AN EVALUATION PROCEDURE FOR NEW DEPOSITS IN BROWNFIELD UNDERGROUND MINES

by

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Abstract

An evaluation procedure for newly discovered deposits at brownfield underground mines has been developed and was tested with a case study. A review of current underground mining methods, rock mass classification systems, mining method selection tools and production rates was undertaken to determine the current state of knowledge. An evaluation procedure was then developed to guide on site engineers in selecting feasible underground mining method(s) and identifying a preferred method(s) based on economic analysis.

The procedure was calibrated with information from the literature review and then tested with a case study of the Bonanza Ledge deposit, recently mined by Barkerville Gold Mines, an exploration and mining company based in Wells, British Columbia. The procedure was then applied to the BC Vein deposit, currently under development by Barkerville Gold Mines. Application of the procedure to the BC Vein deposit allowed estimation of productivity rates and cashflows for technically feasible mining methods. Based on analysis of productivity rates and cashflows, a recommended mining method was identified. This case application has demonstrated the effectiveness of the developed procedure for evaluation of brownfield projects.

Acknowledgements

I would like to thank Dr. Euler De Souza of the Robert M. Buchan Department of Mining at Queen's University for guiding me through this process. I would also like to express my sincerest gratitude to my colleagues and supervisors who have helped develop my career through the years. In particular, I want to thank Paul Ratte, General Manager of Barkerville Gold Mines Ltd., for his support in allowing access to data for the Bonanza Ledge and BC Vein deposits.

And finally, I could not have completed this thesis without the loving support of my family. Firstly, my parents and sister, for being my sounding board during the times when I was struggling with my studies. And most importantly, my wife and children for being with me every step of the way. You're my motivation to keep pushing through even when times are tough.

Table of Contents

List of Figures

List of Tables

Chapter 1

Introduction

1.1 Background

A potential mining project must go through several stages of evaluation before it reaches production. These include mineral prospecting and exploration, scoping assessment, feasibility assessment, detailed design, development and finally production. The completion of each stage typically results in the decision to invest additional financial resources to proceed to the next phase.

Scoping identifies possible mining methods, tonnages, grade and operating costs for a given deposit which can lead to the justification of a more detail feasibility study.

A feasibility study of a mining project provides a high-level engineering and economic assessment of the viability of the project. The study is a formal procedure for assessing various factors that directly or indirectly affect the project. Once the factors deemed relative to the project have been defined and studied, as many variables as possible are quantified to determine the potential value of the project (Hartman, 1992). Feasibility studies must contain sufficient information to allow a decision to proceed with the detailed design and development of the project. The feasibility study must provide justification for senior management to commit funds and other resources for the detailed project design, construction, and operation/closure of the project.

During the development of a feasibility study, key risks must be identified and addressed across several areas in order to determine the validity of the project. These areas include:

- geology,
- metallurgy,
- engineering,
- social/environmental permitting,
- project economics,

- financing.

The completion of a feasibility study that leads to a decision to proceed with the project enables other parallel activities to commence. Parallel activities could include detailed mine design, finalization and submission of permit applications, and development of site-specific systems.

Detailed engineering includes completion of detailed designs based on the project scope and conceptual designs approved in the feasibility study. This then leads into the Engineering, Procurement and Construction Management (EPCM) step in which issuing of “for construction” designs, delivery of construction and equipment specifications and scope of work packages for contract documents must be undertaken.

To bring a potential mine project to production, there are several steps that must be carried out to optimize profitability. Before a project can commence, detailed geological, economic and sustainability studies must be documented. Using this initial information, a feasibility assessment can be conducted, and a decision can be made on the technical, financial and social validity of the project. An important component of a feasibility study is identification of the mining method most likely to maximize financial returns within the conditions posed by mine specific constraints.

Often, given the detailed level of analysis required for a full feasibility study, existing operations may lack the technical resources to complete such studies in-house and will outsource the work to consultants. Prior to the work being outsourced, justification for the cost of consulting services must be made by in-house staff. This procedure is intended to provide a scoping assessment of a deposit that, when appropriate, would allow the development of a business case to support funding of a feasibility study. These initial steps of the assessment will also help focus the detailed feasibility study by eliminating mining methods that are not feasible due to technical, or regulatory constraints.

A brownfield expansion involves an investment that adds incremental mineral resources to an existing mine and making use of existing site resources and infrastructure. This can reduce the significance of total capital expenditures as many of capital costs have already been occurred in support of existing operations.

Brownfield projects may be subject to previous constraints from the initial project such as permit conditions or previous mining activities limiting options on future activities.

Once a mine has gone into production, additional geotechnical and geometric data is generated, both as a result of continuing exploration work and the development of the orebody. This new information may challenge some of the assumptions used in the initial mine design, requiring a refinement of the initial mining method(s). Also, additional ore bodies within the vicinity of the initial mine may be assessed for development at a later stage in the mine life.

Smaller mines generally do not have a large technical team with capacity to look at planning beyond the current phases of mine development and production. There is a need for a simplified evaluation procedure to allow on site technical staff to objectively evaluate, on an ongoing basis, the current mine operation as well as to evaluate different options for the development of future deposits within the vicinity of the mine. There exists sufficient, relevant research to allow the development of a simplified procedure for assessing the performance of existing mining operations and for determining the technical and financial feasibility of developing adjacent deposits. Expansion of mining operations into adjacent deposits while making use of existing surface infrastructure is considered to be “brownfield development”.

This thesis uses existing evidence-based research to develop such a simplified evaluation procedure. The procedure is tested against a case study of the recently completed Bonanza Ledge Mine (Phase 1). Following testing, the procedure is then used to assess the technical and economic feasibility of mining the BC Vein Deposit (Phase 2).

1.2 Objectives

The objective of this thesis is to develop an evaluation procedure for a brownfield underground deposit, which can be used by technical staff to:

- Identify technically feasible mining methods based on consideration of orebody characteristics,
- Estimate productivity rates and cashflows associated with technically feasible mining methods, and
- Recommend technically feasible mining method(s) for the detailed feasibility assessment.

1.3 Scope

The scope of the thesis includes a literature-based review of:

- underground mining methods (Section 2.1),
- rock mass classification systems (Section 2.2),
- empirical mining method selection and design tools (Sections 2.3 and 2.4),
- production capacity and cost estimates (Section 2.5).

The results of the literature review were used to develop a procedure to identify technically suitable mining methods. Mining methods determined to be technically feasible were then analyzed to estimate a range of anticipated production rates, grade and mining costs. Revenues are calculated based production rates, grade and market price. The results of the procedure are used to generate projected annualized cashflows.

Once the procedure was developed, it was tested on data from the recently mined Bonanza Ledge (Phase 1) Deposit. The results from the procedure were compared to the actuals from the mine to test the predictability of the procedure.

Following the testing and satisfaction of the predicted results, the procedure was applied to the BC Vein (Phase 2) Deposit to identify the most suitable mining method and to estimate likely range of production rates and cashflows.

Phase 2 is considered a Brownfield Development as the majority of required support infrastructure is in place as a result of the Phase 1 development.

Chapter 2

Literature Review

2.1 Underground Mining Methods

Once an ore body has been defined, the process of selecting technically feasible method(s) of mining can begin (Hustrulid et al., 2001). Underground mining methods can be divided into three general categories (Figure 1):

- Pillar Supported - methods that use rock pillars to provide overall ground stability.
- Artificially Supported – methods that rely on the use of fill to provide support to the surrounding rock mass during mining.
- Unsupported/Caving- methods in which caving is induced as part of the mining process.

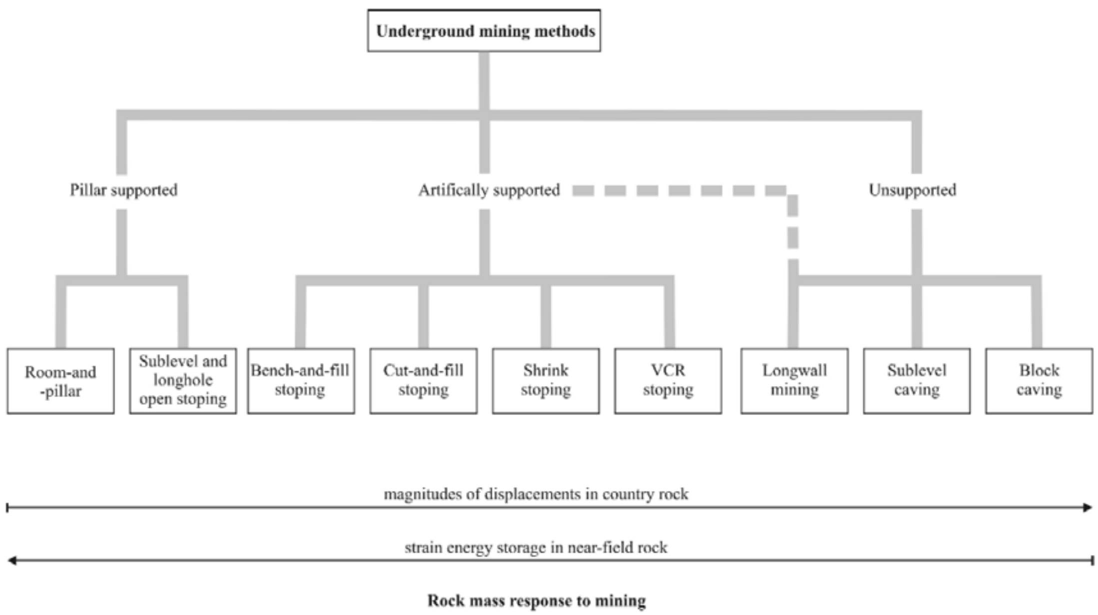

Figure 1: Underground Mining Methods (Brady and Brown, 2006)

2.1.1 Pillar Supported Methods

2.1.1.1 Room and Pillar

In Room and Pillar mining methods, horizontal stopes called “rooms” are extracted along the ore body with pillars being left in place to support the hanging wall. Room and pillar mining methods are generally used in flat or shallow dipping ore bodies of limited thickness and with competent hanging walls. The hanging wall needs to be competent in order to maximize the span between pillars (i.e. the size of the rooms) and the overall extraction of ore.

