TECHNICAL MEMORANDUM

DATE February 18, 2016 **REFERENCE No.** 1520790-014-TM-Rev0-3000

TO Ryan Brown
Mount Polley Mining Corporation

CC Jetzen Loo and Richard Goodwin

FROM Donald Tolfree and Darren Kennard **EMAIL** Donald_Tolfree@golder.com; Darren_Kennard@golder.com

MOUNT POLLEY UNDERGROUND STOPE STABILITY REVIEW

1.0 INTRODUCTION

This technical memorandum presents a summary of a site visit and underground inspection performed by Donald Tolfree of Golder Associates Limited (Golder) to the Mount Polley Mine – Boundary Zone underground (Mount Polley) from July 27 through July 30, 2015. The results of a stability back-analysis of the current open stope (Block A), and recommendations for future stope mining adjacent to Block A are also presented.

The objective of the underground inspection was to map representative rock mass ground conditions, and ground support and stope wall performance in the vicinity of Block A. The underground tour focused on the areas close to the stope void and, for mapping purposes, areas that were considered representative of the ore body and host rock.

Quantification of the stability condition of the existing Block A using the field observations and empirical means was carried out to provide a calibrated stability baseline. Estimated hypothetical maximum unsupported and supported stope sizes for the rock mass present in the area were determined to aid in the planning of the extraction of the next four stopes.

The analyses focused on the potential stability condition of the rock mass resulting as stopes adjacent to Block A are sequentially extracted. The current mine plan requires the use of cemented backfill to allow extraction of ore adjacent to Block A. The comments regarding the details of the backfilling plan including stability of the backfill, mix design, and delivery system are not part of the scope of this work.

A presentation of the initial findings was delivered on the last day on site and is included in Appendix A.

2.0 BACKGROUND

The Boundary zone is composed of three main rock types which are breccia, red monzonite and grey monzonite. There are also porphyry dykes that present in the underground workings, some minor faulting on the overcut of the Block A and a large shallow dipping fault that intersects the stopes on the 782 level (Rockland 2013). The stope limits are economically based; therefore the host rock and ore body are composed of the similar material.

Hughes and Associates (2012) reported that majority of the rockmass had a rock mass rating (RMR_{76}) of "fair" (values between 41 and 60), with ratings ranging from "Poor" (21 to 40) to "Good" (61 to 80). It was also reported that the rock mass is heavily jointed with six joint sets mapped.

The stoping area in the Boundary Zone is approximately 63 m long, 20 m wide and 100 m high and is approximately 200 m below ground surface (to the top of the stopes). The stoping area was initially separated into four blocks (A, B, C and D) as shown in Figure 1. The original mine plan assumed a standard bottom-up panel retreat stoping sequence with stopes that were approximately 30 m high, and cemented rockfill as backfill after each lift, to maintain regional stability. A mine design update combines Blocks B and C and a portion of Block D into one stope Big Block B, outlined in yellow in Figure 1.

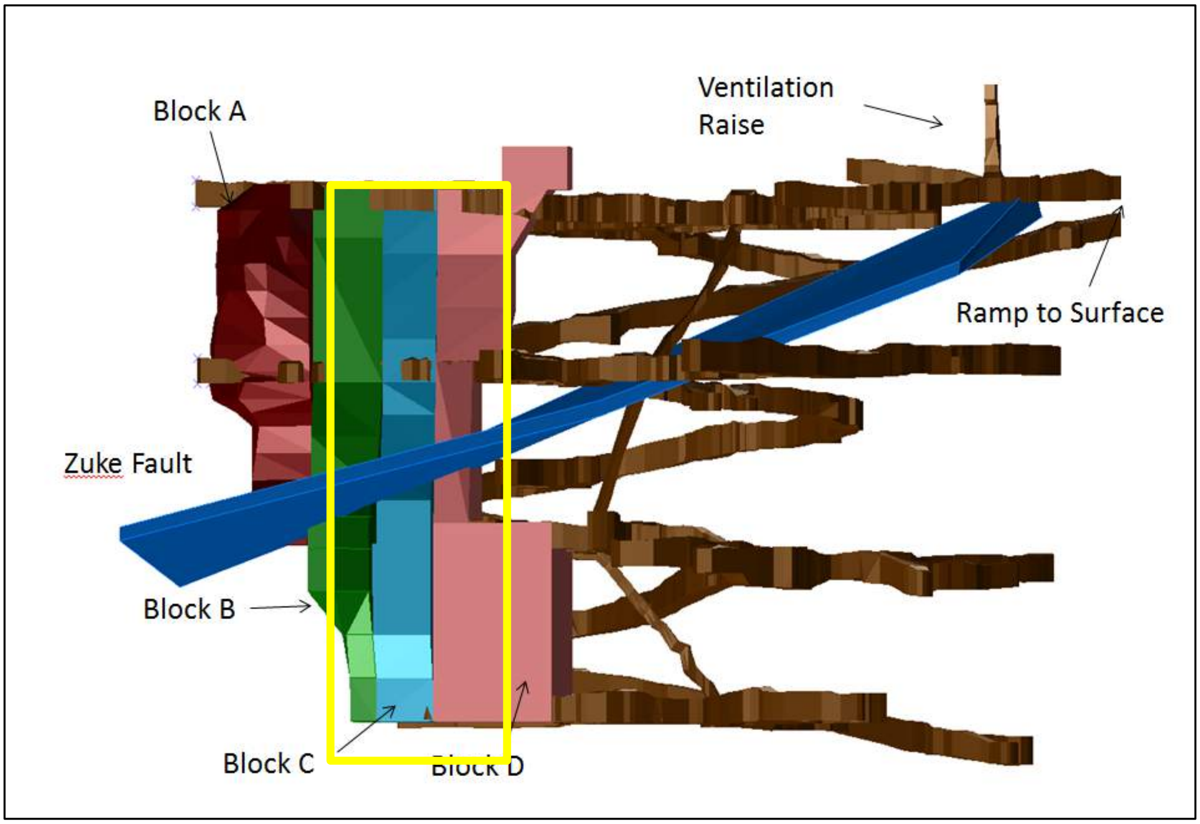

Figure 1: Longitudinal Section of the Boundary Zone Showing the Stoping Blocks and Development

Ground support approaches adopted for the area were based on recommendations provided by Hughes and Associates (Hughes 2012) which were dependant on the opening type (e.g., stope development openings or stope backs and walls), and generally called for increasing support for decreasing rock mass rating. General minimum ground support recommendations for stopes included 12 m long, fully grouted cable bolts be installed in the back of the stope on a 1.5 m by 1.5 m pattern and 9 m long cable bolts on pattern ranging from 2.0 m by 2.0 m to 1.7 m by 1.7 m be installed in the stope walls where possible, dependant on the rock mass rating. There was no distinction made between the footwall and hangingwall (sidewalls), and end walls. The recommendations also include cable bolting the vertical stope walls from the intermediate drift (for example, the 812 Level of Block A). The empirical assessment used to support development of these stope ground support recommendations assumed a hydraulic radius of 4.3 m for the back and 6.0 m for the vertical walls and an unconfined compressive strength (UCS) of 100 MPa (Hughes 2010).

The first stope mined, Block A, was designed to be 20 m long, 25 m wide and 67 m high, which represents a hydraulic radius of 6 m for the back and 9 m for the sidewall walls. The block was excavated approximately one year prior to the site visit. Cavity monitoring surveys were not available and therefore the designed dimensions were assumed to be representative of the void created (i.e., it was assumed there has been no significant over break).

3.0 OBSERVED CONDITIONS

The underground tour consisted of visiting the overcut of Block A (the 842 Level) to observe the condition of the stope back, visiting the intermediate levels of 812 and 752 to collect mapping data and visiting the area of the Zuke fault.

