Model Answers for Paper 3.3 2011-05

Q1.

Candidates must demonstrate an understanding of the concepts and should have sufficient knowledge of the environment in which they work.

a) .　　8 Marks

Candidates must be able to describe the formation of the orebody. They should describe geology features and indicate their orientations. The type of minerals, grades and dimensions of the mineralised zone should be described.

b)　　12 marks

Describe important characteristics for each rock type e.g.:

- Mineralogy
- Intact rock properties
- Brittleness/ductility
- Joint characteristics
- Foliation/lamination
- Alteration and tendency to weather (durability)

Where there is limited information, further testing and investigation should be indicated

Candidates should demonstrate and understanding of the concepts and be able to identify relevant material characteristics. Emphasis can be on the characteristics that will have the greatest influence.

c)　　5 Marks

Vertical stress = ρgh, k ratio

Comment on whether stress measurements have been undertaken or are required.

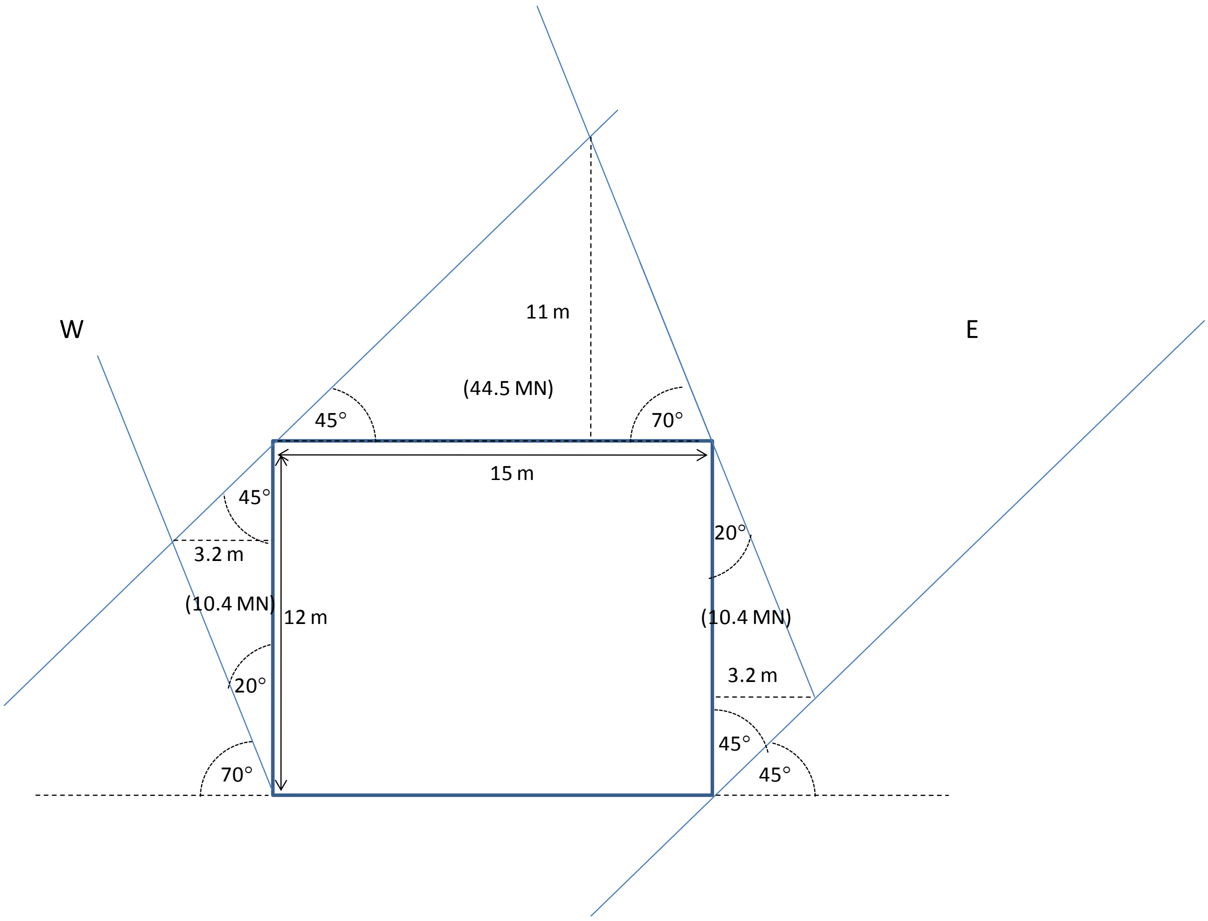
Stress to strength ratio

Comment on excavation stability in low/high stress environment as appropriate.

