CHAMBER OF MINES OF SOUTH AFRICA

EXAMINATION PAPER

SUBJECT: CERTIFICATE IN ROCK MECHANICS PAPER 3.3 : MASSIVE UNDERGROUND MINING (HARD AND SOFT ROCK) **SUBJECT CODE:** COMRME **EXAMINATION DATE:** OCTOBER 2014 **TIME**: 3 HOURS	**EXAMINER:** J WALLS **MODERATOR:** W JOUGHIN **TOTAL MARKS:** [100] **PASS MARK:** (60%)

NUMBER OF PAGES:

THIS IS NOT AN OPENBOOK EXAMINATION – ONLY REFERENCES PROVIDED ARE ALLOWED

SPECIAL REQUIREMENTS:

1. Answer **all questions**. Answer the questions **legibly** in English.
2. Write your **ID Number** on the outside cover of each book used and on any graph paper or other loose sheets handed in.

NB: Your name **must not** appear on any answer book or loose sheets.

3. Show all calculations **and check calculations on which the answers are based.**
4. Hand-held electronic calculators may be used for calculations. Reference notes may not be programmed into calculators.
5. Write **legibly** in ink on the **right hand page** only – **left hand pages will not be marked**.
6. Illustrate your answers by means of sketches or diagrams wherever possible.
7. **Final answers** must be given to an accuracy which is typical of practical conditions, However be careful not to use too few decimal places during your calculations, as rounding errors may result in incorrect answers

   NB Ensure that the correct unit of measure (SI unit) are recorded as marks will be deducted from answers if the incorrect unit is used. (even if the calculated value is correct).
8. In answering the questions, full advantage should be taken of your practical experience as well as data given.
9. Please note that you are not allowed to contact your examiner or moderator regarding this examination.
10. Cell phones are **NOT** allowed in the examination room.

QUESTION 1 – OREBODY CHARACTERISATION

To select the most suitable mining method for economic extraction of an orebody and determine design criteria for stable underground excavations, the orebody and host rock characteristics, amongst other parameters, must be investigated.

1.1 Describe the geotechnical parameters required for rock mass classification using Laubscher's Mining Rock Mass Rating system. (3)

1.2 Describe a site investigation program that you would undertake to obtain the data and how the parameters are applied to the classification system. (10)

1.3 Discuss any other factors that influence selection of a suitable mining method. (3)

1.4 How you would apply the parameters obtained in (1.1) to (1.3) to the selected approach? Discuss any assumptions you must make and how this affects the design. (4)

[20 MARKS]

QUESTION 2 – CAVING

A massive, near-vertical copper orebody has been extracted to a depth of 600 m by surface mining. Underground extraction is planned by cave mining where the undercut is situated at 1 050 m below surface. A horizontal section through the orebody is roughly oblong, spanning approximately 150 m in width and 600 m in length. The orebody is fairly well jointed with a Mining Rock Mass Rating (MRMR) approximately 50, hosted within slightly less jointed pyroxenite with an MRMR approximately 70.

2.1 Determine the cave angle and the extent of the zone of failure at 800 m below surface. Comment on the stability of an access ramp that approaches to within 100 m from the limit of the orebody at this depth? Illustrate your answer using a sketch. (7)