At the time of the visit the back of Block A seemed stable. However, there is evidence that could suggest the installed ground support is taking load. Figure 2 shows the approximate location of photographs that show various cable bolts that may be taking load and are shown in Photographs 1 and 2. Untwisting of a cable bolt is indicative of the bolt taking load and observations suggest this may be the case for these cables. Such loading can be indicative of loss of confinement (tension) in the stope walls and localised block movement.

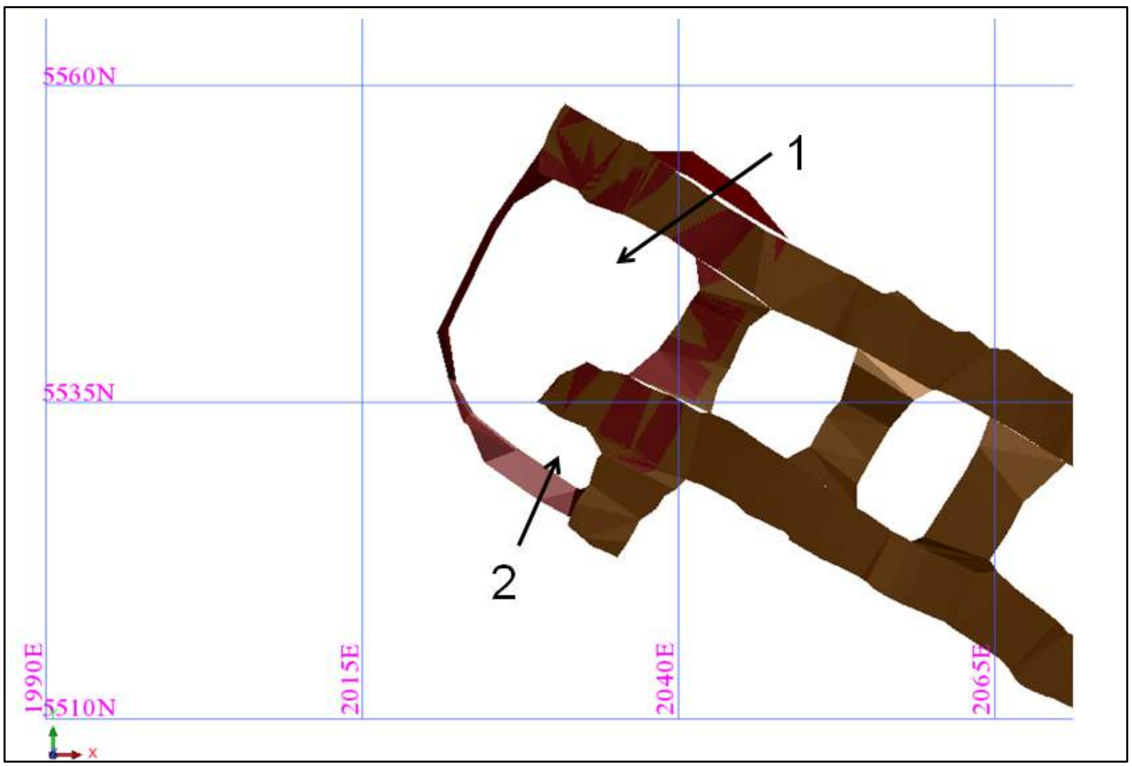

Figure 2: Plan View of the Location of the Pictures Taken Underground

Photograph 1: The Back of Block A Showing a Cable Bolt that is Untwisting (Picture 1)

Photograph 2: The South Wall of Block A showing a Cable Bolt that is Untwisting (Picture 2)

The Zuke fault is moderate to shallow dipping, and impacts the lower portion of Block A, and midpoint of Blocks B through D (Figure 5). The fault was observed on the 812 Level (Figure 6) where it is characterized as gouge filled and at a minimum 30 cm thick and at a maximum greater than 1 m thick (see Photograph 3).

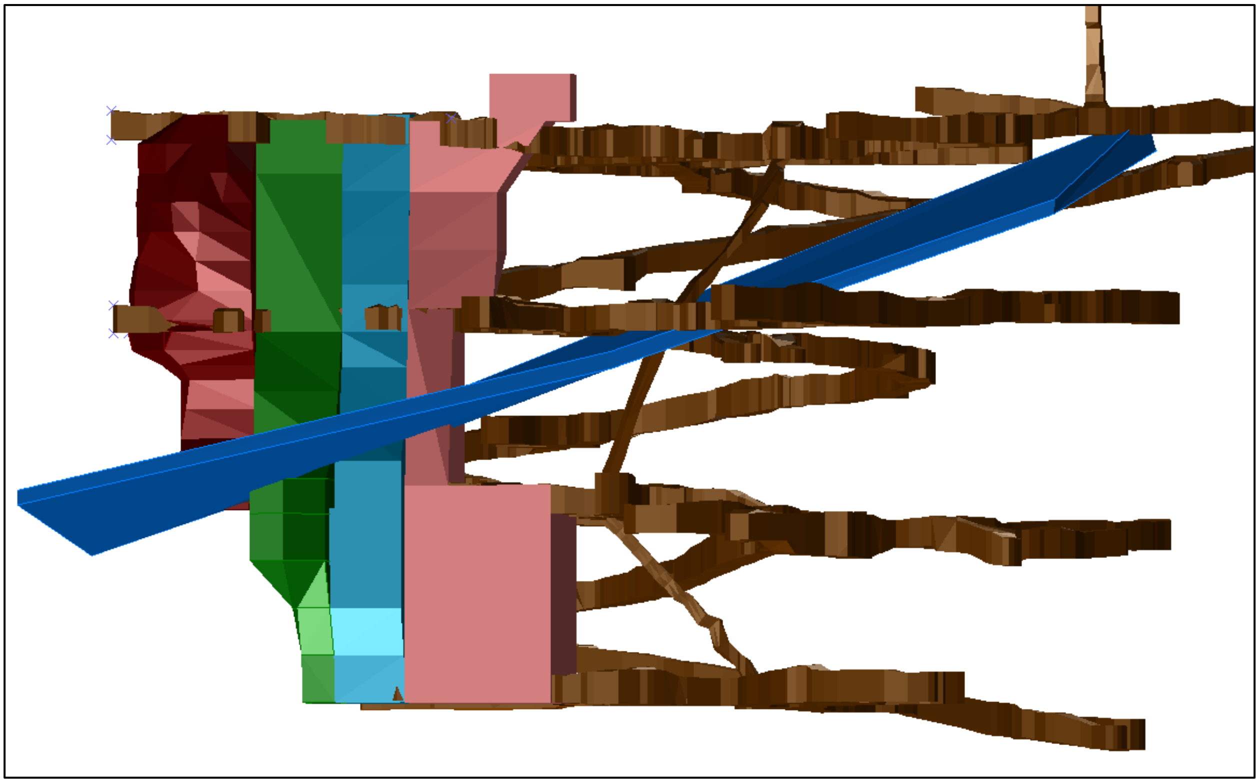

Figure 3: Longitudinal Section Showing the Location of the Zuke Fault Relative to the Stoping Blocks