Pillars are usually left in regular patterns to evenly distribute the load of the hanging wall amongst the pillars. Pillars can be left in square or elongated shapes depending on the stress loading experienced during the life of the mine. As pillars are left behind for support, they are not generally considered part of the mine’s reserves.

There are three main types of room and pillar mining:

- Classic Room and Pillar
- Post Room and Pillar (Post-Pillar)
- Step Room and Pillar

2.1.1.1.1 Classic Room and Pillar

The Classic Room and Pillar method is normally used in flat-bedded deposits of medium to large thickness (>3m). Horizontal stopes (rooms) are mined, starting at the top and are benched down in steps, as illustrated in Figure 2. Some narrower deposits can be mined in a single horizontal slice.

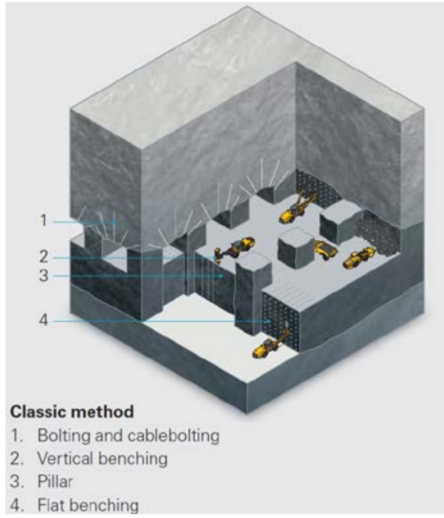

Figure 2: Classic Room and Pillar Mining Method (Atlas Copco, 2014)

2.1.1.1.2 Post Room and Pillar (Post-pillar)

The Post Room and Pillar method combines the techniques of cut and fill and classic room and pillar methods to mine inclined ore bodies dipping between 20°-55° with a larger vertical height (>3m). Mining begins from the bottom of the ore body, removing a horizontal slice while leaving pillars behind to support the roof (Figure 3). The rooms are then backfilled with hydraulic tailings or waste rock at which point mining of the next horizontal slice can begin on top of the fill. Pillars are extended through the fill, which provides confinement for the pillars, allowing for a greater bearing capacity. This allows for a higher recovery compared to the classic room and pillar method.

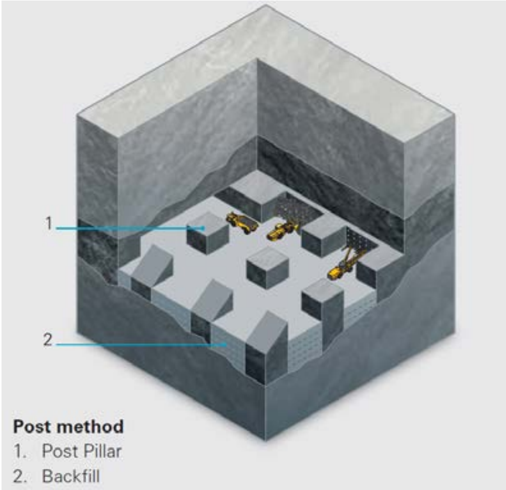

Figure 3: Post Room and Pillar (Post-Pillar) Mining Method (Atlas Copco, 2014)

2.1.1.1.3 Step Method

The Step Room and Pillar method is used in ore bodies with dips of 15°–30° and a thickness of 2-5m. Travel ways are angled across the dip of the ore body at grades useable by wheeled equipment (Figure 4). Drifts are then developed along the strike of the ore body from one travel way until breaking into another travel way. Once the initial strike drift is completed, another drift is excavated one step below and adjacent to the first drift. Elongated pillars are left between the drifts to support the roof. Mining continues downward until the area has been mined out.

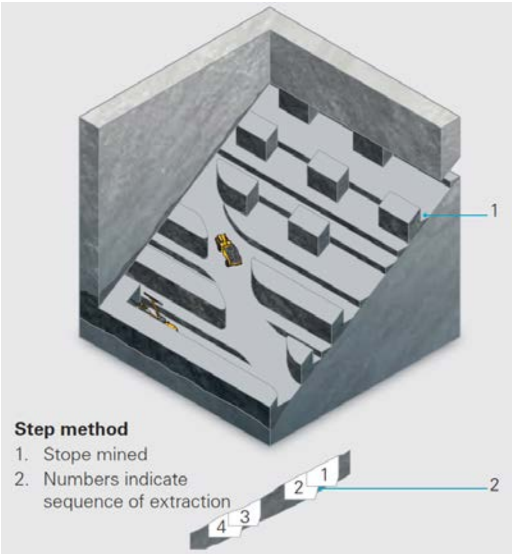

Figure 4: Step Room and Pillar Mining Method (Atlas Copco, 2014)

2.1.1.2 Sublevel Open Stoping

Sublevel Open Stoping (generally referred to as sublevel stoping) is used for mining large orebodies with ~~a~~ steep dips, uniform shape and distinct contacts (Figure 5). Both the hanging wall and footwall need to be of competent rock to minimize dilution and the ore body ideally of higher rock mass quality. Large open stopes are extracted by use of blastholes and then stopes are backfilled. This allows for the recovery of ore pillars between backfilled stopes.

Sublevels are excavated within the orebody to allow for long hole drilling of the stopes. Drawpoints are located below the stope to allow safe mucking out by Load-Haul-Dump (LHD) vehicles and for transport to nearby ore passes or into trucks or rail cars for haulage.

Sublevel stoping allows for selective mining and flexibility by enabling multiple mining stopes to be active simultaneously.

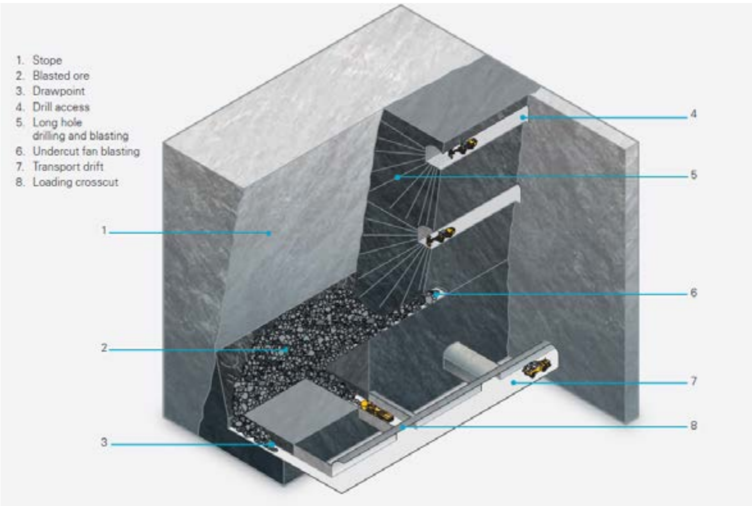

Figure 5: Sublevel Open Stoping (Atlas Copco, 2014)

2.1.1.3 Longhole Stoping

Longhole Stoping is similar to sublevel open stoping but uses longer and larger diameter blast holes. Levels are generally spaced 60m vertically apart (Figure 6) with the Longhole method compared to 30-40m for the Sublevel Open Stoping method. As such, the footwall and hanging wall must be more competent than the rockmass that can be used for sublevel stope mining. Blasted ore is sometimes temporarily left in place to provide support for the walls.

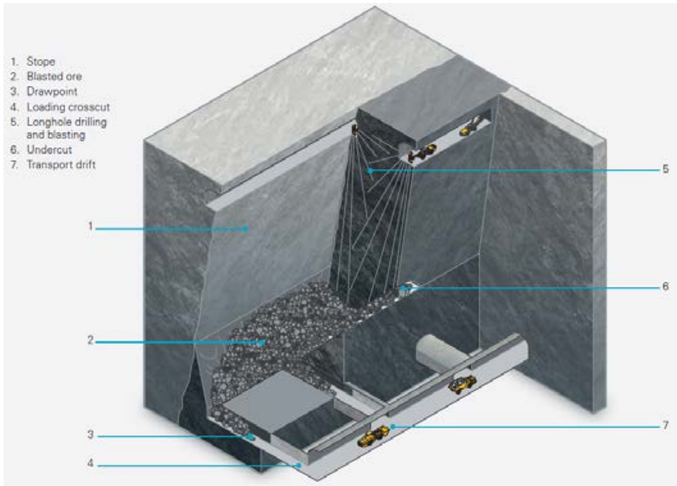

Figure 6: Longhole Stoping (Atlas Copco, 2014)

2.1.2 Artificially Supported Methods

2.1.2.1 Shrinkage Stoping

Shrinkage Stoping is a highly selective and labour intensive mining method (Figure 7). Ore is mined in horizontal slices from the bottom of the stope and advances up. The ore is drawn from the stope until there is enough room to allow access to work on the next horizontal slice. This allows the walls to be supported by the blasted ore until the stope has been completely excavated. At this point, the remaining blasted ore is drawn out from the stope bottom.

Shrinkage stoping is used in orebodies with steep dips, having generally a competent ore rock mass and defined footwall and hangingwall ore boundaries. The ore needs to be resistive to oxidation as the ore must be left in place until the end of mining the stope.

Similar to sublevel open stoping, drawpoints are first developed at the bottom of the stope. A raise is then developed along the dip of the stope to provide access for workers, services and ventilation.