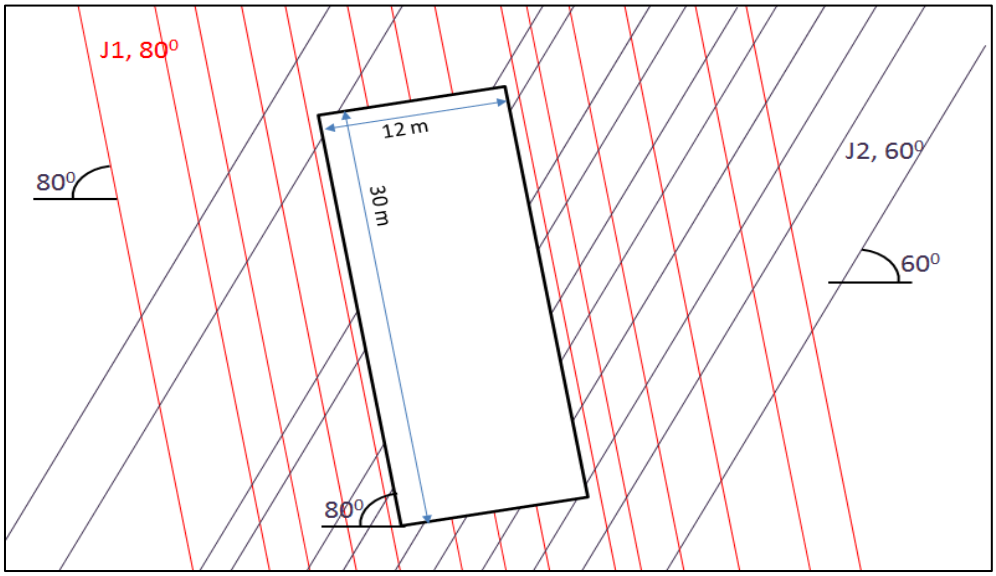
2.2 Discuss how stress-driven damage can be averted on the production level by strategic sequencing of the undercut. Use sketches to illustrate your answer (6)

2.3 The orebody is intersected by high strength dolerite intrusions which result in uneven distribution in fragmentation and drawdown of the caved material as well as large block sizes. As a result, the production level is expected to undergo damage. Discuss strategies other than sequencing of the undercut that you could recommend to protect the production level. (7)

[20 MARKS]

QUESTION 3 – POTVIN'S STABILITY METHOD FOR LONG-HOLE OPEN STOPING

Longhole open stoping is used to mine a 12 m wide orebody, dipping at 80^0 to the west with a known strike length of 600 m. Mining will commence at a depth of 150 m below surface. The individual stopes will be 40 m long and 30 m high. Elastic modelling has been carried out to determine the stress distribution around the stope (results are included with the tables of information). A geotechnical investigation has revealed that the rock mass quality is quite good (Q' = 14) and the intact rock strength is 120 MPa. Two major joint sets, striking roughly parallel to the strike of the orebody, influence the rock mass surrounding the excavation, illustrated below.



3.1 Calculate the modified stability number (N') for the stope crown and each of the side-walls. Present your answer in a table and list the stress values, strength to stress ratio, critical joint set, relevant angles, failure mechanism and the values of the A, B and C parameters. (7)

3.2 Calculate the hydraulic radius for the crown and walls. Determine whether the stope crown and walls are stable, transitional or caving and estimate the ELOS. Plot these values on the modified stability and ELOS graphs and hand in the annotated graphs with your script. (6)

3.3 Calculate the percentage dilution and indicate what action is required, if any. Present your results and comments in a table. (7)

[20 MARKS]

QUESTION 4 – ROCK BREAKING

Underground mining in hard-rock settings is highly dependent on successfully executing efficient blasting procedures.

4.1 In ground conditions which are susceptible to damage when subjected to blasting, it may be necessary to produce excavations with smooth, undisturbed profiles. With the aid of sketches, discuss and compare the principles of perimeter blasting by pre-splitting and smooth blasting. (8)

4.2 Sketch the blast fracture pattern that you would observe around a blast socket in an environment where there is a significant difference between the major and minor principal stresses. Clearly indicate the direction of principal stresses in your sketch. (3)