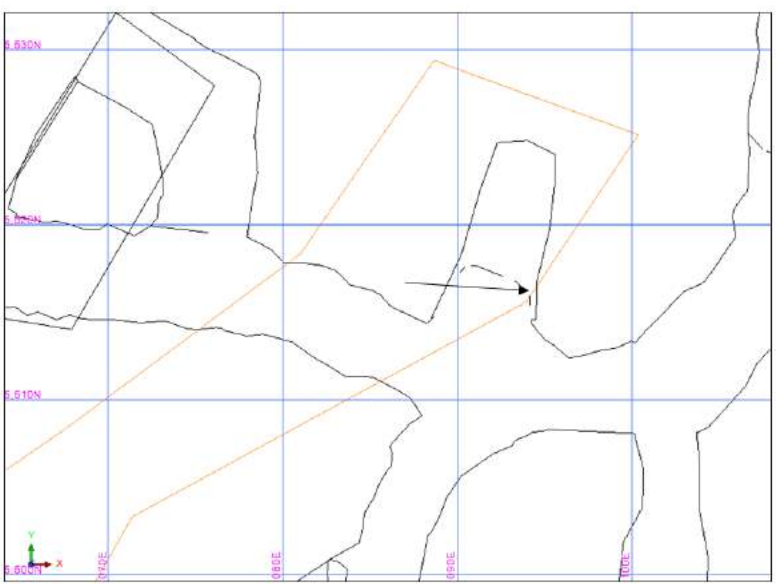

Figure 4: Location of the Zuke Fault Picture (812 Level)

Photograph 3: Picture of the Zuke Fault (Picture 3)

4.0 COMPARISON GEOTECHNICAL MAPPING

Geotechnical mapping using the Q rock mass classification system (Barton 1974) was completed on the 812 and 752 levels. The Q system is commonly used in underground mining and tunnelling projects and derives a numeric measurement of rock mass quality. Common applications of the system include its use in empirical based assessments of ground support requirements and open stope sizing.

The elements of the Q system are summarized into equation 1:

(1) $Q = RQD/Jn * Jr/Ja * Jw/SRF$

Where:

RQD = Rock quality designation with a minimum of 10

Jn = Joint number

Jr = Joint roughness

Ja = Joint alteration

Jw = Joint water factor

SRF = Stress reduction factor

The RQD/Jn parameter represents block size. The Jr/Ja parameter represents the inter-block shear strength and the Jw/SRF parameter represents the active stress. For the purposes of this evaluation, the Jw/SRF parameter was set to 1 because the active stress is accounted for in the stope stability assessment.

The main purpose of the mapping was to become familiar with the rock mass and to compare Golder's assessment of rock mass quality to what was assumed in previous stability assessments. Table 1 presents a summary of the mapping data collected by Golder and a comparison with the results from Hughes and Associates (2012) and Rockland Ltd. (2013). For the purposes of the comparison it is assumed that the Geological Strength Index (GSI) is equivalent to RMR_{76} (Bieniawski,1976) and Q is related to RMR_{76} with equation 2 assuming that the stress equivalent ratio and joint water parameter equal 1.

(2) $RMR_{76} = 9 * \ln Q + 44$

There is a good correlation between the different mapping exercises.

Table 1: Comparison of Mapping Results

Source	Q	RMR_{76}
Hughes 2010	3.4 to 10.3	55 to 65
Rockland 2013*	3.4 to 10.3	55 to 65
Golder 2015	4.9	58

*Rockland estimated a GSI

5.0 STOPE STABILITY ASSESSMENT

The stability condition of the current void, Block A, was assessed using the Mathews method (Stewart and Forsyth, 1995) and Potvin's stability chart (Potvin et al., 1989) approaches. These methods use an empirical relationship between depth of mining, rock mass quality, and the size of the various walls of an open stope void (back, endwalls, sidewalls which for a dipping stope can be divided between hangingwall and footwall). The Mathews approach is commonly used to size unsupported stopes and the Potvin method includes approaches for assessment of stope support requirements (e.g., cable bolting).

Geotechnical parameters, summarized in the form of a stability number (N) or a modified stability number (N'), which is calculated as follows:

N or N' = Q' * A * B * C

Where:

Q' = modified Rock Mass rating (Q)

A = stress factor

B = rock defect factor

C = design surface orientation factor

The A factor estimates the impact of insitu stress on the stability of the opening and uses the unconfined compressive strength (UCS) of the rock. The insitu stress was estimated using a rock density of 2,700 kg/m^3, a depth to the mid height of the stope of 240 m, and an insitu stress ratio of 1.5. UCS test results were not available, therefore an estimated 100 MPa, based on the rock type (Monzonite), was used. This value is consistent with previous work. However, some laboratory intact rock strength testing is required to confirm this value.

The B factor is dependent the dip of the major joint sets. Mapping indicates that there are numerous joint sets in the Boundary zone of which three are steeply dipping relatively the vertical walls. There is one planar set that will be influential in the back stability. The impact of this factor on the stability number is different in the Mathew method than the Potvin method.

For the Mathews method, the C factor is dependent on the dip of the design face. For the Potvin method, the C factor is dependent on the mode of failure, sliding or gravity fall/slabbing, and is either the dip of the controlling joint set, or the dip of the design face. In both cases, Block A is a vertical stope.

5.1.1 Mathews Method Results

Input data for Mathews method approach and results are summarized in Table 2.

Table 2: Summary of Mathews Method Stability Number Inputs for Block A

	Q'			**A**	**B**	**C**	**HR**	**Stability Number (N)**		
	Low	Average	High					Low	Average	High
Back	1.5	5	12	0.45	0.5	1	5.6	0.3	1.1	2.7
Vertical End	1.5	5	12	1.00	0.4	8	7.6	4.8	16.0	38.4
Sidewall (Hangingwall / Footwall)	1.5	5	12	1.00	0.4	8	9.0	4.8	16.0	38.4

Empirical based predictions of the stability condition of an unsupported stope are made by plotting the resulting range of stability number (N) with the hydraulic radius of a particular stope surface. The hydraulic radius (HR) is calculated by the division of the area of the surface over the perimeter of the surface

Golder has extensive experience with use of the unsupported Mathews charts to understand open stope stability. Comparisons between the relative difference between the predicted condition of unsupported Block A and the observed condition of supported Block A will be used to assess the impact of future adjacent stoping.

Figure 5 shows that the back of an unsupported Block A plots in the unstable zone, and the vertical walls plot in the transition zone between stable and unstable zones. The stress factor A for the Back and the Sidewalls suggests that these stope surfaces will be subject to relaxation when the span / width ratio exceeds about 2.5 for the Backs, and width / height ratio that exceeds about 2.5 for the Sidewalls.

Definitions of stable, unstable, and major failure are shown on the design charts, (Stewart and Forsyth, 1995). Stope surfaces plotting in the stable zone would generally be considered stable in an unsupported condition over the long term. Surface plotting in the unstable zone would typically be supported with regular cable bolts to ensure long term stability. Surfaces plotting in the major failure zone would exhibit larger scale failures that may eventually reach a stable configuration. Surfaces plotting in the caving zone could exhibit an uncontrollable caving situation that could propagate to surface under certain conditions.

Given the state of overall stability in Block A and the observed local ground support loading (untwisting cables), the predicted Unstable condition of the stope backs appears suitable. This provides confidence that the input parameters chosen and the empirical guidance on anticipated open stope stability are reasonable and can be used to comment on the feasibility of excavating stopes larger than Block A. Such relaxation increases the probability of localised wedge movement and unravelling around cable bolts. This failure mode is offset in Block A by the step in the stope wall on the 812 level and the presence of the cable bolts.