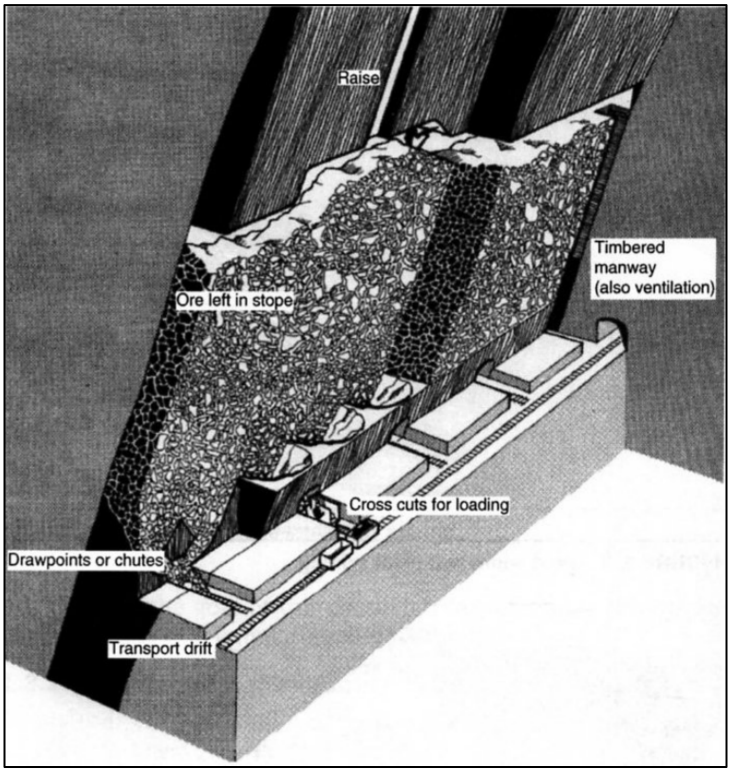

Figure 7: Shrinkage Stoping (Hartman 1982)

2.1.2.2 Cut and Fill

In the Cut and Fill mining method, ore is removed in horizontal slices and these are then backfilled with either waste rock or mill tailings mixed with cement (Figure 8). The fill provides support for the stope walls and depending on the type of Cut and Fill method adopted, provides a stable roof or working platform for the next horizontal stope slice.

Cut and Fill mining can be implemented in two ways:

- Overhand Cut and Fill
- Underhand Cut and Fill

With the underhand cut and fill approach, the ore remains beneath the active stope and the roof consists of backfill. With overhand cut and fill mining, the ore is in the roof and mining takes place on top of the backfill.

Cut and fill mining is used for steeply dipping orebodies in competent rock masses. It is a selective mining method which is ideal for irregular orebody shapes and erratic mineralization. As the method requires significant backfill and support, it is generally more expensive, even when mechanized than for other methods.

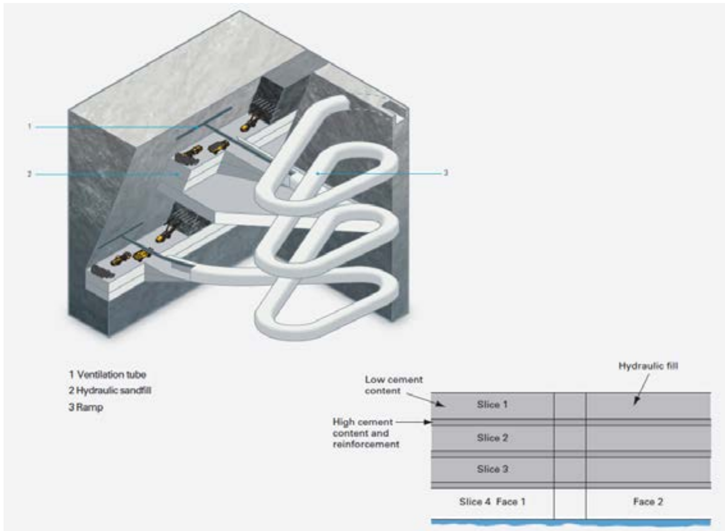

Figure 8: Cut and Fill (Atlas Copco, 2014)

2.1.2.3 Vertical Crater Retreat Stoping

Vertical Crater Retreat (VCR) is a mining method that is used in orebodies with steep dips and a competent rockmass. Blastholes are drilled through a stope from top to bottom and then a horizontal slice is blasted out across the entire width of the stope bottom. Part of the blasted ore is mucked out until there is enough room for the next slice blast above. The blasted ore provides support to the wall during the mining of the stope. Blasting of horizontal slices continues upwards until the stope in mined out. Once mined out, the stope is backfilled with hydraulic fill or cemented backfill to allow for extraction of the adjacent stope (Figure 9).

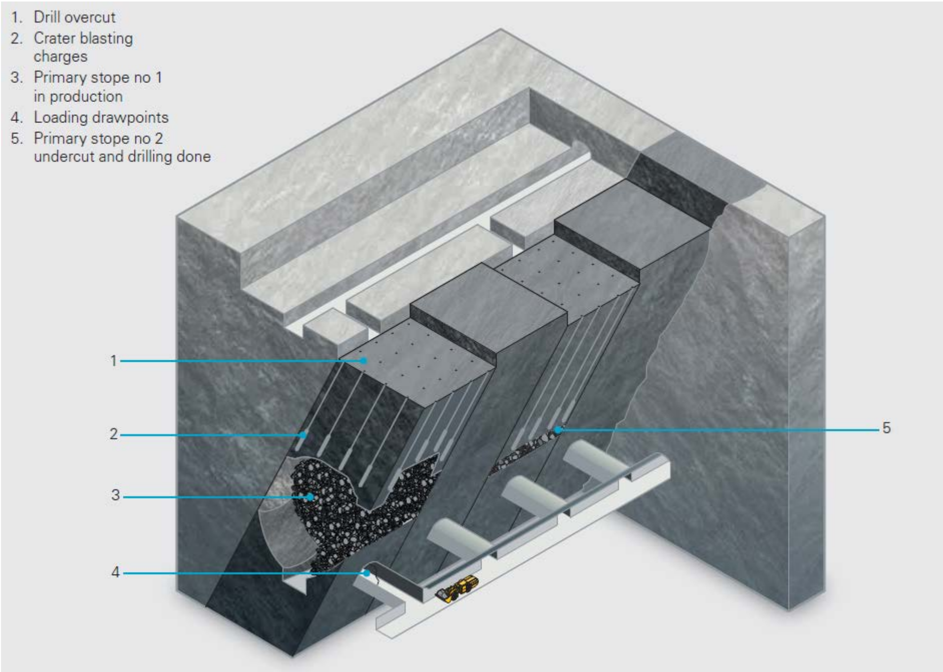

Figure 9: Vertical Crater Retreat (Atlas Copco, 2014)

2.1.3 Unsupported Methods

2.1.3.1 Sublevel Caving

Sublevel Caving is a mining method used in large deposits with a steep dip (Figure 10). The ore generally needs to be competent with few structures and the hanging wall should have some mineralization. Development is placed in the footwall and as such, the footwall needs to be competent to ensure the longevity of the development. The ore should have smaller fragmentation than waste to determine when mucking from a drawpoint should stop.

Sublevel caving requires significant upfront development as several levels need to be established before production can start. Drifts are developed through the orebody at regularly spaced intervals from the footwall to the hanging wall. Once several levels have been established, the drilling of blastholes first occurs on the topmost level to initiate production. Blasting is done in sections above and along the length of the drifts and is mucked out until waste starts to report to the drawpoint at which point, the next section blast is taken. Mining retreats from the hanging wall to the footwall. Once blasting has progressed to a certain point on a level, the level below can start drilling and blasting operations using similar sequencing. As production advances, the hanging wall caves onto the ore, creating a cover. Initial recoveries are low on the topmost levels, but as the interaction between drawpoints increases through production, recoveries increase. Sublevel Caving mines sometimes have a scavenger level at the bottom of the deposit to maximize recovery of the orebody at the end of the mine life.

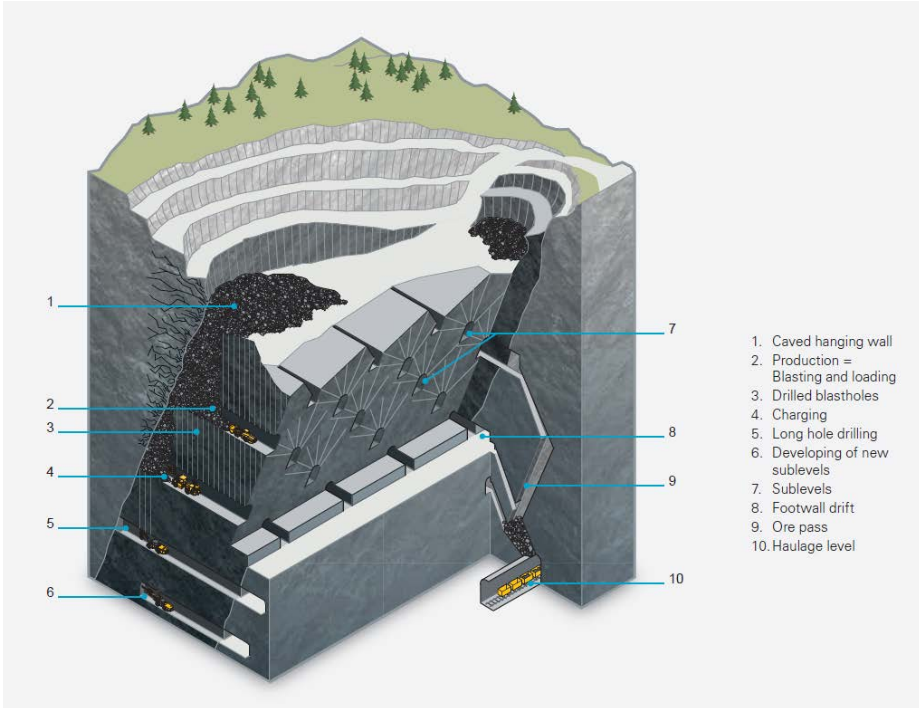

Figure 10: Sublevel Caving (Atlas Copco, 2014)

2.1.3.2 Block Caving

Block Caving is a large-scale mining method that relies on gravity to cave the ore (Figure 11). Orebodies need to be massive in size and consist of weak ore that fractures into managed sizes once sections are undercut. The orebody must be surrounded by competent rock to ensure that the required infrastructure can support the mine life. Given the large footprint of block cave mines, the mine needs to be located in an area in which surface subsidence is permitted.