Q2.

a) *12 Marks*

Candidates should be able to identify that the strike of the joints is parallel to the long axis of the excavation and draw a section across the width of excavation showing the joints intersecting the corners of the excavation. One wedge in the roof and one in each side must be shown. (4)



Roof wedge

Wedge height

$$h = \frac{15 \times \tan 45 \times tan\,70}{tan45 + tan70}$$

h = 11.0 m (2)

Wedge Weight

W = 0.5 base x height x length x unit weight

W = 0.5 x 15 x 11 x 20 x 27

W = 44,450 kN = 44.5MN (2)

Side wedges

The two side wedges are the same size, just inverted.

Wedge height

$$h = \frac{12 \times tan20 \times tan45}{tan20 + tan45}$$

h = 3.2 m (2)

Wedge Weight

W = 0.5 base x height x length x unit weight

W = 0.5 x 12 x 3.2 x 20 x 27

W = 10,370 kN = 10.4MN (2)

Calculations per unit length are also acceptable.

b) *5 Marks*

Number of bolts

N = Wf/B

N = 44450 x 1.5/250 = 267 (3)

Area/cable

Area per bolt = (15x20)/267

= 1.12 m^2/cable

Pattern

15 cables per row (1.0 m spacing)

18 rows (1.1 m spacing)

Total cables = 270 (2) for any reasonable pattern correctly calculated

c) *7 Marks*

Choose wedge with 70° sliding angle (1)

Number of bolts

a = (90-70) + 15 = 35 (1) calculation of angle of bolt relative to critical joint plane

$$N = \frac{10370 \times (sin70 - cos70 \times tan21 - 0 \times A)}{250 \times (cos35 \times tan21 + 1.5 \times sin35)}$$

N = 45 *(3)*

Area/cable

Area per bolt = (12x20)/45

= 5.32 m^2/cable

Pattern

5 cables per row (2.4 m spacing)

9 rows (2.2 m spacing)

Total cables = 45 *(2) for any reasonable pattern correctly calculated*

d) *1 Marks*

Change the orientation of the excavation such that the major joints intersect at 45°. The more complex wedge geometries can be calculated using software such as Unwedge. If 380kN cables are used this would reduce the number of holes to be drilled. (1)

Q3.