4.3 List and discuss measurement techniques that can be used in the field to evaluate the performance of explosives. (6)

4.4 List and briefly discuss rock properties that determine the performance of a particular explosive. (3)

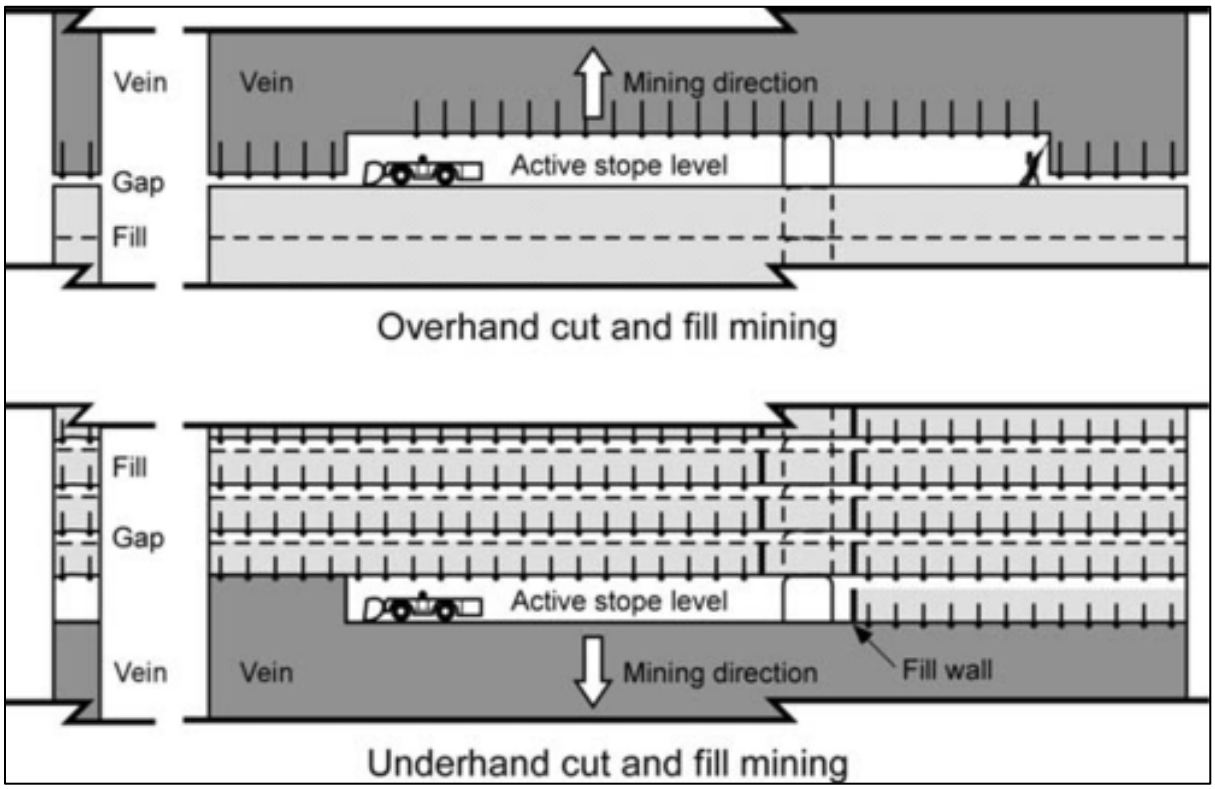
[20 MARKS]

QUESTION 5 –BACKFILL

Backfill can be generated from several types of constituent materials and fulfil various functions at an underground mine site.

5.1 Compare and contrast the characteristics, advantages and disadvantages of three types of backfill systems. Present your answer in a table. (10)

5.2 Discuss the use of backfill in the mining methods illustrated below. Would you recommend the backfill be cemented or uncemented? Discuss critical performance requirements. (5)



5.3 Access into secondary mining areas often requires some development through or undercutting beneath backfill such as drift development or crown pillar removal. Discuss hazards and failure mechanisms you would anticipate, with suggestions for reinforcement strategies. (5)

[20 MARKS]

TOTAL MARKS: [100]

REFERENCE SHEET

Cave angle and failure zone (Laubscher 1990)