The general layout of a block cave mine consists of three levels:

- Undercut
- Extraction
- Haulage

The undercut level is developed within the ore and is used to initiate the cave. Drawbells connecting the undercut and extraction levels are blasted in order to provide a drawpoint for the ore to flow down into.

Once the cave has been initiated, a systematic draw from the drawbells occurs on a continuous basis to prevent any large voids from being developed above.

As block caving caters itself to high volumes, it is a low cost, high productivity method with good ore recovery and moderate dilution.

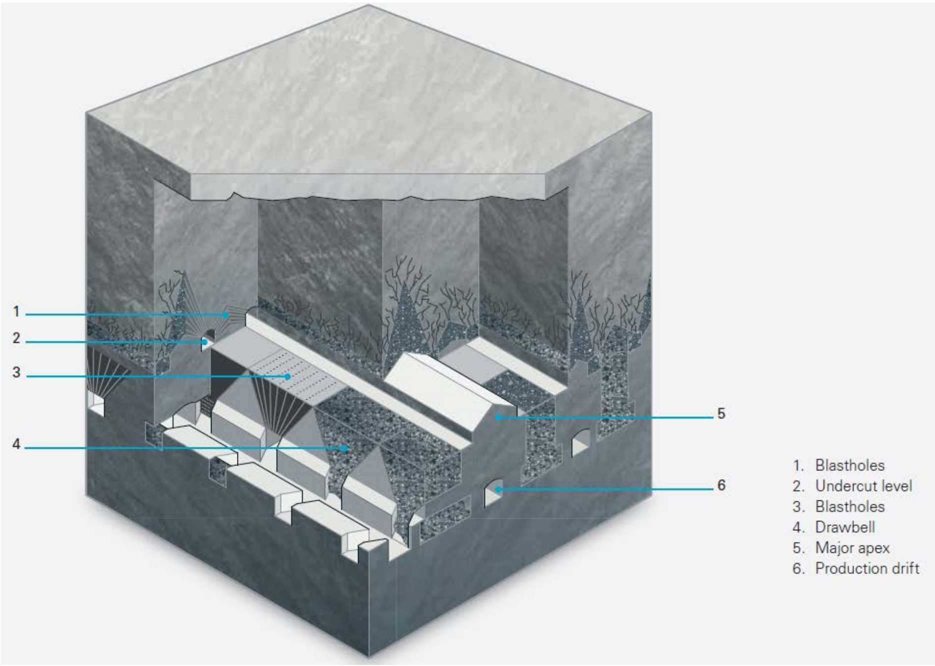

Figure 11: Block Caving (Atlas Copco, 2014)

2.1.3.3 Longwall

Longwall mining is used for narrow and generally flat-lying deposits of uniform thickness and a large horizontal extent. Common types of deposits include coal, potash, or conglomerate reefs. Longwall mining can be used in both hard and soft rock (Figure 12 and Figure 13).

Ore is extracted along a long working face with the area close to the face being supported to allow workers and equipment access. Mined out areas away from the face are allowed to subside and collapse.

Haulage drifts are developed on either side of the mining area which determines the length of the longwall face. A drift is developed between the haulage drifts which creates the working face for the mining area. The ore is then mined in vertical slices, retreating away from the collapsed area. Softer minerals such as coal or potash cater to the use of continuous miners which mechanically scrape the working face to remove the ore. The ore is moved onto a conveyor and out to the haulage drifts. The hanging wall along the longwall face is locally supported by a system of hydraulically operated props. The props are advanced with the coal mining face and the unsupported hanging wall behind the face is permitted to collapse.

Longwall mining is also used for mining narrow, reef-type deposits (Figure 13). Handheld pneumatic drills are used to drill and blast out the reef while pillars made of timber and/or concrete are used to support the reef hanging wall. In this case, the method is highly selective but requires a large, low cost workforce.

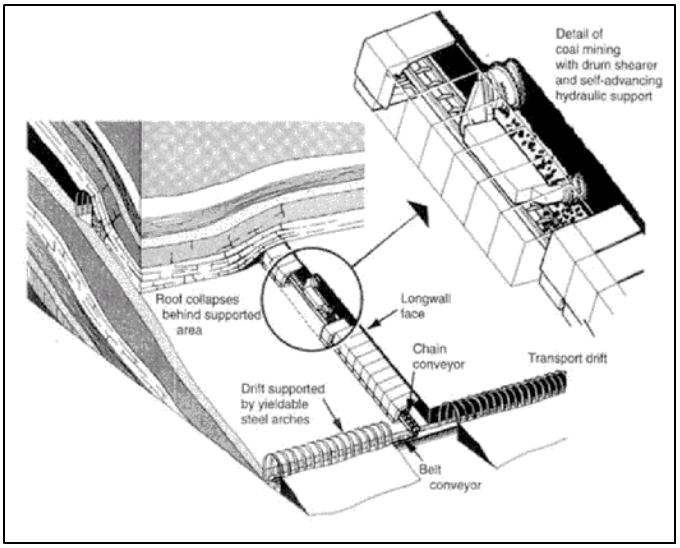

Figure 12: Longwall mining in coal (Hustrulid et. al., 2001)

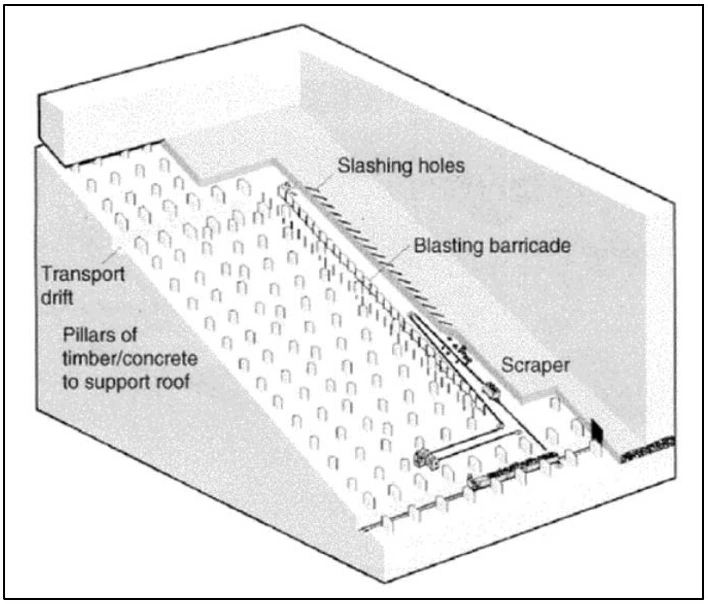

Figure 13: Longwall Mining Gold Reef (Hustrulid et. al., 2001)

2.1.4 Underground Mining Method Characteristics

Table 1 summarizes the typical orebody characteristics that are associated with the various underground mining methods described in the previous sections. At times, more than one mining method may be technically feasible for a given orebody. Other factors must be considered in conjunction with the technical inputs to determine the preferred mining method. Some of these factors include:

- regulatory/environmental constraints,
- production capacity restraints,
- cashflow considerations, etc.

Table 1: Underground Mining Method Characteristics (Hartman, 1992)

	Self-Supporting Methods		Artificially Supported Methods			Caving Methods		
Method	**Room and Pillar**	**Sublevel Stoping**	**Shrinkage**	**Cut and Fill**	**Vertical Crater Retreat**	**Sublevel Caving**	**Block Caving**	**Longwall**
Size	Thickness <90m, large areal extent	6-30m, fairly large extent	1-30m, fairly large extent	2-30m, fairly large extent	6-30m, fairly large extent	Large, extensive vertical or areal extent; thickness >6 m)	Very large areal extent; thickness >30m	Large areal extent, thin bedded (1-5m); uniform thickness
Shape	Tabular, lens	Tabular, lenticular	Tabular, lenticular	Tabular, can be irregular, discontinuous	Tabular, lens	Tabular or massive; may be moderately irregular	Massive or thick tabular, fairly regular	Tabular
Dip	<30°	60-90°	60-90°	>45°	>45°	>60°	Fairly steep (>60°); can be fairly flat if very thick	Low (<12°); preferably flat, uniform
Depth	<900m	<2400m	<2400m	<2400m	<900m	<1200m	>600m	150m - 3500m
Ore Strength	Moderate to strong	Fairly strong to strong	Strong. Should not pack, oxidize	Moderate to strong, may be less competent than with unsupported methods	Moderate to strong	Moderate to fairly strong, requiring blasting	Weak to moderate or even fairly strong, preferably soft or friable, fractured or jointed, not blocky	Any, but should crush rather than yield under roof pressure; preferable material that is weak and can be cut by continuous miner
Rock Strength	Moderate to strong	Moderate to strong	Strong to fairly strong	Weak to fairly weak	Fairly Strong to Strong	Weak to fairly strong; may be blocky, but should be fractured or jointed and cavable	Weak to moderate, similar to ore in characteristics	Weak to moderate, must break and cave
Ore grade	Low to moderate	Moderate	Fairly high	Fairly high	Low to moderate	Moderate	Low, ideal for disseminated ore	Moderate
Ore uniformity	Variable, lean ore or waste left in pillars	Fairly uniform to uniform	Uniform	Moderate, variable	Fairly Uniform	Moderate	Fairly uniform and homogeneous	Uniform
Mining rate (tpd)	1,500-10,000	1,500-20,000	200-800	500-1,500	1,500-20,000	1,500-70,000	10,000-100,000+	1,500-10,000
Relative mining cost	7-20	7-25	20-50	20-70	7-25	5-15	1-2.5	7-20
Recovery	60-80%	60-80%	78-85%	90-100%	60-80%	80-90%	90-125%	70-90%
Dilution	10-20%	10-20%	<10%	5-10%	10-20%	10-35%	10-20%	10-20%

2.2 Rock Mass Classification

Rock mass classification systems have been developed for over 100 years to formalize empirical approaches to excavation designs, in particular for determining support requirements. Rock mass classification systems place a rock mass into groups or classes based on defined characteristics such as intact rock strength, number of fractures within the rock per unit length and condition of fractures (Bieniawski, 1989). As rock masses exhibiting the same classification are expected to behave in a similar manner based upon historic observation and experience, assignment of a classification provides initial information that is useful in the preliminary stages of mine design.