a) *8 Marks*

Information

Host rock granite MRMR = 55

Depth of base of block cave = 500 m

Cave Angle

Lateral confinement is provided by the broken ore in the cave

Cave angle = 60° *(2)*

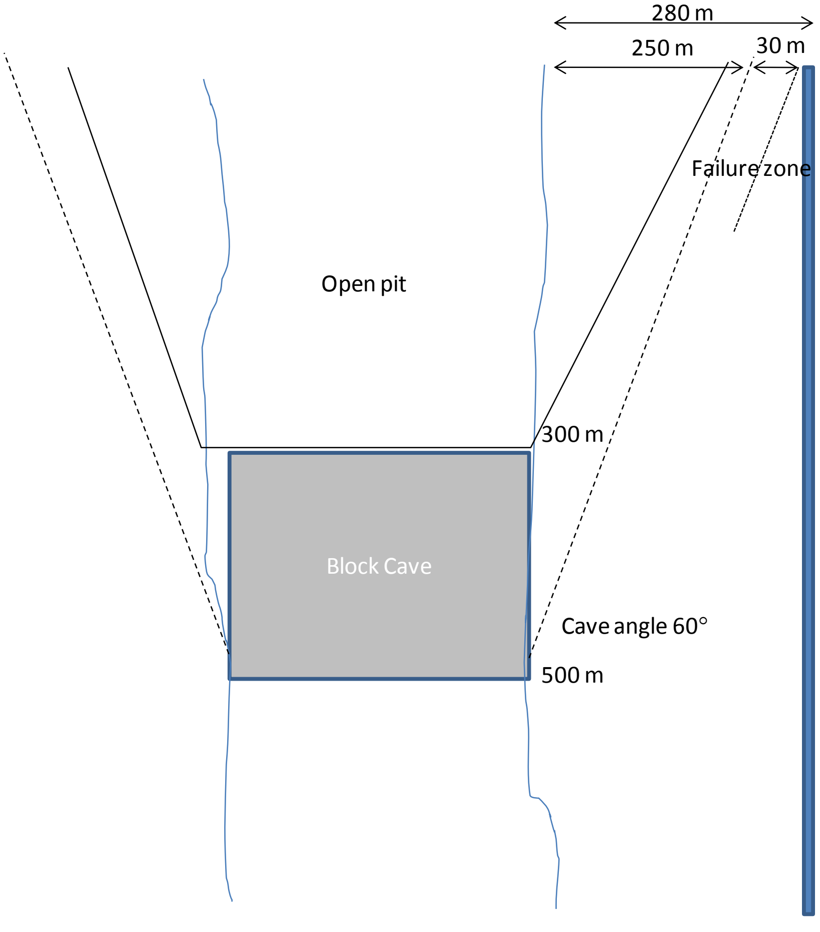
Extent of failure zone

Failure on surface

Extent = 30 m *(2)*

Horizontal distance from limit of orebody to a position on surface:

D = 500 x cos 60° + 30 = 280 m *(2)*

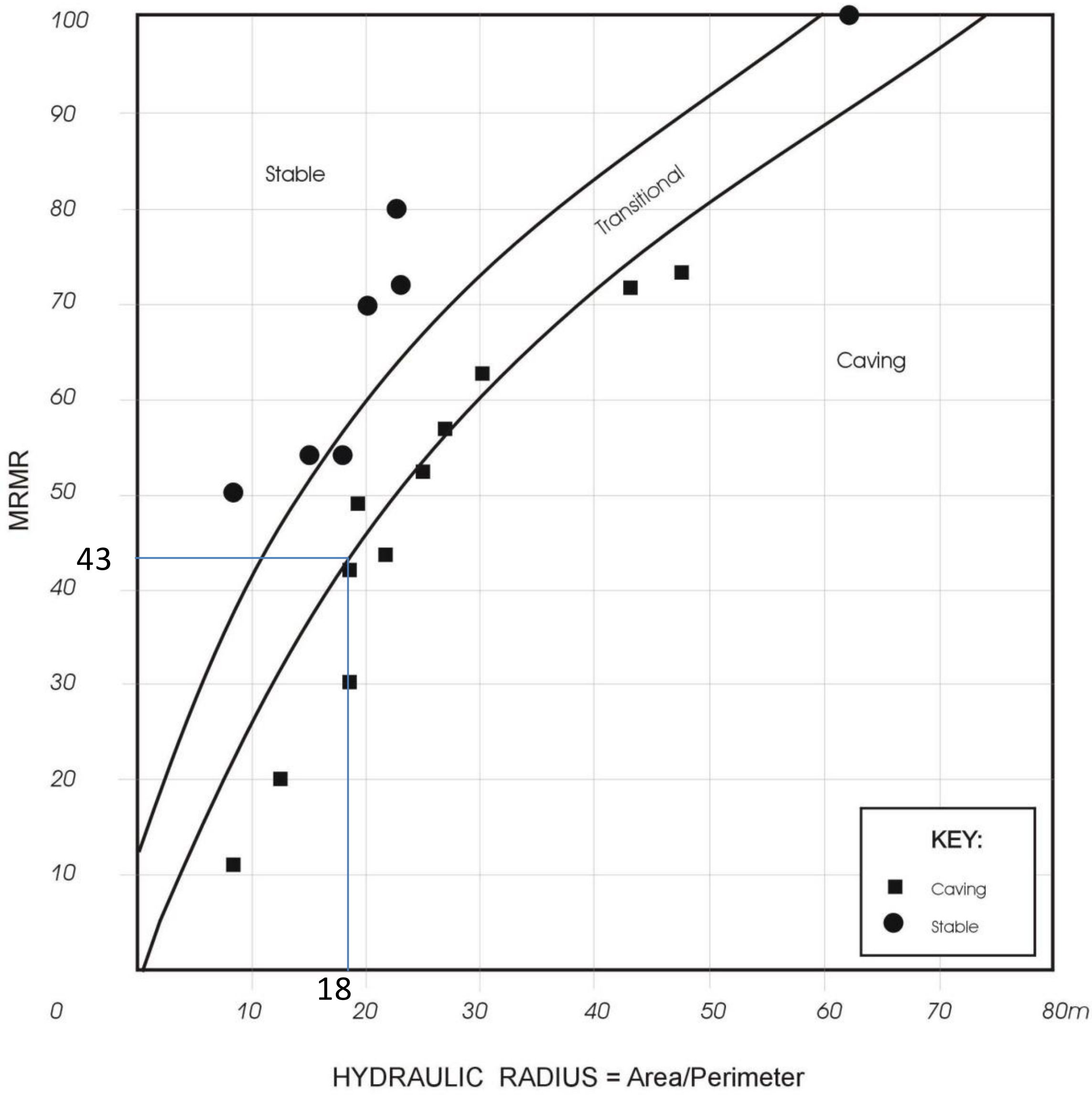


(2)

b) *8 Marks*

Kimberlite MRMR = 43

Hydraulic radius = 18 m (see graph) (3)



Distance to advance for caving

Hydraulic radius: $h = area/perimeter$

$h = wl/(2(w+l))$ w = width, l = length

$2hw+2hl = wl$

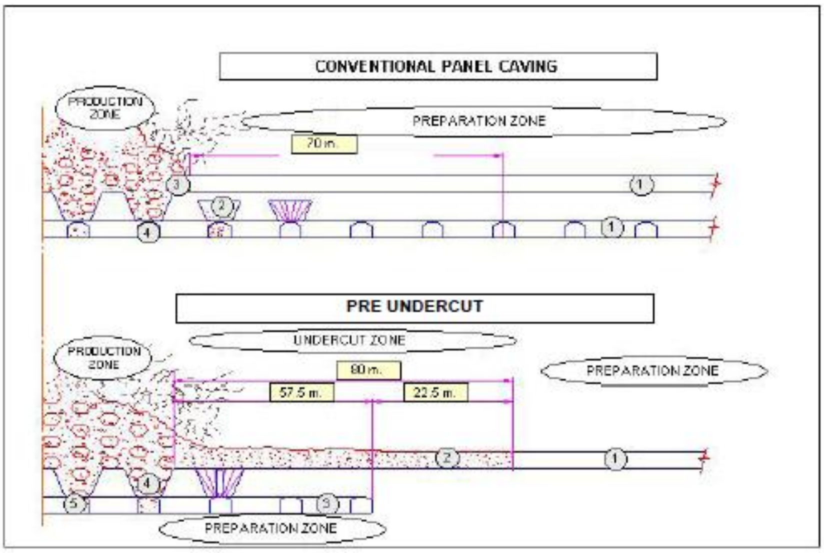
$l = 2hw/(w-2h) = 2 \times 18 \times 80/(80-2 \times 18)$ (4)

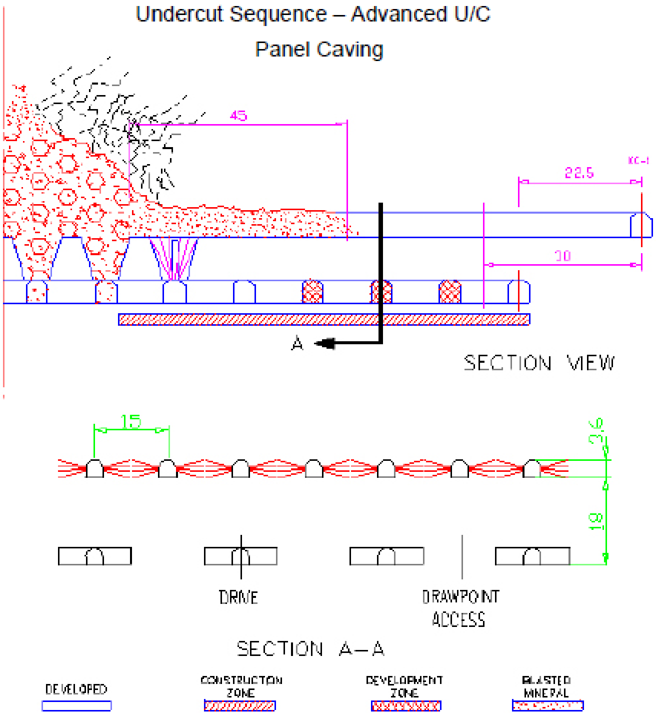
$l = 65$ m (1)

c) *9 Marks*

Describe both pre-undercut and advanced undercut

Brady and Brown pg. 472 to 474. Brown (Block caving handbook) pg 193 to 195





Q4.