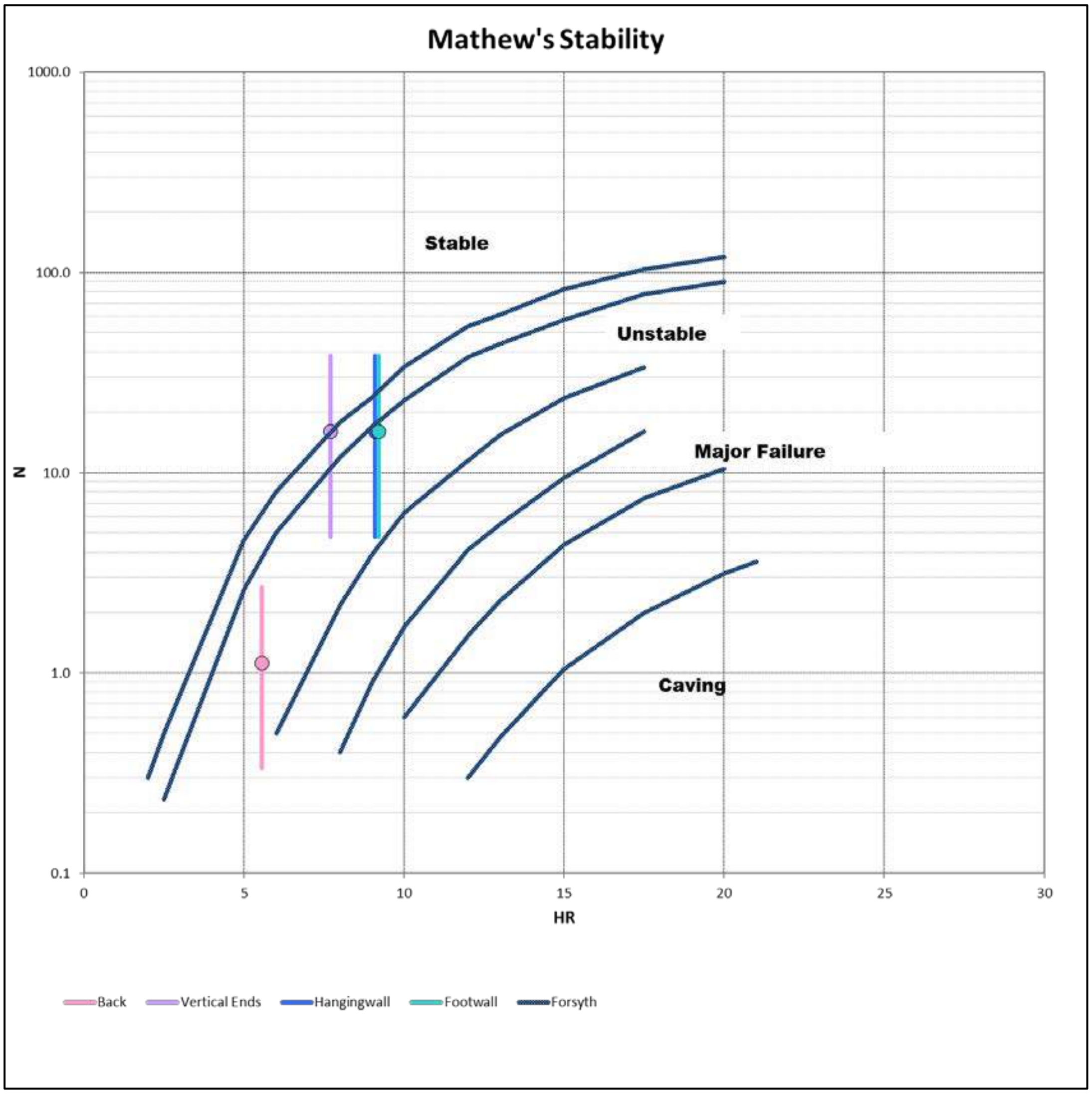

Figure 5: Plot of Block A on the Mathews Stability Graph (Assumes Unsupported Stope Walls)

5.1.2 Potvin's Method

The modified stability number inputs for the Potvin method are summarized in Table 3 and plotted on the empirical stope stability guidance chart for the back, hangingwall/footwall and vertical end in Figure 6.

Table 3: Summary of Potvin's Modified Stability Number Inputs for Block A

	Q'			A	B	C	HR	N'		
	Low	Average	High					Low	Average	High
Back	1.5	5	12	0.45	0.3	2	5.6	0.4	1.3	3.2
Vertical End	1.5	5	12	1.00	0.5	8	7.7	6	20	48
Hangingwall/Footwall	1.5	5	12	1.00	0.5	8	9.1	6	20	48

The various regions of the Potvin chart provide empirical guidance on whether stopes can be expected to be stable or caving when supported with cable bolts.

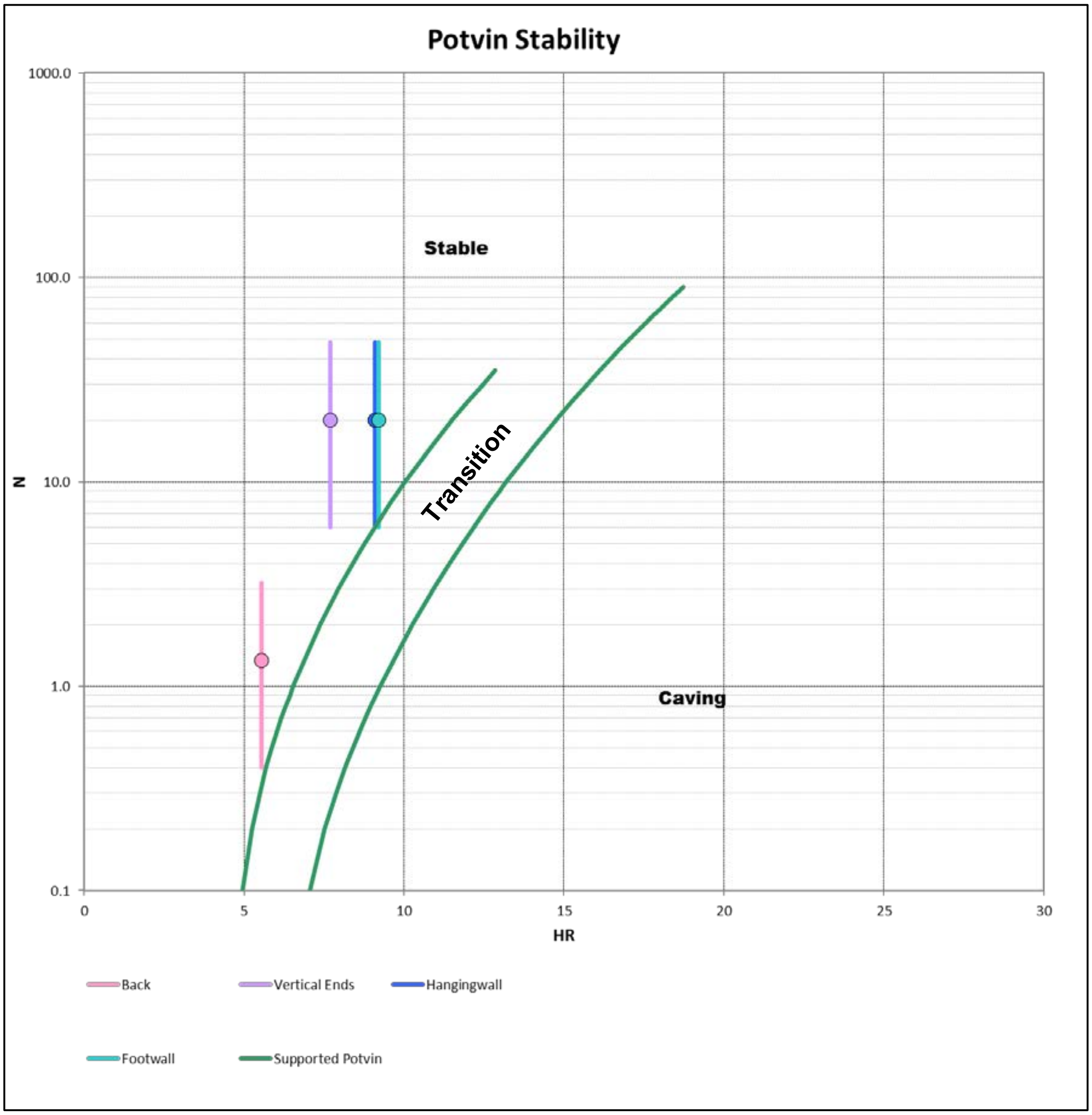

Figure 6: Plot of Block A on the Potvin Stability Graph (Assumes Supported Stope Walls)

The analysis suggests that for Block A, all stope walls can be expected to be stable, but localised regions of reduced rock mass quality plot in the transition zone between stable and caving and can be expected to show some signs of distress. This is consistent with the observed performance of the walls of Block A and the cable bolt support installed, adding confidence to the application of the input values and the overall approach to use in this assessment.