A key parameter used in the identification of feasible mining methods is the rock mass quality of the hanging wall, footwall and orebody which then allows the determination of total ore recovery and dilution, production rates and overall costs.

2.2.1 Terzaghi's Rock Mass Classification

One of the earliest references of rock mass classification for use in tunnel support design was presented by Terzaghi (1946). Terzaghi estimated the loads carried by steel sets based on a descriptive classification of the rock mass. The rock mass classifications used by Terzaghi are as follow:

- *Intact rock contains neither joints nor hair cracks. Hence, if it breaks, it breaks across sound rock. On account of the injury to the rock due to blasting, spalls may drop off the roof several hours or days after blasting. This is known as a spalling condition. Hard, intact rock may also be encountered in the popping condition involving the spontaneous and violent detachment of rock slabs from the sides or roof.*
- *Stratified rock consists of individual strata with little or no resistance against separation along the boundaries between the strata. The strata may or may not be weakened by transverse joints. In such rock the spalling condition is quite common.*

- *Moderately jointed rock contains joints and hair cracks, but the blocks between joints are locally grown together or so intimately interlocked that vertical walls do not require lateral support. In rocks of this type, both spalling and popping conditions may be encountered.*
- *Blocky and seamy rock consists of chemically intact or almost intact rock fragments which are entirely separated from each other and imperfectly interlocked. In such rock, vertical walls may require lateral support.*
- *Crushed but chemically intact rock has the character of crusher run. If most or all of the fragments are as small as fine sand grains and no recementation has taken place, crushed rock below the water table exhibits the properties of a water-bearing sand.*
- *Squeezing rock slowly advances into the tunnel without perceptible volume increase. A prerequisite for squeeze is a high percentage of microscopic and sub-microscopic particles of micaceous minerals or clay minerals with a low swelling capacity.*
- *Swelling rock advances into the tunnel chiefly on account of expansion. The capacity to swell seems to be limited to those rocks that contain clay minerals such as montmorillonite, with a high swelling capacity.*

2.2.2 Deere- RQD

The Rock Quality Designation (RQD) index was developed by Deere (Deere et al, 1967) to quantitatively assess the integrity of the rock mass based on the degree of intactness and physical dimensions of drill core logs. The RQD index can be assessed in the field during the initial core recovery or later at core logging facilities.

The RQD value is defined as a percentage of the total length of intact pieces of core greater or equal to 100mm in length over the total length of the core recovered. Core is to be of at least NW size (54.7mm in diameter), have a minimum total interval length of 2.0 meters and be drilled with a double-tube core barrel. The RQD index equation is as follows:

Equation 1: RQD Index

$$RQD = \frac{\sum(Length\ of\ core\ in\ pieces > 100mm)}{(Total\ length\ of\ core\ recovered)} \times 100\%$$

The results of RQD indexing may be quantitatively related to the structural integrity or the quality of the rock according to Table 2.

Table 2: RQD Rock Quality Condition (Deere, 1967)

Rock Quality Index	**Rock Quality Condition**
<25%	Very Poor
25-50%	Poor
50-75%	Fair
75-90%	Good
90-100%	Very Good

Variation in RQD measurements can occur due to mishandling of retrieved core, orientation of drilling (parallel or perpendicular to foliation) and/or differences in measuring techniques applied. The technique for measuring the RQD index is illustrated in Figure 14.

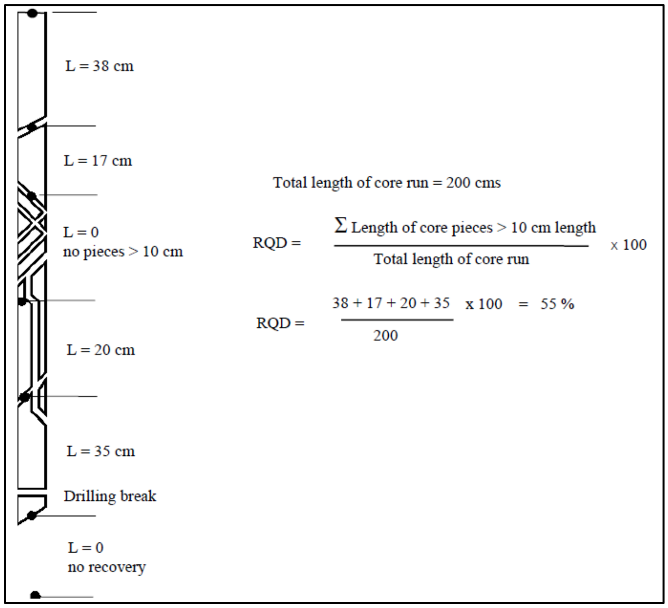

Figure 14: Example of RQD Measurement (Deere, 1967)

2.2.3 Wickham's Rock Structure Rating (RSR)

Wickham et al (1972) developed a classification system for describing the quality of the rock mass and the appropriate support based on their Rock Structure Rating (RSR) classification. Case studies used to develop the system were mostly from small tunnels supported by steel sets, but the system is recognized as making the first reference of shotcrete support in support design considerations.

The RSR system is significant as it introduces the concept of providing a rating to each component to arrive at a numerical value. The formula for the Rock Structure Rating is as follows:

Equation 2: Rock Structural Rating Classification

$$RSR = A + B + C$$

The components of RSR are:

- Parameter A, Geology: General appraisal of geological structure based on:
    - Rock type origin (igneous, metamorphic, sedimentary)
    - Rock hardness (hard, medium, soft, decomposed)

- Geologic structure (massive, slightly faulted/folded, moderately faulted/folded, intensely faulted/folded)

- Parameter B, Geometry. Effect of discontinuity pattern with respect to the direction of the tunnel drive based on:
    - Joint spacing
    - Joint orientation (strike and dip)
    - Direction of the tunnel drive
- Parameter C, Effect of groundwater inflow and joint condition on the basis of:
    - Overall rock mass quality based on A and B combined
    - Joint condition (good, fair, poor)
    - Amount of water inflow (in gallons per minute per 1000 feet of tunnel)

Three tables (Table 3, Table 4 and Table 5) are used to evaluate the rating of each parameter for determining the RSR value (maximum value = 100). A support chart (Figure 15) then provides recommended ground support based on the RSR values for a 24-foot (7.3m) diameter circular tunnel.

Table 3: Rock Structure Rating: Parameter A, General area geology (Wickham et al., 1972)

	Basic Rock Type				Geological Structure			
	Hard	Medium	Soft	Decomposed		Slightly Folded or Faulted	Moderately Folded or Faulted	Intensively Folded or Faulted
Igneous	1	2	3	4				
Metamorphic	1	2	3	4				
Sedimentary	2	3	4	4	Massive			
Type 1					30	22	15	9
Type 2					27	20	13	8
Type 3					24	18	12	7
Type 4					19	15	10	6

Table 4: Rock Structure Rating: Parameter B, Joint pattern, direction of drive (Wickham et al., 1972)

	Strike ⊥ to Axis					Strike || to Axis		
	Direction of Drive					Direction of Drive		
	Both	With Dip		Against Dip		Either direction		
	Dip of Prominent Joints [a]					Dip of Prominent Joints		
Average joint spacing	Flat	Dipping	Vertical	Dipping	Vertical	Flat	Dipping	Vertical
1. Very closely jointed, < 2 in	9	11	13	10	12	9	9	7
2. Closely jointed, 2-6 in	13	16	19	15	17	14	14	11
3. Moderately jointed, 6-12 in	23	24	28	19	22	23	23	19
4. Moderate to blocky, 1-2 ft	30	32	36	25	28	30	28	24
5. Blocky to massive, 2-4 ft	36	38	40	33	35	36	24	28
6. Massive, > 4 ft	40	43	45	37	40	40	38	34

Table 5: Rock Structure Rating: Parameter C, Groundwater, joint condition (Wickham et al., 1972)

	Sum of Parameters *A* + *B*					
	13 - 44			45 - 75		
Anticipated water inflow	Joint Condition [b]					
gpm/1000 ft of tunnel	Good	Fair	Poor	Good	Fair	Poor
None	22	18	12	25	22	18
Slight, < 200 gpm	19	15	9	23	19	14
Moderate, 200-1000 gpm	15	22	7	21	16	12
Heavy, > 1000 gp	10	8	6	18	14	10

[a] Dip: flat: 0-20°; dipping: 20-50°; and vertical: 50-90°

[b] Joint condition: good = tight or cemented; fair = slightly weathered or altered; poor = severely weathered, altered or open