a) *12 Marks*

Back (crown)

Q' = 14

A = 1 *Mining is <300 m below surface and the UCS of the rock is greater than 150 MPa*

B = 1 *The back is horizontal and the critical joint is almost 90° to the back*

C = 3 *Sliding mechanism, dip of critical joint = 90°*

N' = 13.5x1x1x3=40.5 *(2)*

Hydraulic Radius = 9.5 *See Potvin stability graph below (1)*

If the full orebody width is mined, then the maximum length of the stope along strike can be determined as follows:

h = area/perimeter

h = wl/(2(w+l)) *w = width, l = length*

2hw+2hl = wl *(1)*

l = 2hw/(w-2h) = 2x9.5x50/(50-2x9.5)

l = 30.6 m ≈ 30 m *(2)*

Hangingwall

Q' = 14

A = 1 *Mining is <300 m below surface and the UCS of the rock is greater than 150 MPa*

B = 0.3 *The critical joint is parallel to the hangingwall*

C = 7 *Slabbing mechanism, dip of stope face = 80°*

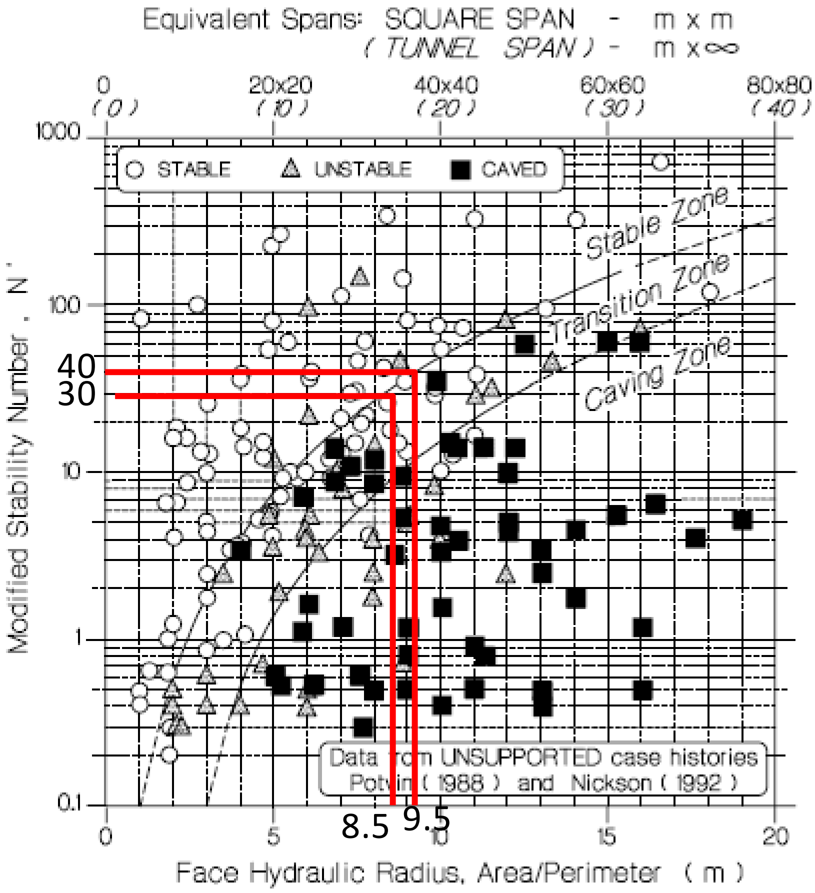
N' = 14x1x0.3x7=29 *(2)*

Hydraulic Radius = 8.5 *See Potvin stability graph below (1)*

If the maximum strike dimension is 30 m (from back calculation above), then the maximum height of the stope can be determined as follows:

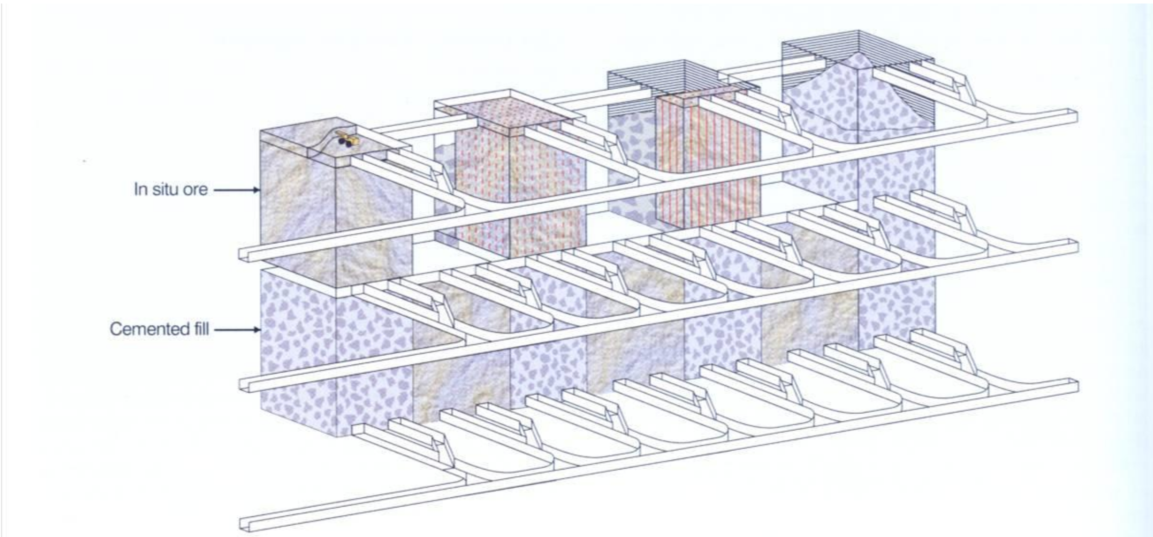
l = 2hw/(w-2h) = 2x8.5x30/(30-2x8.5) *(2)*

l = 39.2 m ≈ 40 m *(1)*



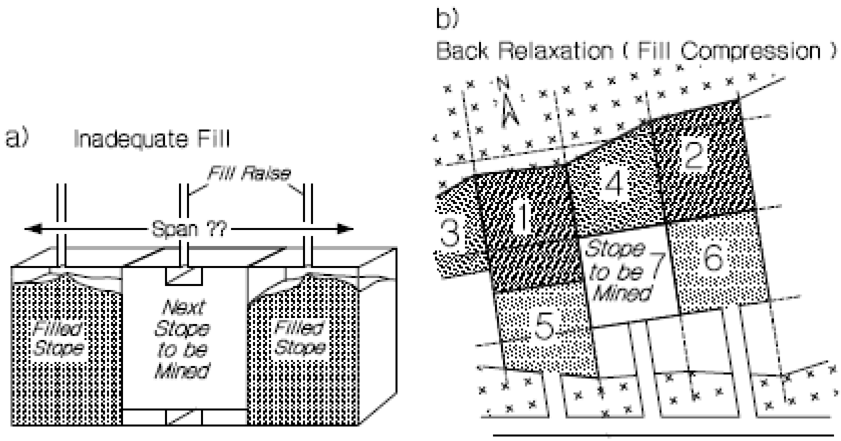
b) *8 Marks*

The strike dimension needs to be limited to 30 m to ensure stability of the stope back and the stope height needs to be limited to 40 m to ensure stability of the hangingwall. It is therefore necessary to use a longhole stoping and post filling method as indicated in the example below. Candidates should describe the method, drilling, blasting cleaning etc. (Sill and bench or ring drilling can be described).



c) *5 Marks*

Hutchinson and Diederichs pg 246. The estimation of stable hydraulic radii determined from the graph or used as input for stability evaluation assumes that the span being considered is fully bounded. This assumption is valid for unfilled stopes which are surrounded by fill (as in alternate block sequencing, for example). The surrounding fill must be tight to the walls and back of the stopes in order to be considered supporting elements. If this is not the case as shown in Fig. a), the true effective span for analysis may be much larger than the nominal stope panel. The same is true if the fill is highly compressible. In such a case, the Stability Graph Method is not applicable to the design of the unmined panel 7 in Fig b), for example.



The risk of injury to personnel can be minimised by ensuring that the stopes are non-entry. Either draw points can be designed at the base of the stope or remote loading of stopes.

Cable bolts can be used to improve the stability of stopes at a cost.