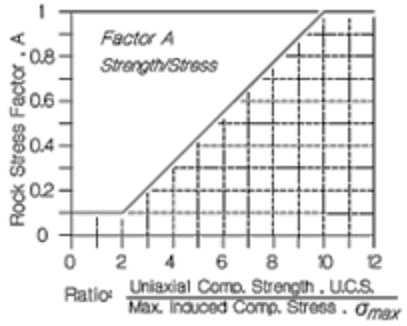
	Condition	Depth (m)	Adjusted *MRMR* 100-81	80-61	60-41	40-21	20-0
Cave angle	No lateral restraint	100	80	70	60	50	40
		500	70	60	50	40	30
	Lateral restraint from caved material	100	90	80	70	60	50
		500	80	70	60	50	40
Extent of failure zone	Surface	100	10 m	20 m	30 m	50 m	75 m
		500	10 m	20 m	30 m	50 m	75 m
	Underground	100	10 m	20 m	30 m	50 m	100 m
		500	20 m	30 m	50 m	100 m	200 m

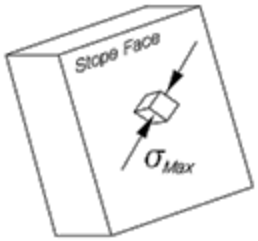
TABLE XIII
MINING METHOD RELATED TO MRMR

Class rating	5 0–20	4 21–40	3 41–60	2 61–80	1 80–100
Block Caving					
Undercut SI, m	1–8	8–18	18–32	32–50	+50
Cavability	Very good	Good	Fair	Poor	Very poor
Fragmentation, m	0,01–0,3	0,1–2,0	0,4–5	1,5–9	3–20
2nd lay-on blast/drill, g/t	0–50	50–150	150–400	400–700	+700
	0–20	20–60	60–150	150–250	+250
Hangups as % of tonnage	0	15	30	45	>60
Dia. of draw zone, m	6–7	8–9	10–11,5	12–13,5	15
Drawpoint span, m					
Grizzly	5–7	7–10	9–12		
Slusher	5–7	7–10	9–12		
LHD, m	9	9–13	11–15	13–18	
Brow support	Steel and concrete Reinf. concrete		Concrete	Blast protection	
Drift support	Lining, rock reinf., repair techniques		Lining, reinf.	Rock reinf.	
Width of point, m	1,5–2,4	2,4–3,5	2,4–4	4	
Direction of advance	Towards low stress		Towards high stress		
Comments	Fine fragmentation, poor ground, heavy support, repairs	Medium fragmentation, good ground, fair support	Medium coarse fragmentation, good drill hangups	Coarse fragmentation, large LHDs, drill hangups	
Sub-level Caving					
Loss of holes	Excessive	Fair	Negligible	Nil	Nil
Brow wear	Excessive	Fair	Low	Nil	Nil
Support	Heavy	Medium	Low	Localized	Nil
Dilution	Very high	High	Medium	Low	Very low
Cave SI, m	1–8	8–18	18–32	32–50	+50
Comments	Not practicable	Applicable	Suitable	Suitable	Suitable, large HW cave area
Sub-level Open Stoping					
Minimum span, m	1–5	5–20	20–30	30–80	100
Stable area, i.e. SI, m	N/A	1–8	8–16	16–35	+35

N/A = Not available

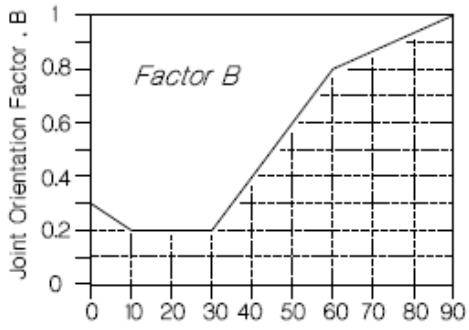
Determine maximum induced tangential stress (compressive) acting at the centre centre of the stope face being considered. Obtain uniaxial compressive strength strength for the intact rock. Evaluate Stress Factor, *A*, using the graph below:



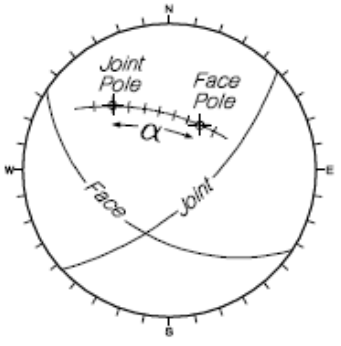


Obtain σ_{max} from 2D or (preferably) 3D numerical stress modelling.