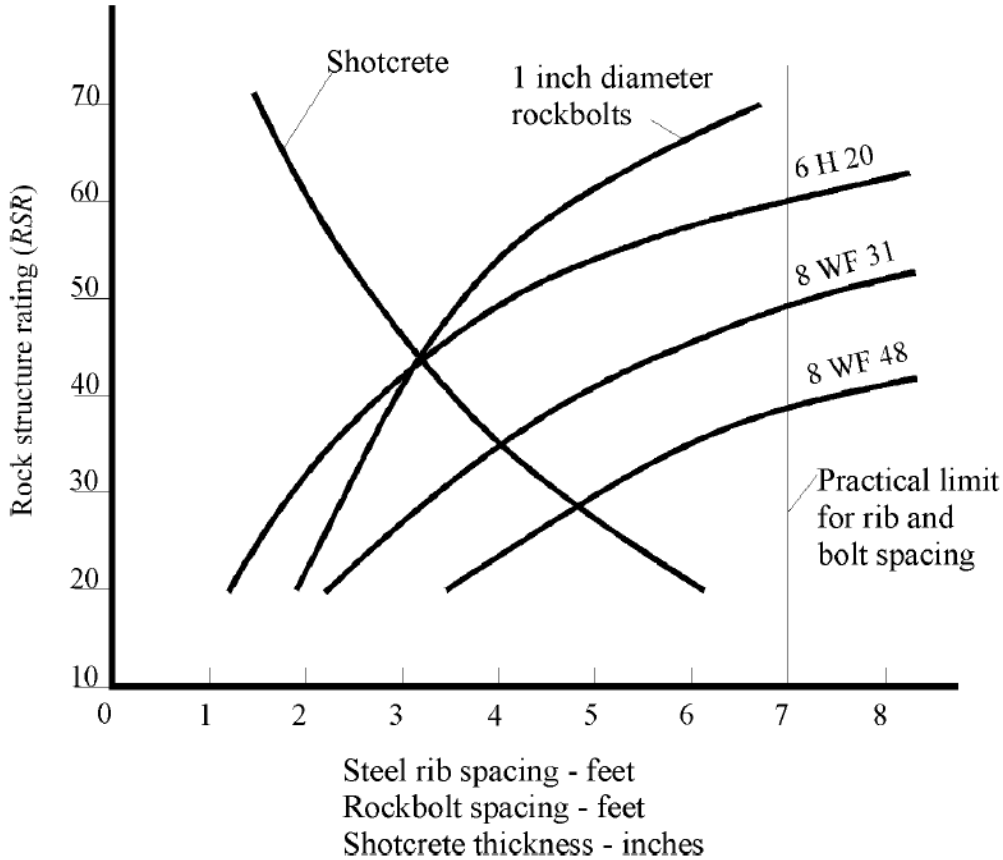

Figure 15: RSR support estimates for a 24 ft. (7.3 m) diameter circular tunnel (Wickham et al., 1972)

2.2.4 Bieniawski's Geomechanics Classification

Bieniawski (1976) developed a rock mass classification system called the Geomechanics Classification or the Rock Mass Rating (RMR) system which utilizes multiple input parameters to classify the rock mass quality. The system has been refined as more cased studies were added and updated versions (including Bieniawski, 1989) were developed. The RMR system uses five input parameters and a joint/discontinuity orientation adjustment factor which include:

- Unconfined Compressive Strength of the rock,
- Rock Quality Designation (RQD),
- Spacing of discontinuities,

- Condition of discontinuities,
- Groundwater condition,
- Orientation of discontinuities.

For each input, a value is assigned based on its condition. The inputs are then summed to provide a number ranging from 0-100. An adjustment is then made based on the impact of the orientation on the jointing/discontinuities. Details of two versions of the Rock Mass Rating system are presented in Table 6 for the 1976 version and Table 7 for the updated 1989 version.

Table 6: Rock Mass Rating system (Bieniawski, 1976)

Parameter				Ranges of Values						
1	Strength of intact rock material	Point-load strength index (MPa)		>8	4 – 8	2 – 4	1 – 2	For this low range, uniaxial compressive test is preferred		
		Uniaxial compressive strength (MPa)		>200	100 – 200	50 – 100	25 – 50	10 – 25	3 – 10	1 – 3
	Rating			15	12	7	4	2	1	0
2	Drill core quality RQD (%)			90 – 100	75 – 90	50 – 75	25 – 50	<25		
	Rating			20	17	13	8	3		
3	Spacing of discontinuities			>3 m	1 – 3 m	0.3 – 1 m	50 – 300 mm	<50 mm		
	Ratings			30	25	20	10	5		
4	Condition of discontinuities			Very rough surfaces Not continuous No separation Unweathered wall rock	Slightly rough surfaces Separation < 1 mm Hard rock walls	Slightly rough surfaces Separation < 1 mm Soft rock walls	Slickensided surfaces or Gouge < 5 mm thick or Separation 1 – 5 mm Continuous	Soft gouge > 5 mm thick or Separation > 5 mm Continuous		
	Rating			25	20	12	6	0		
5	Groundwater	Inflow per 10 m tunnel length (L/min)		None or		< 25 or	25 – 125 or	>125 or		
		Ratio	Joint water pressure / Major principal stress	0 or		0.1 – 0.2 or	0.2 – 0.5 or	>0.5 or		
		General conditions		Completely dry		Moist Only (interstitial water)	Water under moderate pressure	Severe water problems		
	Rating			10		7	4	0		

Table 7: Rock Mass Rating System (Bieniawski, 1989)

A. CLASSIFICATION PARAMETERS AND THEIR RATINGS

	Parameter		Range of values						
1	Strength of intact rock material	Point-load strength index	>10 MPa	4 - 10 MPa	2 - 4 MPa	1 - 2 MPa	For this low range - uniaxial compressive test is preferred		
		Uniaxial comp. strength	>250 MPa	100 - 250 MPa	50 - 100 MPa	25 - 50 MPa	5 - 25 MPa	1 - 5 MPa	< 1 MPa
	Rating		15	12	7	4	2	1	0
2	Drill core Quality *RQD*		90% - 100%	75% - 90%	50% - 75%	25% - 50%	< 25%		
	Rating		20	17	13	8	3		
3	Spacing of discontinuities		> 2 m	0.6 - 2 . m	200 - 600 mm	60 - 200 mm	< 60 mm		
	Rating		20	15	10	8	5		
4	Condition of discontinuities (See E)		Very rough surfaces Not continuous No separation Unweathered wall rock	Slightly rough surfaces Separation < 1 mm Slightly weathered walls	Slightly rough surfaces Separation < 1 mm Highly weathered walls	Slickensided surfaces or Gouge < 5 mm thick or Separation 1-5 mm Continuous	Soft gouge >5 mm thick or Separation > 5 mm Continuous		
	Rating		30	25	20	10	0		
5	Ground water	Inflow per 10 m tunnel length (l/m)	None	< 10	10 - 25	25 - 125	> 125		
		(Joint water press)/ (Major principal σ)	0	< 0.1	0.1, - 0.2	0.2 - 0.5	> 0.5		
		General conditions	Completely dry	Damp	Wet	Dripping	Flowing		
	Rating		15	10	7	4	0		

B. RATING ADJUSTMENT FOR DISCONTINUITY ORIENTATIONS (See F)

Strike and dip orientations		Very favourable	Favourable	Fair	Unfavourable	Very Unfavourable
Ratings	Tunnels & mines	0	-2	-5	-10	-12
	Foundations	0	-2	-7	-15	-25
	Slopes	0	-5	-25	-50	

C. ROCK MASS CLASSES DETERMINED FROM TOTAL RATINGS

Rating	100 ← 81	80 ← 61	60 ← 41	40 ← 21	< 21
Class number	I	II	III	IV	V
Description	Very good rock	Good rock	Fair rock	Poor rock	Very poor rock

D. MEANING OF ROCK CLASSES

Class number	I	II	III	IV	V
Average stand-up time	20 yrs for 15 m span	1 year for 10 m span	1 week for 5 m span	10 hrs for 2.5 m span	30 min for 1 m span
Cohesion of rock mass (kPa)	> 400	300 - 400	200 - 300	100 - 200	< 100
Friction angle of rock mass (deg)	> 45	35 - 45	25 - 35	15 - 25	< 15

E. GUIDELINES FOR CLASSIFICATION OF DISCONTINUITY conditions

Discontinuity length (persistence)	< 1 m	1 - 3 m	3 - 10 m	10 - 20 m	> 20 m
Rating	6	4	2	1	0
Separation (aperture)	None	< 0.1 mm	0.1 - 1.0 mm	1 - 5 mm	> 5 mm
Rating	6	5	4	1	0
Roughness	Very rough	Rough	Slightly rough	Smooth	Slickensided
Rating	6	5	3	1	0
Infilling (gouge)	None	Hard filling < 5 mm	Hard filling > 5 mm	Soft filling < 5 mm	Soft filling > 5 mm
Rating	6	4	2	2	0
Weathering	Unweathered	Slightly weathered	Moderately weathered	Highly weathered	Decomposed
Ratings	6	5	3	1	0

F. EFFECT OF DISCONTINUITY STRIKE AND DIP ORIENTATION IN TUNNELLING**

Strike perpendicular to tunnel axis		Strike parallel to tunnel axis	
Drive with dip - Dip 45 - 90°	Drive with dip - Dip 20 - 45°	Dip 45 - 90°	Dip 20 - 45°
Very favourable	Favourable	Very unfavourable	Fair
Drive against dip - Dip 45-90°	Drive against dip - Dip 20-45°	Dip 0-20 - Irrespective of strike°	
Fair	Unfavourable	Fair	

* Some conditions are mutually exclusive . For example, if infilling is present, the roughness of the surface will be overshadowed by the influence of the gouge. In such cases use A.4 directly.
** Modified after Wickham et al (1972).

