Memorandum

Chamber of Mines Certificate in Rock Engineering

Paper 2 Coal Option June 2004

Question 1 (Quotes from Coal Mine Rock Engineering, van der Merwe and Madden)

(a) **Duncan Swell Test**

"The Duncan swell test measures the unconfined swelling strain in one or more directions when a sample of rock is immersed in water. When testing borehole cores from coal measures strata it is only necessary to measure the swelling strain perpendicular to the laminations since, in rocks liable to swell, that swelling will greatly exceed that in other directions.

Samples are not prepared but are chosen with their ends approximately parallel. This reduces the costs and time involved and, above all, allows the testing of weak samples that would otherwise break up during machining.

The test procedure requires that swelling displacement should continue to be recorded until it reaches a constant level or passes a peak. This can be extremely time consuming and, for practical purposes, is not necessary. For the vast majority of specimens, 90% or more of their final swell will have taken place by the time 30 minutes have elapsed. For this reason a 30 minute swelling strain is determined. "

"At the end of the test the sample is immediately removed from the water. It is then assigned a rating from 1-6 according to its condition. A rating of 1 is assigned to an undisturbed sample and a rating of 6 to a totally degraded one. The swell index of the sample is then determined by multiplying the swelling strain by the condition rating."

Limitations: Difficult to test weak samples, swelling not the same in different directions

Used to predict floor conditions and reaction of floor to water.

(b) **Slake Durability Test**

"This test assesses the resistance offered by a rock sample to weakening and disintegration when subjected to two standard cycles of drying and wetting. The slaking fluid used in all instances is water. The International Standard calls for a representative sample comprising ten rock lumps, each weighing 40 to 60 g. The size of core typically used means that 40 to 60 g lumps can only be obtained from the more competent rock types. If only these rock are tested then the results would be biased towards good floor conditions. For this reason the lump requirement has been modified to 20 to 30 g unprepared lumps. The drying periods have been shortened from 2 to 6 hours to 0.5 to 2 hours in order to speed up the procedure and because the lumps are smaller.

Conventionally, a high swell index implies a poor rock, conversely a high slake durability index implies a good rock. To avoid confusion Buddery and Oldroyd

(1992) present the slake durability index as $100 - I_{d2}$. Both floor indices therefore increase as expected floor conditions get worse."

Limitations: Not a direct indicator, need a data base with back analysis to link index to floor conditions.

Used to predict mainly floor but also roof conditions.

(c) Impact Splitting Index

"An Impact Splitting Test was developed whereby a constant impact is applied to every 20 mm of drill core by dropping a 1,5 kg chisel from a pre-determined height according to core diameter, for example, 100 mm onto TNW core (60 mm diameter). When designing for roof support 2.0 m of the immediate roof is tested. The strata is divided into geotechnical units which are different to the geological units. The units are tested and the mean fracture spacing for each is obtained. The impact splitter cause weak or poorly cemented bedding planes and laminations to open, thus giving an indication of the likely in-situ behaviour when subjected to bending stresses, in some instances compounded by blasting."

Limitations: Not a direct measure of roof conditions, need correlation with data base.

Used for predicting roof conditions.

(d) RQD

"RQD is defined as the length of core in excess of 100 mm divided by the total length of a particular strata unit expressed as a percentage, and is used in several of the Rock Mass rating systems. However, the skill of the drill operator can influence the RQD result."

Determined by adding the lengths of units in a drill core longer than 100 mm and expressing that length as a percentage of the total length of core investigated.

Question 2

(a)

$$Weight = Spacing^2 x(0.5x15 + 0.6x25)x1.5 \quad kN$$

Bolt spacing (m)	Weight to be supported (kN)
1.0	*33.75*
1.5	*75.9375*
2.0	*135*
2.5	*210.9375*
3.0	*303.75*
3.5	*413.4375*

$$Bolt - strength = \sigma \frac{\pi d^2}{4} = 600000 \frac{\pi(.018)^2}{4} = 153kN$$

Thus the most suitable spacing from the table will be 2.0 m, as at greater spacings the load is greater than the strength of the steel.

(b)

Rebar length = anchor length + thickness of weak material

$$Shear - resistance = \tau = \frac{Load}{Area} = \frac{Load}{\pi dl}$$

Therefore $l = \dfrac{Load}{\pi d\tau} = \dfrac{135}{\pi(0.025)2000} = 0.86m$

Thus rebar length = 0.86 + 1.1 = 1.97 m

(c)

Required resin volume = Anchor length x (Cross section area of hole – Cross section area of rebar)

$= 0.86 \times \pi/4(d^2_h - d^2_r)$

$= 0.86 \times \pi/4 \times (0.025^2 - 0.018^2)$

$= 0.000203\ m^3$

$= 203\ cm^3$

$$Length = \frac{4Volume}{\pi d^2} = \frac{4x203}{\pi x2.2^2} = 53.4cm$$

(d)

Suspension is adequate under the conditions of a weak or thinly laminated layer in the immediate roof, followed by a strong layer above, which is strong enough to support itself plus the weight of the material underneath for the given road width. Joints should be largely absent or widely spaced and approximately vertical and there should not be a high magnitude of horizontal stress.

