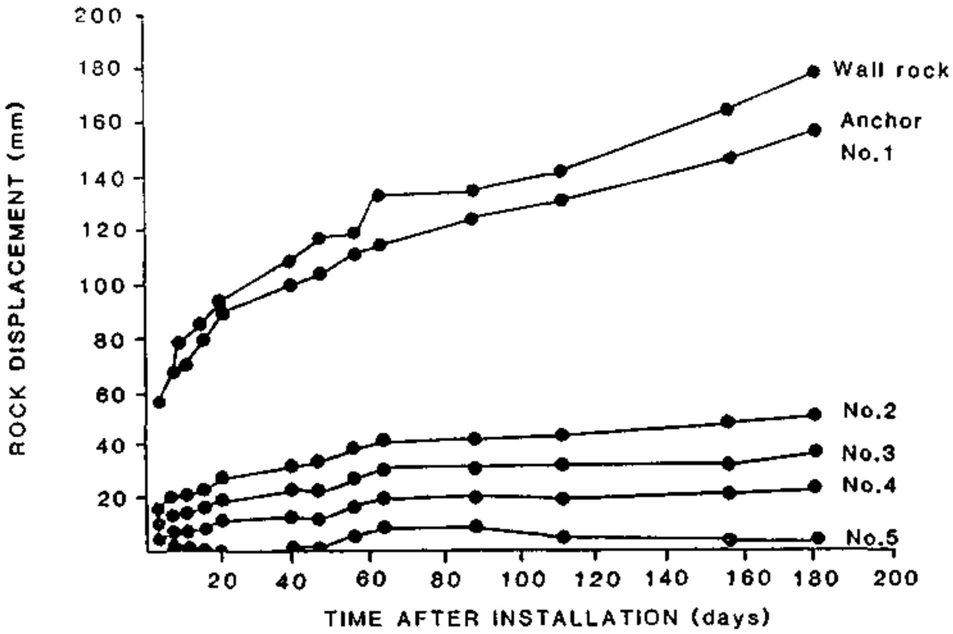

Figure 3— Rock displacement in relation to anchor position.

Table 1- Measured values of yield zone and tunnel closure

Measured Parameters	Site No.	Extensometer Station A	B	C	D	E	F	Average Value
Yield zone*(m)	1	5.00	4.38	4.41	4.40	4.88	—	4.61
	2	2.8	0.5	0.3	1.4	2.5	0.3	1.30
	3	0.66	1.55	1.50	1.83	0.59	-	1.226
Tunnel Radial Closure (C) (mm)	1	66.0	97J0	179.0	94 0	90.0	—	89.0
	2	67.0	7.0	9J0	14.0	60.0	40.0	32.8
	3	11.0	58.0	66.0	66.0	6.0	-	41.4

•* Measured from tunnel rock excavated surface (L)

proximately circular form in the upper supported region. The measured extent of the yield zone formation at Site 1 was 4.4 to 5 m.

At each site a yield zone was observed to develop after a few metres of roadway advance. In the case of Site 1, results indicate that initial movements of the roof were due