Bieniawski (1989) also published a set of guidelines for tunnel support based on the RMR value (Table 8). The guidelines were developed for a 10m span horseshoe-shaped tunnel constructed using drill and blast techniques, with a vertical stress of less than 25 MPa (<900m below surface).

Table 8: Guidelines for excavation and support of 10m span rock tunnels (Bieniawski, 1989)

Rock mass class	Excavation	Rock bolts (20 mm diameter, fully grouted)	Shotcrete	Steel sets
I - Very good rock *RMR*: 81-100	Full face, 3 m advance.	Generally no support required except spot bolting.		
II - Good rock *RMR*: 61-80	Full face , 1-1.5 m advance. Complete support 20 m from face.	Locally, bolts in crown 3 m long, spaced 2.5 m with occasional wire mesh.	50 mm in crown where required.	None.
III - Fair rock *RMR*: 41-60	Top heading and bench 1.5-3 m advance in top heading. Commence support after each blast. Complete support 10 m from face.	Systematic bolts 4 m long, spaced 1.5 - 2 m in crown and walls with wire mesh in crown.	50-100 mm in crown and 30 mm in sides.	None.
IV - Poor rock *RMR*: 21-40	Top heading and bench 1.0-1.5 m advance in top heading. Install support concurrently with excavation, 10 m from face.	Systematic bolts 4-5 m long, spaced 1-1.5 m in crown and walls with wire mesh.	100-150 mm in crown and 100 mm in sides.	Light to medium ribs spaced 1.5 m where required.
V – Very poor rock *RMR*: < 20	Multiple drifts 0.5-1.5 m advance in top heading. Install support concurrently with excavation. Shotcrete as soon as possible after blasting.	Systematic bolts 5-6 m long, spaced 1-1.5 m in crown and walls with wire mesh. Bolt invert.	150-200 mm in crown, 150 mm in sides, and 50 mm on face.	Medium to heavy ribs spaced 0.75 m with steel lagging and forepoling if required. Close invert.

2.2.5 Barton et al.'s Rock Tunneling Quality Index (Q)

Developed in 1974 by Barton, Lien and Lunde at the Norwegian Geotechnical Institute, the Tunneling Quality Index (Q) is a quantitative rock mass classification based on 212 hard rock tunnel case studies in Scandinavia (Bieniawski, 1989). The Q value varies on a logarithmic scale from 0.001 to 1000 and is defined as:

Equation 3: Rock Tunneling Quality Index (Q)

$$Q = \frac{RQD}{Jn} \times \frac{Jr}{Ja} \times \frac{Jw}{SRF}$$

where

RQD	is the Rock Quality Designation
Jn	is the joint set number
Jr	is the joint roughness number
Ja	is the joint alteration number
Jw	is the joint water reduction factor
SRF	is the stress reduction factor

The Q index can be considered as a function of three parameters defined as:

1. Block Size *(RQD/Jn)*
2. Inter-block shear strength *(Jr/Ja)*
3. Active stress *(Jw/SRF)*

Details of the Tunnel Quality Index are presented in Tables 9-11.

Table 9: Classification of individual parameters, Q (Barton et al, 1974)

DESCRIPTION	VALUE	NOTES	
1. ROCK QUALITY DESIGNATION	*RQD*		
A. Very poor	0 - 25	1. Where *RQD* is reported or measured as ≤ 10 (including 0),	
B. Poor	25 - 50	a nominal value of 10 is used to evaluate *Q*.	
C. Fair	50 - 75		
D. Good	75 - 90	2. *RQD* intervals of 5, i.e. 100, 95, 90 etc. are sufficiently	
E. Excellent	90 - 100	accurate.	
2. JOINT SET NUMBER	J_n		
A. Massive, no or few joints	0.5 - 1.0		
B. One joint set	2		
C. One joint set plus random	3		
D. Two joint sets	4		
E. Two joint sets plus random	6		
F. Three joint sets	9	1. For intersections use $(3.0 \times J_n)$	
G. Three joint sets plus random	12		
H. Four or more joint sets, random,	15	2. For portals use $(2.0 \times J_n)$	
heavily jointed, 'sugar cube', etc.			
J. Crushed rock, earthlike	20		
3. JOINT ROUGHNESS NUMBER	J_r		
a. Rock wall contact			
b. Rock wall contact before 10 cm shear			
A. Discontinuous joints	4		
B. Rough and irregular, undulating	3		
C. Smooth undulating	2		
D. Slickensided undulating	1.5	1. Add 1.0 if the mean spacing of the relevant joint set is	
E. Rough or irregular, planar	1.5	greater than 3 m.	
F. Smooth, planar	1.0		
G. Slickensided, planar	0.5	2. J_r = 0.5 can be used for planar, slickensided joints having	
c. No rock wall contact when sheared		lineations, provided that the lineations are oriented for	
H. Zones containing clay minerals thick	1.0	minimum strength.	
enough to prevent rock wall contact	(nominal)		
J. Sandy, gravely or crushed zone thick	1.0		
enough to prevent rock wall contact	(nominal)		
4. JOINT ALTERATION NUMBER	J_a	ϕr degrees (approx.)	
a. Rock wall contact			
A. Tightly healed, hard, non-softening,	0.75		1. Values of ϕr, the residual friction angle,
impermeable filling			are intended as an approximate guide
B. Unaltered joint walls, surface staining only	1.0	25 - 35	to the mineralogical properties of the
C. Slightly altered joint walls, non-softening	2.0	25 - 30	alteration products, if present.
mineral coatings, sandy particles, clay-free			
disintegrated rock, etc.			
D. Silty-, or sandy-clay coatings, small clay-	3.0	20 - 25	
fraction (non-softening)			
E. Softening or low-friction clay mineral coatings,	4.0	8 - 16	
i.e. kaolinite, mica. Also chlorite, talc, gypsum			
and graphite etc., and small quantities of swelling			
clays. (Discontinuous coatings, 1 - 2 mm or less)			

Table 10: Classification of individual parameters, Q (Barton et al, 1974) cont'd

DESCRIPTION	VALUE	NOTES	
4, JOINT ALTERATION NUMBER	J_a	ϕr degrees (approx.)	
b. Rock wall contact before 10 cm shear			
F. Sandy particles, clay-free, disintegrating rock etc.	4.0	25 - 30	
G. Strongly over-consolidated, non-softening clay mineral fillings (continuous < 5 mm thick)	6.0	16 - 24	
H. Medium or low over-consolidation, softening clay mineral fillings (continuous < 5 mm thick)	8.0	12 - 16	
J. Swelling clay fillings, i.e. montmorillonite, (continuous < 5 mm thick). Values of J_a depend on percent of swelling clay-size particles, and access to water.	8.0 - 12.0	6 - 12	
c. No rock wall contact when sheared			
K. Zones or bands of disintegrated or crushed	6.0		
L. rock and clay (see G, H and J for clay	8.0		
M. conditions)	8.0 - 12.0	6 - 24	
N. Zones or bands of silty- or sandy-clay, small clay fraction, non-softening	5.0		
O. Thick continuous zones or bands of clay	10.0 - 13.0		
P. & R. (see G.H and J for clay conditions)	6.0 - 24.0		
5. JOINT WATER REDUCTION	J_w	approx. water pressure (kgf/cm^2)	
A. Dry excavation or minor inflow i.e. < 5 l/m locally	1.0	< 1.0	
B. Medium inflow or pressure, occasional outwash of joint fillings	0.66	1.0 - 2.5	
C. Large inflow or high pressure in competent rock with unfilled joints	0.5	2.5 - 10.0	1. Factors C to F are crude estimates; increase J_w if drainage installed.
D. Large inflow or high pressure	0.33	2.5 - 10.0	
E. Exceptionally high inflow or pressure at blasting, decaying with time	0.2 - 0.1	> 10	2. Special problems caused by ice formation are not considered.
F. Exceptionally high inflow or pressure	0.1 - 0.05	> 10	
6. STRESS REDUCTION FACTOR		*SRF*	
a. Weakness zones intersecting excavation, which may cause loosening of rock mass when tunnel is excavated			
A. Multiple occurrences of weakness zones containing clay or chemically disintegrated rock, very loose surrounding rock any depth)		10.0	1. Reduce these values of *SRF* by 25 - 50% but only if the relevant shear zones influence do not intersect the excavation
B. Single weakness zones containing clay, or chemically dis-tegrated rock (excavation depth < 50 m)		5.0	
C. Single weakness zones containing clay, or chemically dis-tegrated rock (excavation depth > 50 m)		2.5	
D. Multiple shear zones in competent rock (clay free), loose surrounding rock (any depth)		7.5	
E. Single shear zone in competent rock (clay free). (depth of excavation < 50 m)		5.0	
F. Single shear zone in competent rock (clay free). (depth of excavation > 50 m)		2.5	
G. Loose open joints, heavily jointed or 'sugar cube', (any depth)		5.0	