A predictive Potvin analysis was carried out for the scenario of Big Block B (see Figure 1). The modified stability number inputs for the Potvin method for excavation of Big Block B are summarized in Table 4 and plotted on the empirical stope stability guidance chart for the back, sidewalls and end walls in Figure 10.

Table 4: Summary of Potvin's Modified Stability Number Inputs for Block A

	Q'			A	B	C	HR	N'		
	Low	Average	High					Low	Average	High
Back	1.5	5	12	0.35	0.3	2	5.6	0.3	1.1	2.5
Vertical End	1.5	5	12	1.00	0.5	8	7.7	6	20	48
Hangingwall/Footwall	1.5	5	12	1.00	0.5	8	9.1	6	20	48

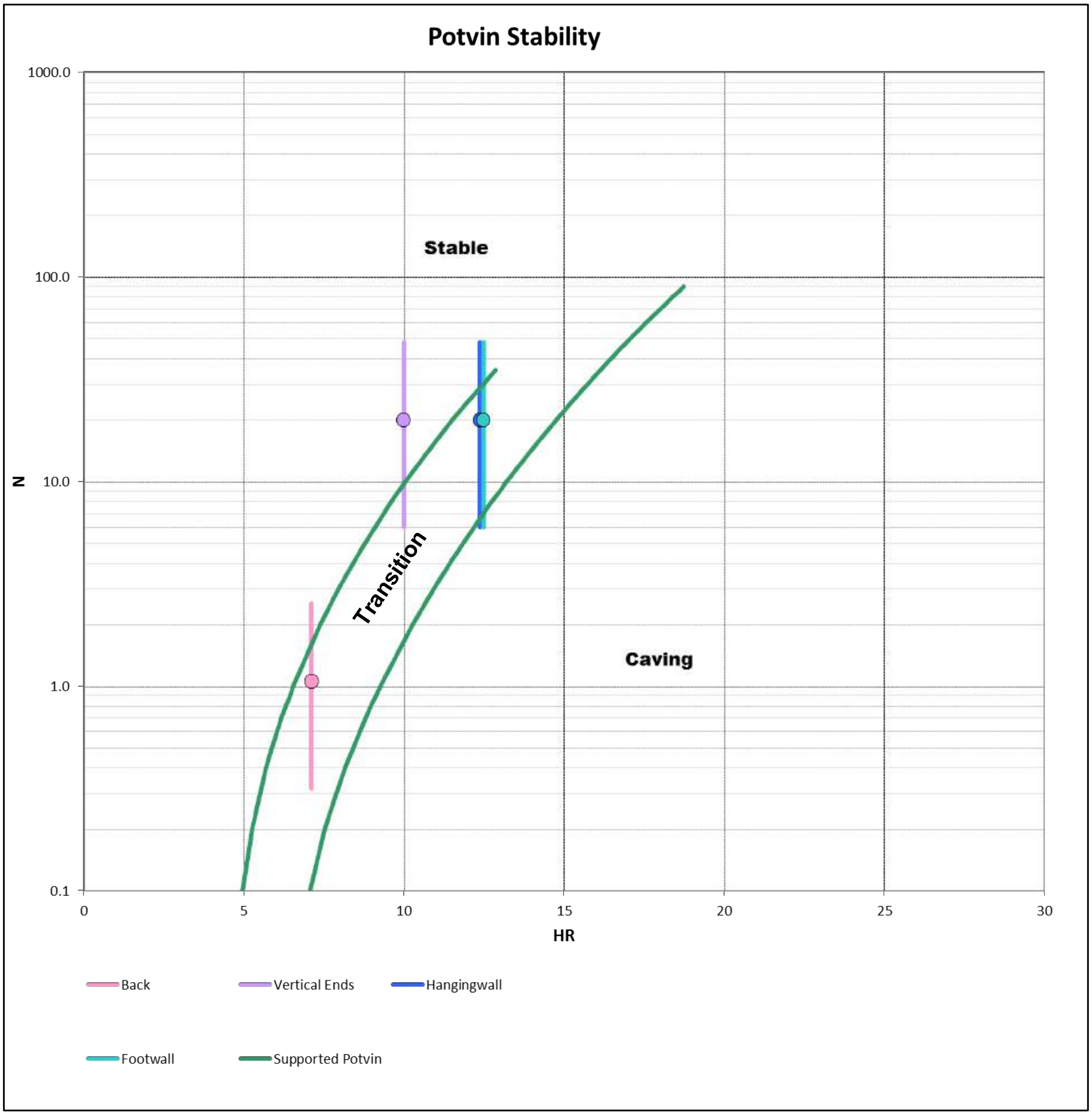

Figure 7: Plot of Block B and C Mined Together on the Potvin Stability Graph (Assumes Supported Stope Walls)

The analysis predicts that both the back and the sidewalls (hangingwall and footwall) plot in the transition zone between stable and caving for cable bolted stopes. Mining Big Block B would likely result in significantly worse ground conditions than observed in Block A or if Block B and C were mined separately. Creating an opening that plots in the transition zone could lead to significant dilution, production interruptions, loss of ore and generally localized failure.

6.0 DISCUSSION AND SUMMARY

It is understood that the mine would like to increase the size of the originally planned stopes to reduce the number of backfill cycles. The current opening, Block A, was stable at the time of the visit, but shows signs that the ground support may be taking load. The empirical stability assessment of the supported Block A shown in Figure 9 reflects the observation that it is stable and suitably supported, providing some confidence in the approach. The assessment suggests that the stope could be slightly enlarged. Figure 10 indicates that mining a stope with dimensions similar to Big Block B could lead to difficult mining conditions caused by sidewall and back instability.

The empirical relationships presented by Mathews and Potvin plots are representative of the behavior of the rock walls of a single stope, in isolation (e.g., a primary stope or Block A). The behavior of the rock walls once the secondary stope (e.g., Block B) is mined are not well represented by these methods, even if backfilling is used in the primary stope. Once mined, the rock walls of the secondary stope will initially behave as if the backfill is not there and may start to fail. However, the failure will likely be partly confined by the backfill in the primary stope, reducing the impact of any rock wall failure. The point to remember is that when a new stope is excavated adjacent to a previously mined, tight backfilled stope, conditions in the second stope will be relatively worse.

Therefore, it is recommended that the future stopes be supported as previously recommended and have maximum dimensions similar to Block A. Back analysis suggests that a bigger stope could be opened, in isolation. However, future voids will be next to existing stopes and the side walls of these voids will possibly behave worse than the back analysis suggest increasing the risk of dilution and production difficulties.

Also, the empirical stability assessments applied do not represent not rigorous engineering. They represent a manner in which experience gained during early stoping at can be applied to understand the behaviour of future mining. Until Mount Polley determines their maximum achievable stope size in practice, the empirical stability assessments should only be thought of as rough guidance.

Tight filling is recommended for back stability. If tight filling is not employed then the back of the stope will get progressively bigger as additional stopes are extracted which increases the likelihood of a large back failure. The impacts of such a failure could be limited to the immediate back of the stope in the form of large loose which could potentially fall on equipment working in the stope, or the failure could also destabilize the surrounding infrastructure including the 842 level, upper ramp and ventilation access, and possibly propagate to surface.