Horizontal Back	Inclined Wall	Vertical Wall	True Angle between Face & Joint	Potvin Factor *B*
Joints / Face			$\alpha = 90°$	1.0
	α		$\alpha = 60°$	0.8
			$\alpha = 45°$	0.5
			$\alpha = 30°$	0.2
			$\alpha = 0°$	0.3

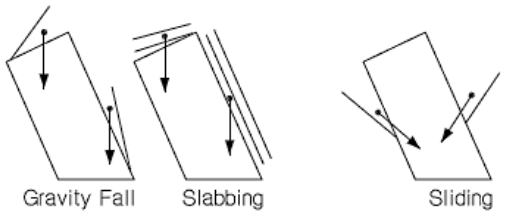


True Angle Between Face and Joint
(Angle α Between Poles)

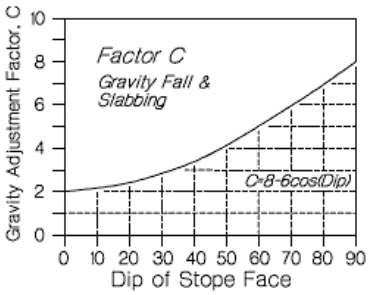


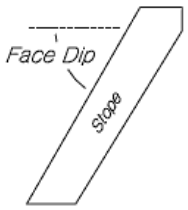
Determination of the minimum or true angle between two planes = Angle α between poles

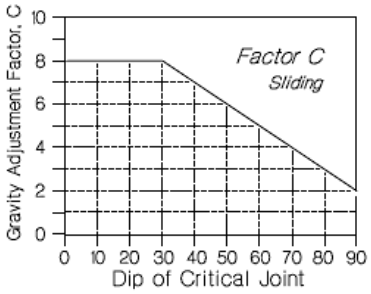
1) Determine the most likely mode of structural failure in case study using the figures below:

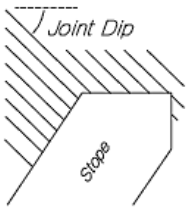


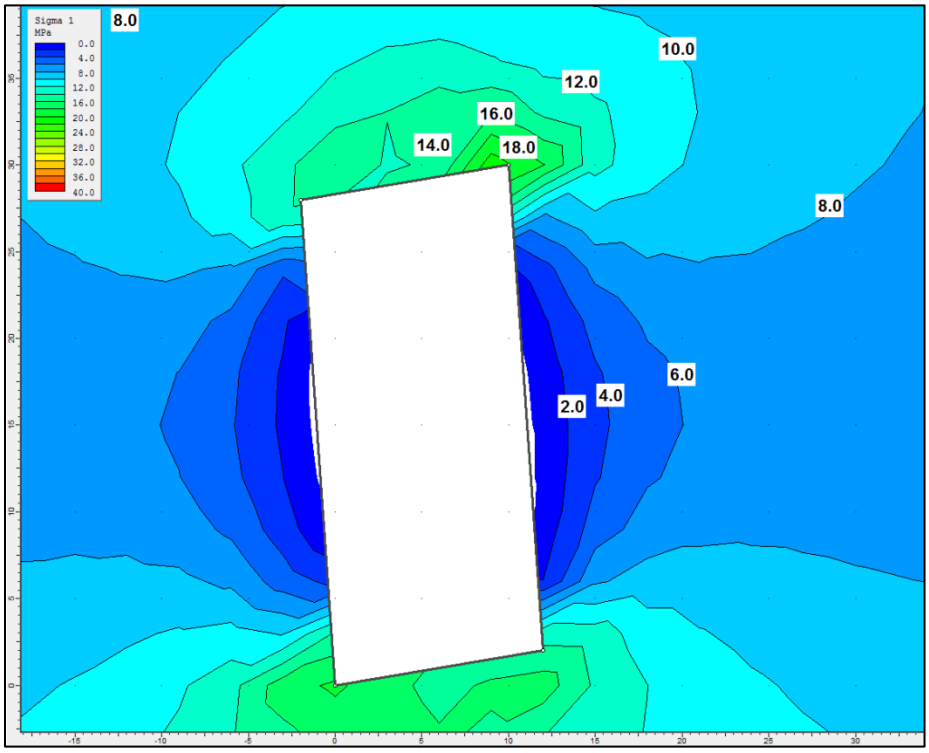
2) Next determine the gravity adjustment factor, C, based on the failure mode using the appropriate chart below.