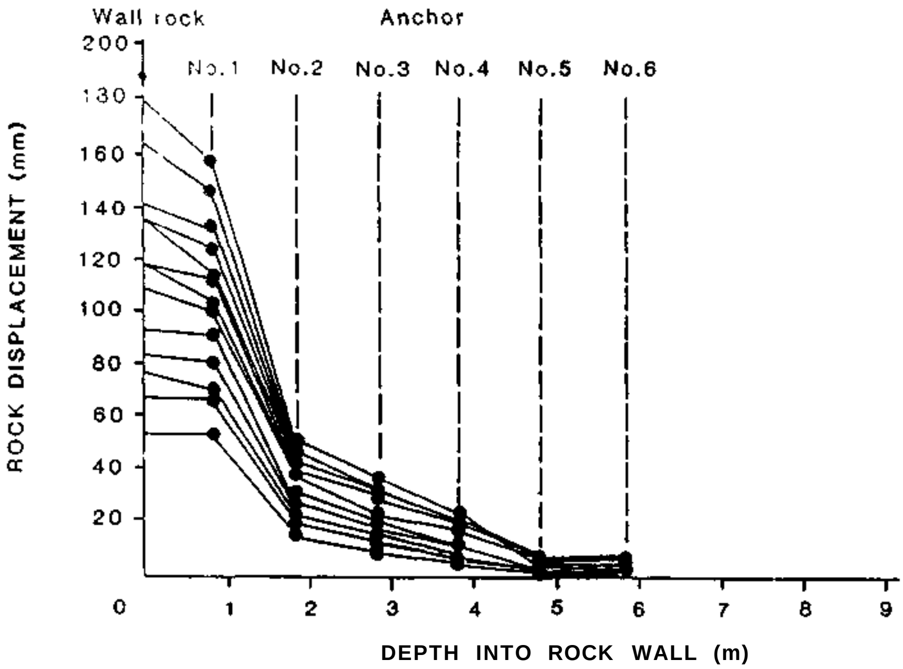

Figure 4— Rock displacement in relation to anchor position.

to bed separation. Figure 3 shows the large differential movement between anchor points 1 and 2 which is clearly indicative of bed separation. In this particular case it was noticed that the arch support was not in contact with the rock wall, and therefore, no support resistance was offered at the initial stage of strata displacement. From the analysis of Site 2 and 3, it was concluded that the yield zones were fully formed after a period of three days, equivalent to tunnel advances of 9 m. The results from Site 1 shows that where a large number of weak sedimentary rocks are encountered, time dependent failure aspects start becoming more predominant. Therefore, the yield zone continues to extend with time and tunnel advance.

6. TUNNEL STABILITY ANALYSIS

The measured tunnel deformations were compared to the predicted values of roadway closure as obtained from a number of design formulae, (mentioned earlier) namely Wilson(6), Ladanyi(5) and Bray(4). The predicted tunnel closures for the three underground sites are presented in comparison to the measured values in Tables 1 and 2. The measured and predicted values of yield zone extent are also shown. As can be seen from these results, the degree of correlation is variable. This is to be expected as the assumptions upon which the formulae are based are not necessarily the conditions experienced in the underground installations. By modifying the input parameters it is possible to obtain a better agreement with the measured values. It is suggested that the use of a rock classification technique which encompasses the Carboniferous Coal Measures weak rock conditions is required and further development in this area is of great importance.

The predicted formulae calculates the anticipated closure of a circular tunnel in a homogenous rock mass. However, it has been necessary to base initial stability assessments on ground supported by steel arched systems. From these results the use and validity of methods which predict the amount of closure to be expected in a tunnel with a circular cross-section driven in the Coal Measure Strata can be assessed.

Table *1*— Predicted values of yield zone and tunnel closure

Predicted Parameters	Site No.	Methods of Prediction Wilson [3]	Layandi [4]	Bray [5]
Yield Zone* (m)	1	3.7	0.2	0.6
	2	5.5	0.3	0.7
	3	6.0	0.5	1.0
Tunnel Radial Closure (C) (mm)	1	36.0	10.0	11.0
	2	76.0	13.0	7X)
	3	78.0	17J0	10JO

* Referred to distances from tunnel rock excavated surface (L)

7. CONCLUSIONS

The general conclusions drawn from these studies are listed:

i. The extensometers incorporated in this monitoring scheme have proved successful in their application; the speed and ease of installation of pre-assembled extensometers proved of considerable benefit where time was short as at an operational tunnel face.

ii. The extent of the rock yield zone at all three sites was measured using the extensometer system. The relationship to lithology and rock strength parameters has been demonstrated to be linked with the extent of the yield zone around the tunnel.

iii. In competent rock formations the yield zone was developed after a relatively short period of time and tunnel advance, 3 days and 9 m respectively. However, in the weaker rock formations the complete developmentof the yield zone was found to be time related.

iv. The predictive techniques used gave variable degrees of correlation with actual displacement values. It is recommended that the application of a rock classification system would be of considerable help in achieving closer correlation between observed and predicted tunnel closure values. Representation of tunnel rock strength values and properties in a suitable simplified form as is generally required for use in tunnel stability prediction formulae is still proving to be of major concern.

v. It is very important to note that most of the parameters investigated and discussed in this paper which are essential to tunnel design techniques, are stress dependent, i.e. lateral and normal stress. Virgin strata stresses together with those induced by mining excavation require to be more comprehensively studied in order to relate laboratory rock test data to field condifions. Predictive formulae require to accommodate realistic stress field situations.

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