The potential ultimate height of a back failure zone will depend on the bulking factor of the rock and the vertical height of the unfilled void. Mount Polley should consider risks associated with back failure to surrounding surface and underground infrastructure such as ventilation raises and accesses ramps, if the stopes are not tight filled.

The ground observed in the Boundary zone is heavily jointed and it is believed that this will be the governing factor in stope stability. Cablebolting will be useful to help tie the discrete blocks formed by the joints together improving stability, as is shown in Block A. However, if the blocks height were to increase, the chance of relaxation in the vertical walls due to a lack of confinement increases, which increases the risk of failure around the cablebolts.

Finally, the Zuke fault will have an increasing influence on stope stability. Projections of the fault show that it may intercept the remaining stoping blocks between the 782 and 812 Levels, possibly complicating efforts to support stope walls near it. If failure does occur around this geological feature it is possible that the failure will precipitate up around the upper workings.

7.0 CLOSURE

We trust that this memorandum meets your current needs regarding the stability of the Block A void. If you have any questions or comments, please do not hesitate to contact us.

GOLDER ASSOCIATES LTD.

Donald Tolfree, P.Eng.
Senior Mining Engineer

Darren Kennard, P.Eng.
Associate, Geotechnical Engineer

DJT/DTK/jc/ls/jc/bb

Attachment 1: Mount Polley Underground Stope Stability Review

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Boundary Zone – July 30, 2015

Mount Polley Underground Stope Stability Review

Golder Associates

Summary of Site Visit

- Reviewed the following reports to determine rock mass quality parameters (RMR and Q)
  - Geotechnical recommendations – Underground (PB Hughes and associates April 16, 2010)
  - Minimum Ground Support Recommendations (PB Hughes and associates August 21, 2015)
  - Annual Inspection Report – Review of Ground Control Aspects of Underground Mining

- Discussed the mine plan with the Mount Polley Staff
  - Reviewed the DXFs of the proposed stopes

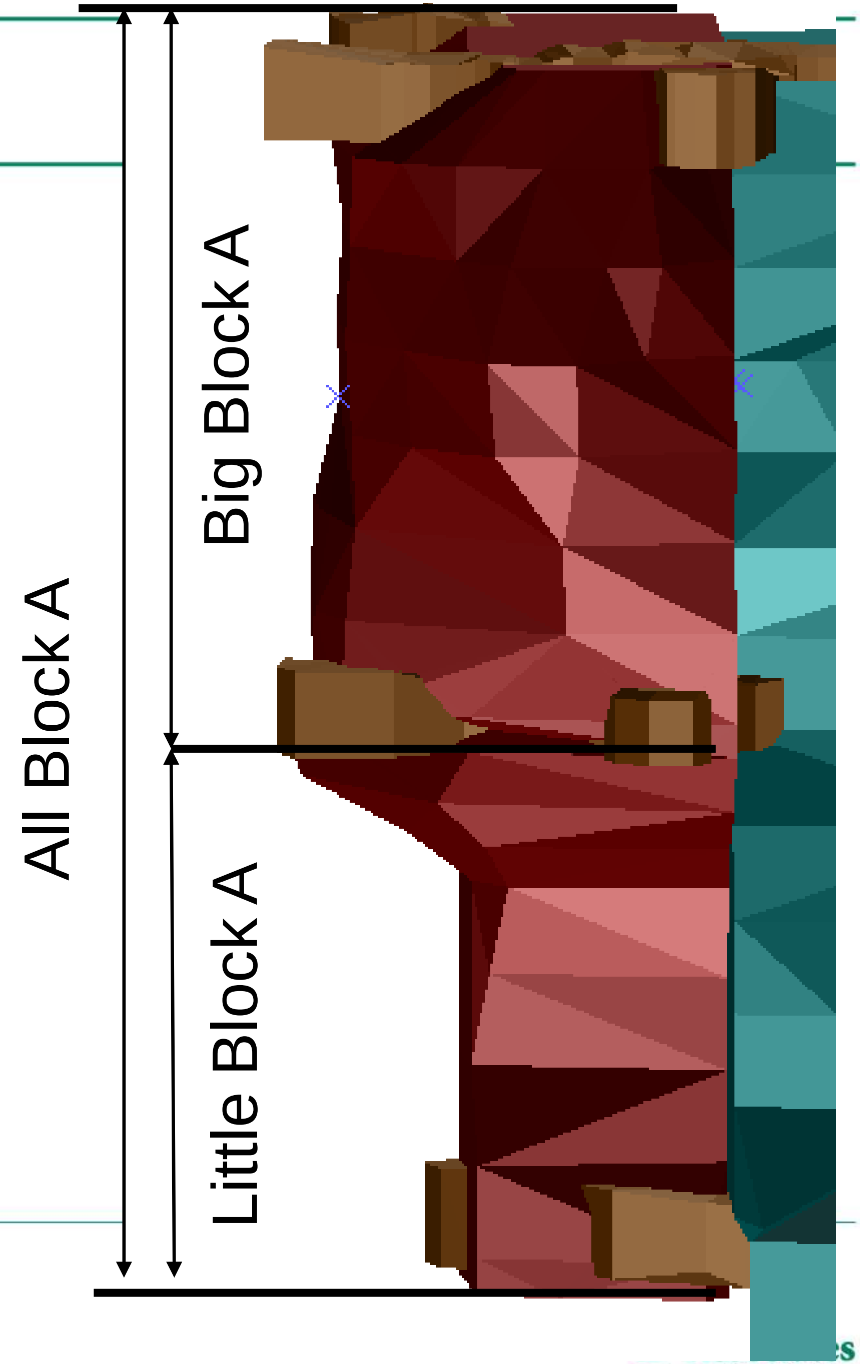
- Visited the following areas:
  - 752, 782, 812, 842 mining levels
  - Open Block A stope