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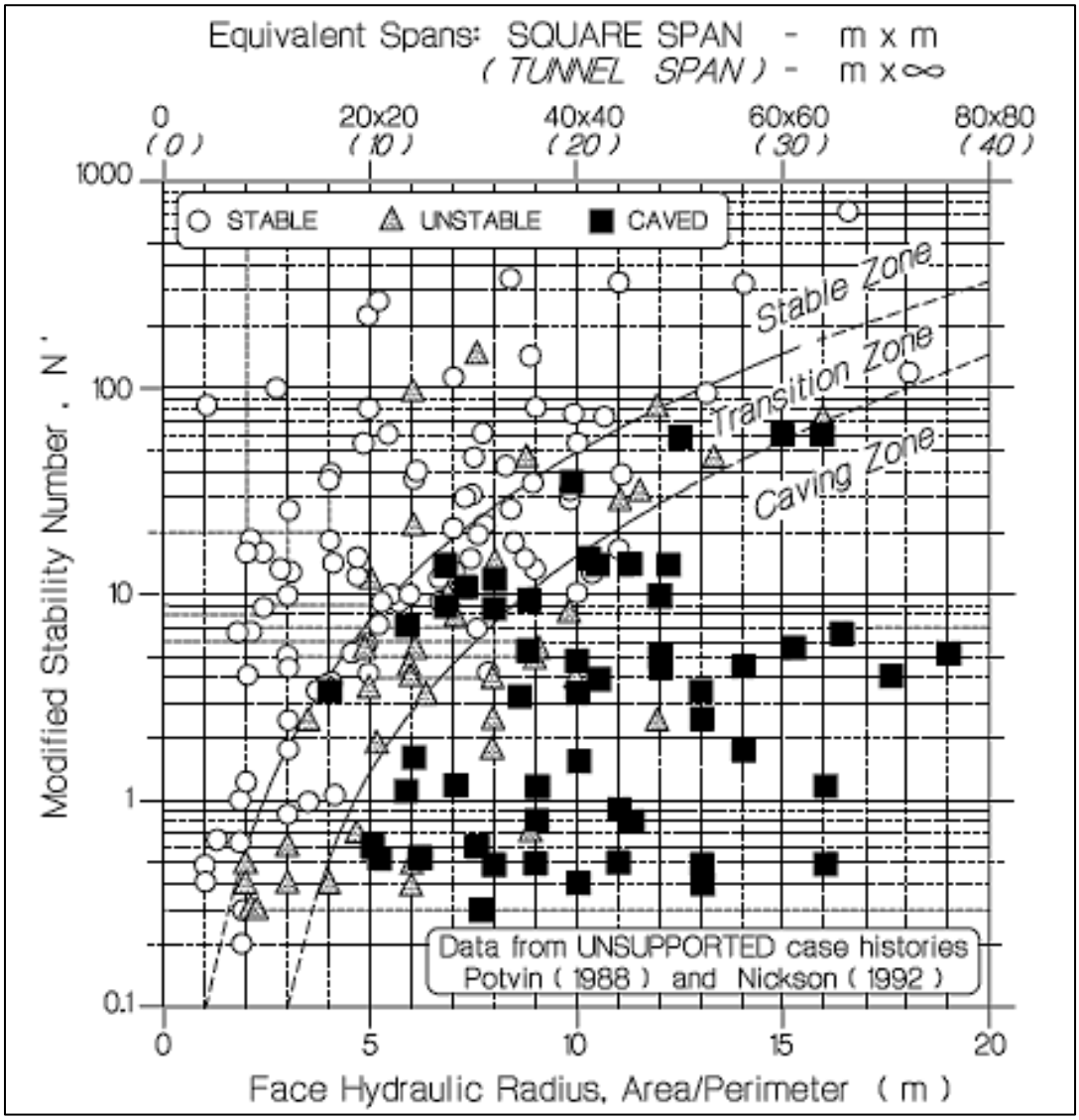