Beam creation is required where there is no strong layer suitable for suspension in the roof, and the only solution then is to use artificial roof supports to strengthen the roof material and so create a beam that is strong enough to bridge the given road width.

(e) (Extracted from Coal Mine Rock Engineering)

Step 1: Calculate the minimum thickness of the beam to be created

Step 2: Calculate maximum permissible sag in the centre of newly created beam

Step 3: Determine position of bolt at edge

Step 4: Calculate the maximum permissible inter layer displacement

Step 5: Compare the maximum allowable displacement, $\Delta\ell_d$*, with the possible displacement*

Step 6: Determine bolt length

Step 7: Calculate the required length of anchor that will have the required resistance

Step 8: Optimise resin capsule lengths

Question 3

(a)(i) Roof failure (Extracted from Coal Mine Rock Engineering)

Where roof failure occurs, the pillar height increases and the pillar strength is reduced. In severe cases, this can result in pillar failure, although the rubble from the roof has some strengthening effect on the pillars through the lateral confinement it supplies.

Top or bottom coaling can also accelerate pillar failure, especially in cases where jointing is present in the pillars. The higher the mining height, the greater the probability that joints may daylight in the pillar sides, which could accelerate the failure process.

(a)(ii) Floor failure (Extracted from Coal Mine Rock Engineering)

Where the floor material is weaker than the roof or the pillars, the pillars may punch into the floor. If the floor material exhibits plastic behaviour, the material may be displaced laterally underneath the pillars and the friction between this material and the pillars can induce sufficient tension in the pillars to result in tensile failure.

(a)(iii) Progressive scaling (Extracted from Coal Mine Rock Engineering)

When pillar widths are reduced by scaling, the safety factors are reduced by a double mechanism: the pillar stress increases and the strength is reduced. Once the combination of reduced strength and increased load exceeds the critical limit, the pillar will fail.

(b) The Tributary Area Theory is used to simplify the calculation of pillar loads in bord and pillar mining. It is based on the assumption that each pillar carries the load of the overburden above the pillar and the area around the pillar described by the centre lines of the surrounding roadways.

It is limited in use to areas where the pillars are approximately of the same size, the layout is regular and the panel width is at least equal to the mining depth.

Question 4

(a)

$$Load = \frac{0.025HC^2}{w^2} = \frac{0.025x60x14^2}{9^2} = 3.62MPa$$

$$Strength = 7.2\frac{w^{0.46}}{h^{0.66}} = 7.2\frac{9^{0.46}}{3^{0.66}} = 9.58MPa$$

$$SF = \frac{Strength}{Load} = \frac{9.57}{3.62} = 2.65$$

(b)
After splitting, pillars are 9 x 2 m with 5 m roadways.

Equivalent pillar width, w_e is

$$w_e = \frac{4A}{C} = \frac{4x9x2}{2(9+2)} = 3.27m$$

$$Load = \frac{.025H(w_1+B)(w_2+B)}{w\ w_2} = \frac{.025x60x14x7}{9x2} = 8.17MPa$$

$$Strength = 7.2\frac{w_e^{0.46}}{h^{0.66}} = 7.2\frac{3.27^{0.46}}{3^{0.66}} = 6.01MPa$$

$$SF = \frac{6.01}{8.17} = 0.74$$

According to attached table, probability to have a stable lay-out is 0.087. Thus the probability of failure is 1 - 0.087 = 0.913 (or 91.3%).

(c)
Extraction ratio is:

$$e = 1 - \frac{w_1 w_2}{C_1 C_2} = 1 - \frac{2x9}{7x14} = 0.82$$

Equivalent height is:

h_e = h x e = 3 x 0.82 = 2.46 m

Expected subsidence in pillar failure at depth < 80 m = $0.8h_e$ = 1.97 m

(d)
The width-to-height ratio of the pillars is 3.27/3 = 1.09.

At width-to-height ratio > 4, pillar failure cannot be violent. As this ratio is much lower than that, violent failure can be expected.

(Note: It is also acceptable if the candidate bases the criterium on the minimum pillar width as opposed to the equivalent width.)

(e)
The logical alternative method for secondary mining is checker board stooping.

If that is done, the pillar loads prior to checker boarding will double while the strength of the remaining pillars will remain the same. Consequently, the safety factors will halve, then equal to 2.65 / 2 = 1.33.

Then the failure probability will be 1 – 0.97 = 0.03 (or 3%).

The width-to-height ratio of the pillars will be 9 / 3 = 3.

Failure could still be violent, but only if the overburden fails completely (i.e. the system stiffness = 0). If failure occurs, it will be less violent.

The extraction ratio will be $\dfrac{28x28 - 2x9x9}{28x28} = 0.79$

This is slightly lower than with splitting, but the benefit is better stability and less danger of collapse during the process of mining.