Review – Results

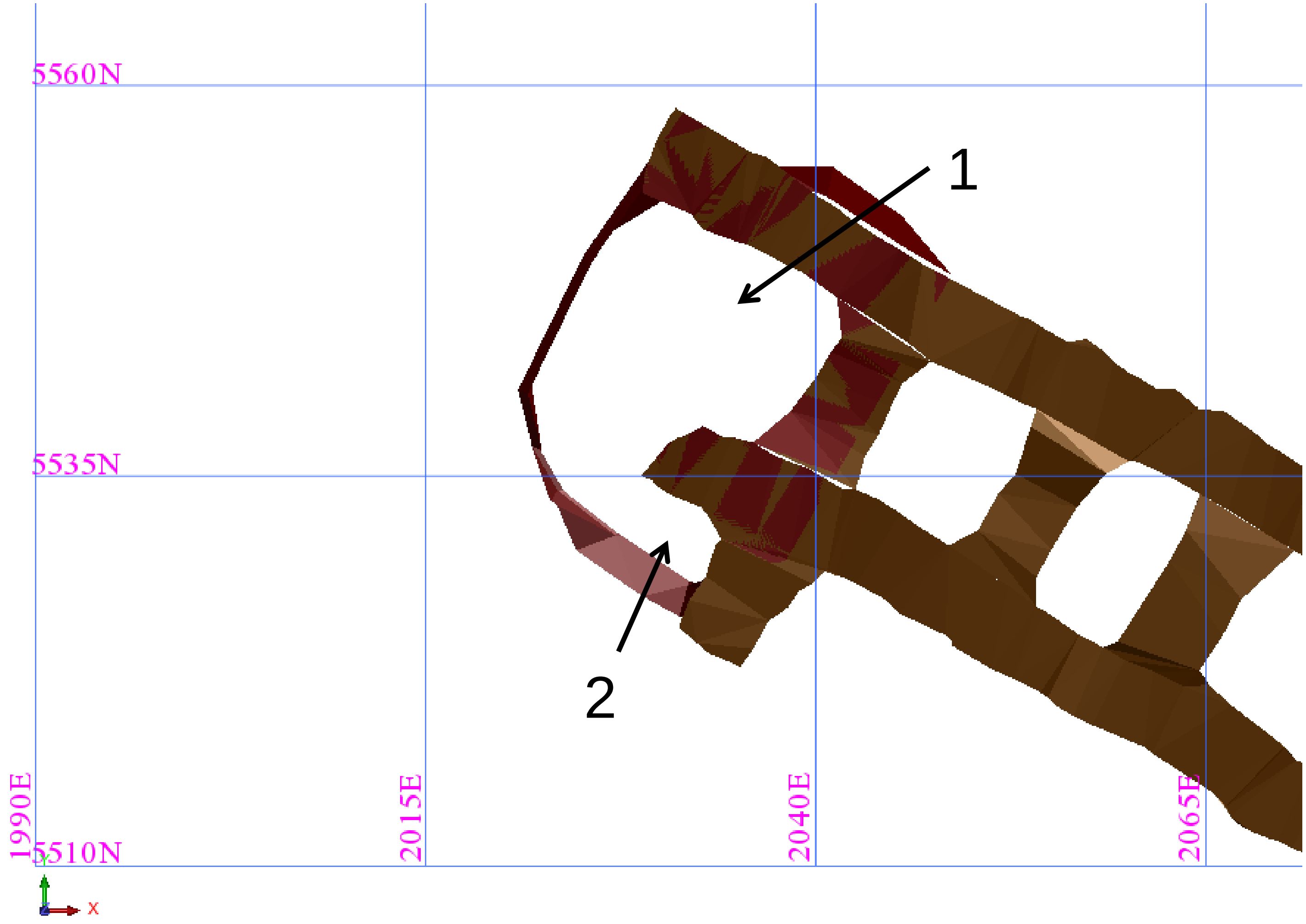
- Rock strength
  - Based on logging by Mount Polley geologist
  - Ranged from R1 to R5
- At least four joint sets
- Variable RQD, majority of estimates in the 25-50 range
  - Based on early borehole mapping for the adit
- Q' range from a high of 2.43 to a low of 0.07 with an average of 0.64
- RMR range from Poor to Good with most estimates in Fair

Stope Dimensions

- Big Block A
  - Width 24m
  - Length 20m
  - Height 35m
- Little Block A
  - Width 15m
  - Length 15m
  - Height 30m
- All Block A
  - Width 24m
  - Length 20m
  - Height 65m



Picture Locations



Stope Back

South Wall

North Wall

Underground Mapping

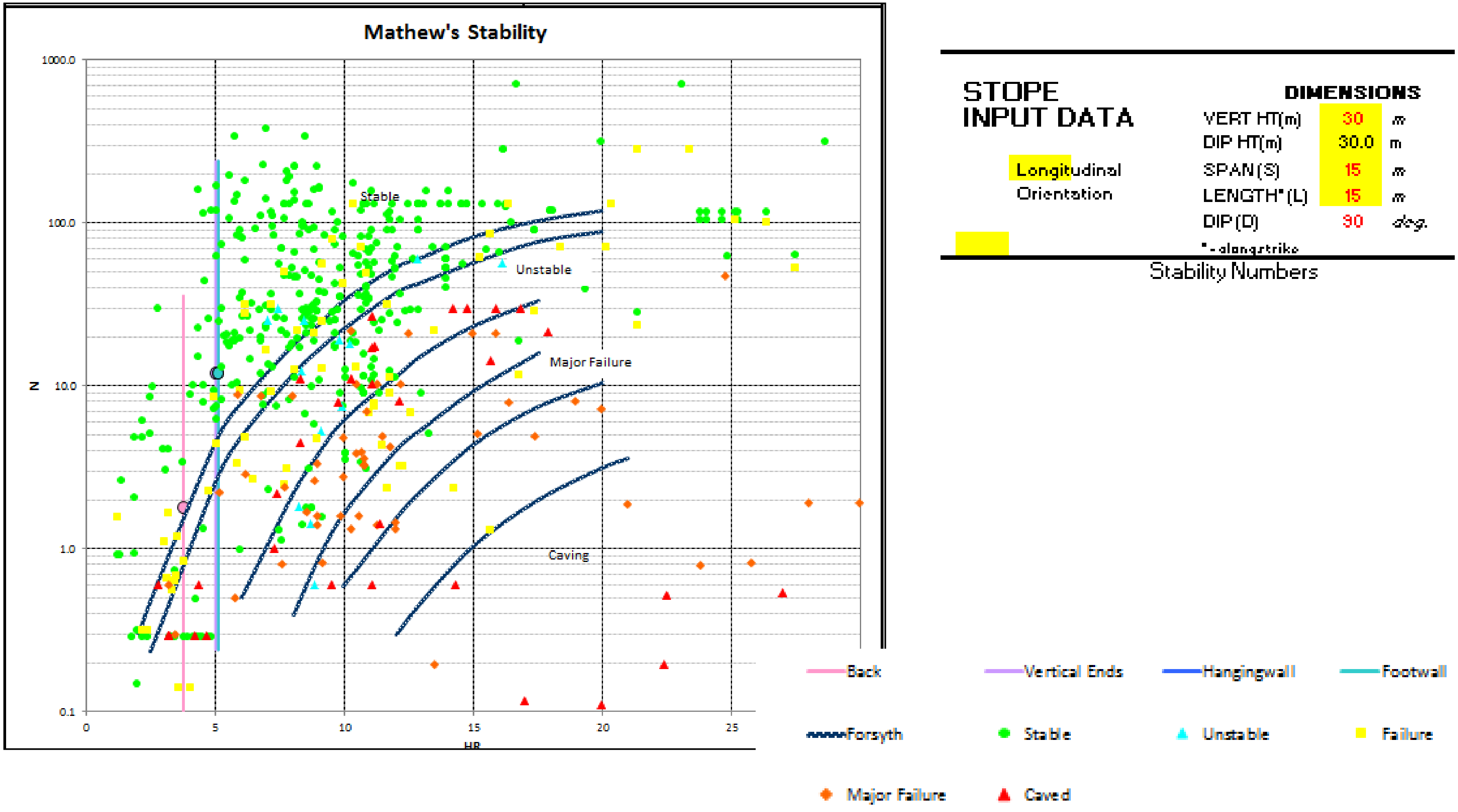
- Confirmation mapping was completed on 752 and 812 levels
  - Agreed with RQD and Joint number estimates

Preliminary Mathews Analysis

- Hydraulic Radius
  - Area / Perimeter (stope size)

- Stability Number (N)
  - Q’ x A x B x C
  - Q’ modified from Barton’s Q tunneling index
    - Q’ = RQD / Jn * Jr / Ja
  - A Stress factor
  - B Joint factor
  - C Stope orientation factor