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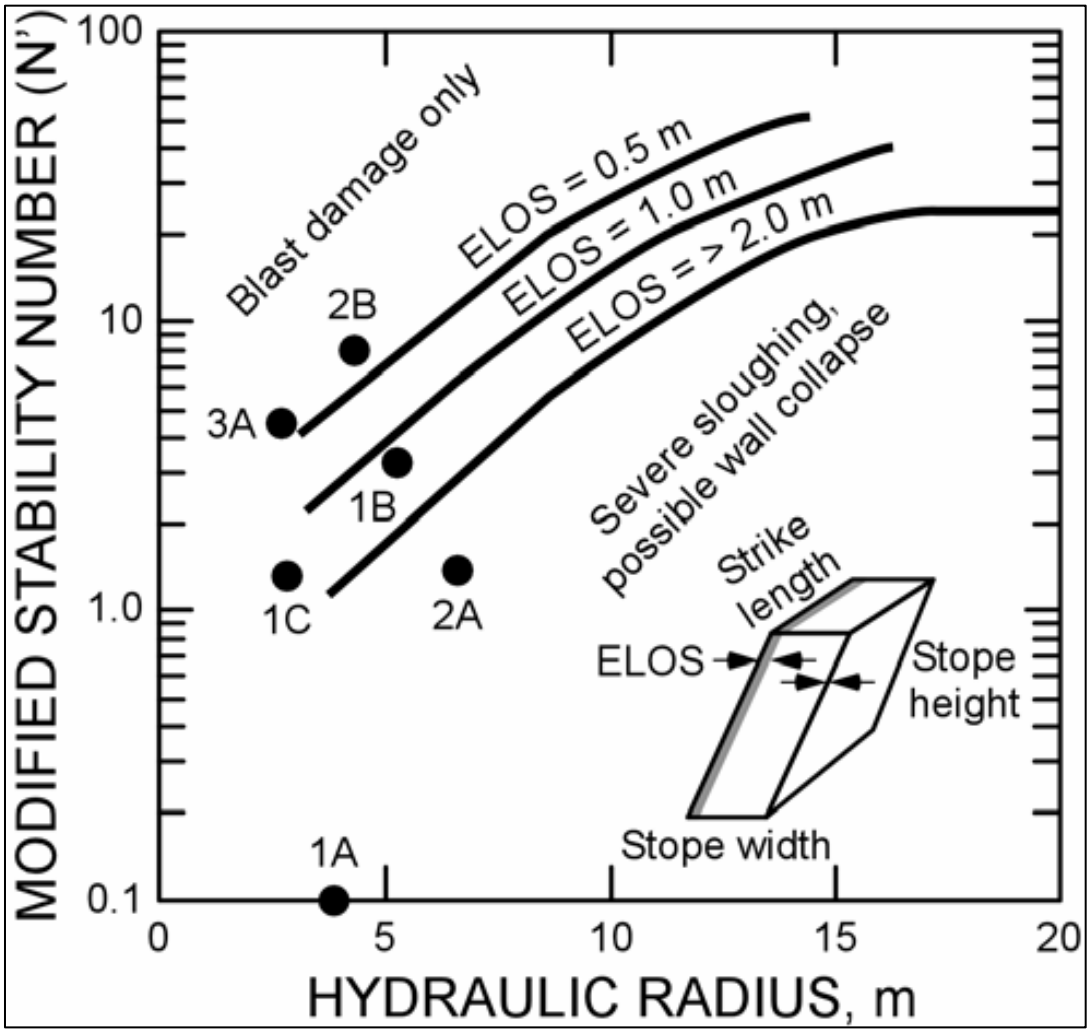

Equivalent Linear Overbreak / Slough