Little Block A



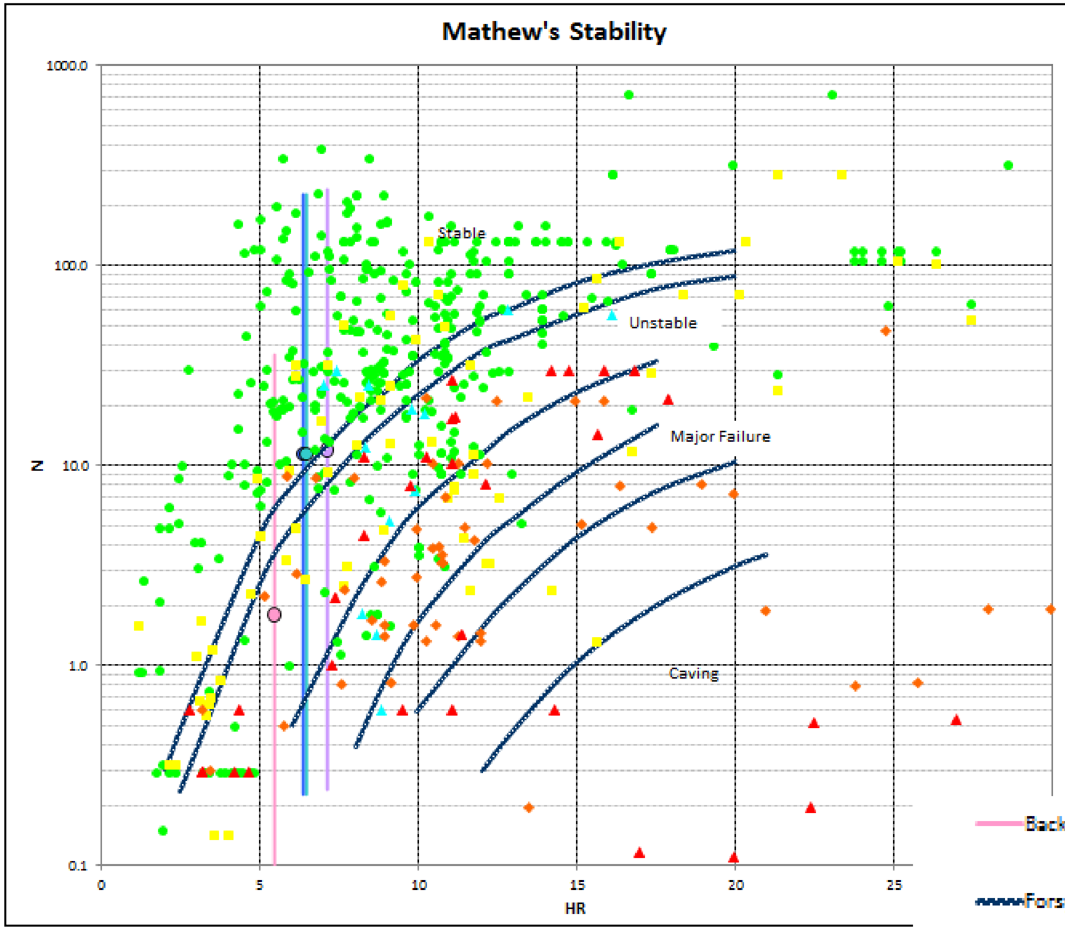
STOPE INPUT DATA		DIMENSIONS		
	VERT HT(m)	30	m
Longitudinal Orientation	DIP HT(m)	30.0	m
	SPAN (S)	15	m
	LENGTH* (L)	15	m
	DIP (D)	90	deg.

*- along strike

Stability Numbers

Golder Associates

Big Block A



STOPE INPUT DATA

Longitudinal Orientation

DIMENSIONS		
VERT HT(m)	35	m
DIP HT(m)	35.0	m
SPAN(S)	24	m
LENGTH*(L)	20	m
DIP(D)	90	deg.

*-along strike

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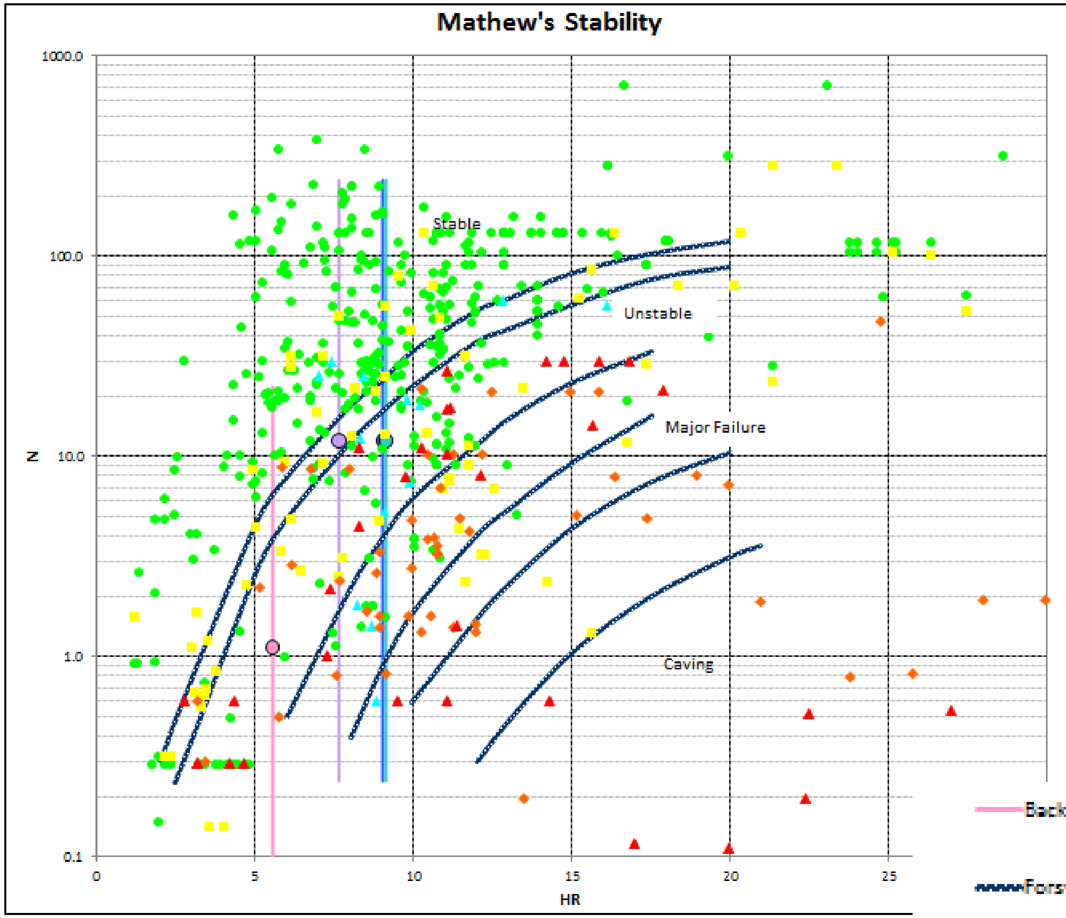
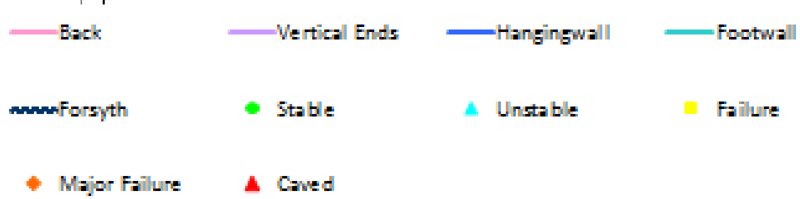
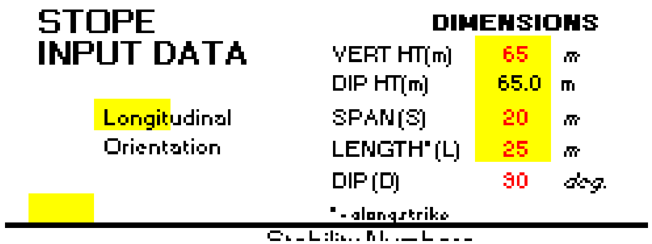
All Block A







Vertical walls may be in tension

Golder Associates

Comment

- Mathews method is an empirically based analysis
  - Lines plotted don’t include support
  - Cables are providing support

- A Factor governed by insitu stresses, stope sizes, and rock strength
  - Estimates for these have been used and may change the results notably the tension in the vertical walls