Table 11: Classification of individual parameters, Q (Barton et al, 1974) cont'd

DESCRIPTION	VALUE			NOTES
6. STRESS REDUCTION FACTOR			*SRF*	
b. Competent rock, rock stress problems				
	σ_c/σ_1	σ_t/σ_1		2. For strongly anisotropic virgin stress field
H. Low stress, near surface	> 200	> 13	2.5	(if measured): when $5 \leq \sigma_1/\sigma_3 \leq 10$, reduce σ_c
J. Medium stress	200 - 10	13 - 0.66	1.0	to $0.8\sigma_c$ and σ_t to $0.8\sigma_t$. When $\sigma_1/\sigma_3 > 10$,
K. High stress, very tight structure	10 - 5	0.66 - 0.33	0.5 - 2	reduce σ_c and σ_t to $0.6\sigma_c$ and $0.6\sigma_t$, where
(usually favourable to stability, may				σ_c = unconfined compressive strength, and
be unfavourable to wall stability)				σ_t = tensile strength (point load) and σ_1 and
L. Mild rockburst (massive rock)	5 - 2.5	0.33 - 0.16	5 - 10	σ_3 are the major and minor principal stresses.
M. Heavy rockburst (massive rock)	< 2.5	< 0.16	10 - 20	3. Few case records available where depth of
c. Squeezing rock, plastic flow of incompetent rock				crown below surface is less than span width.
under influence of high rock pressure				Suggest *SRF* increase from 2.5 to 5 for such
N. Mild squeezing rock pressure			5 - 10	cases (see H).
O. Heavy squeezing rock pressure			10 - 20	
d. Swelling rock, chemical swelling activity depending on presence of water				
P. Mild swelling rock pressure			5 - 10	
R. Heavy swelling rock pressure			10 - 15	

ADDITIONAL NOTES ON THE USE OF THESE TABLES

When making estimates of the rock mass Quality (*Q*), the following guidelines should be followed in addition to the notes listed in the tables:

1. When borehole core is unavailable, *RQD* can be estimated from the number of joints per unit volume, in which the number of joints per metre for each joint set are added. A simple relationship can be used to convert this number to *RQD* for the case of clay free rock masses: $RQD = 115 - 3.3 J_v$ (approx.), where J_v = total number of joints per m^3 ($0 < RQD < 100$ for $35 > J_v > 4.5$).
2. The parameter J_n representing the number of joint sets will often be affected by foliation, schistosity, slaty cleavage or bedding etc. If strongly developed, these parallel 'joints' should obviously be counted as a complete joint set. However, if there are few 'joints' visible, or if only occasional breaks in the core are due to these features, then it will be more appropriate to count them as 'random' joints when evaluating J_n.
3. The parameters J_r and J_a (representing shear strength) should be relevant to the weakest significant joint set or clay filled discontinuity in the given zone. However, if the joint set or discontinuity with the minimum value of J_r/J_a is favourably oriented for stability, then a second, less favourably oriented joint set or discontinuity may sometimes be more significant, and its higher value of J_r/J_a should be used when evaluating *Q*. The value of J_r/J_a should in fact relate to the surface most likely to allow failure to initiate.
4. When a rock mass contains clay, the factor *SRF* appropriate to loosening loads should be evaluated. In such cases the strength of the intact rock is of little interest. However, when jointing is minimal and clay is completely absent, the strength of the intact rock may become the weakest link, and the stability will then depend on the ratio rock-stress/rock-strength. A strongly anisotropic stress field is unfavourable for stability and is roughly accounted for as in note 2 in the table for stress reduction factor evaluation.
5. The compressive and tensile strengths (σ_c and σ_t) of the intact rock should be evaluated in the saturated condition if this is appropriate to the present and future in situ conditions. A very conservative estimate of the strength should be made for those rocks that deteriorate when exposed to moist or saturated conditions.

2.2.6 The Stability Graph Method

Introduced in 1981 by Mathews et. al., the stability graph method was developed to assess the stability of designed spans of stope walls. The original database of 26 case histories from three mines was expanded to 175 case histories from 34 mines by Potvin (1988) with a re-calibration of the stability number factors

(Suorineni, 2010) resulting in the modified stability graph. Potvin and Milne (1992) and Nickson (1992) further expanded the database to over 350 case histories.

The modified stability graph consists of two factors: a stability number *N'* and the hydraulic radius or shape factor *S* which are plotted against one another to determine the overall stability of the stope size and shape (Figure 16).

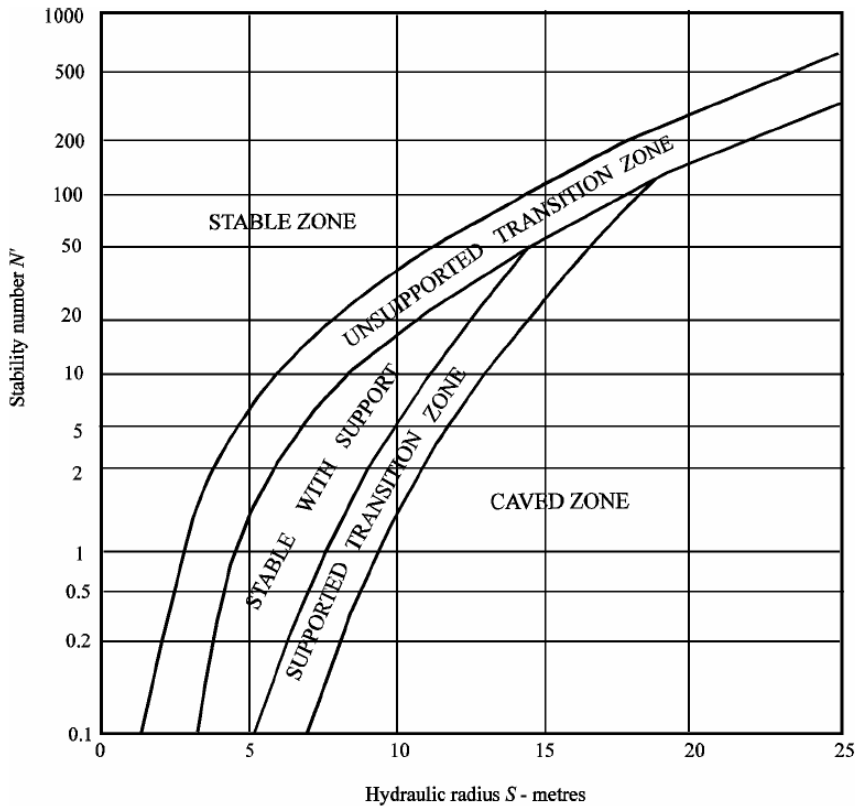

Figure 16: Stability graph (Nickson, 1992)

The hydraulic radius, or shape factor, S, for the stope surface is defined in Equation 4:

Equation 4: Hydraulic Radius (Shape Factor)

$$S = \frac{Cross\ sectional\ area\ of\ surface\ analysed}{Perimeter\ of\ surface\ analysed}$$

The stability number, *N'*, is defined in Equation 5.

Equation 5: Modified Stability Number

$$N' = Q' x\ A\ x\ B\ x\ C$$

where

Q’ is the modified Q Tunneling Quality Index as defined:

$$Q' = \frac{RQD}{Jn} \times \frac{Jr}{Ja}$$

A is the rock stress factor

B is the joint orientation adjustment factor

C is the gravity adjustment factor

Q’ is used instead of Q since the stability number applies a stress factor A, which accounts for the stress quotient (Jw/SRF) of Q.

Figure 17 illustrates how each of the adjustment factors is calculated for the stability number.

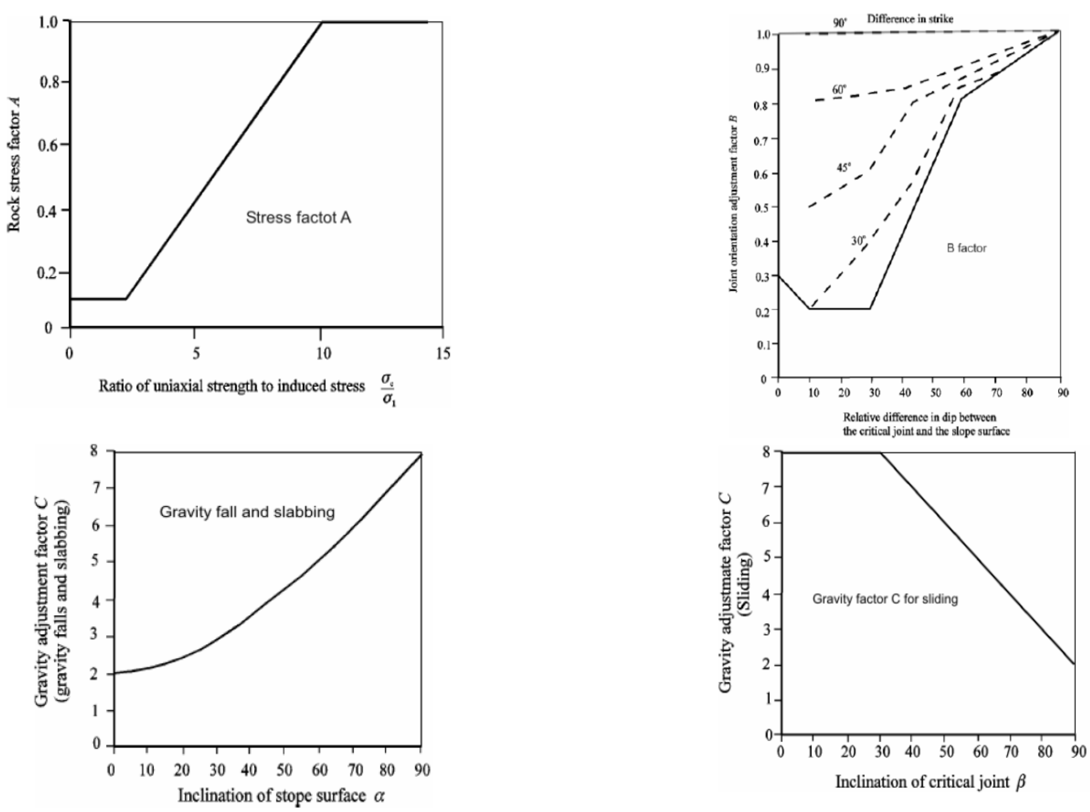

Figure 17: Empirical charts for determination of factors A, B and C (Potvin, 1988)

2.3 Mining Method Selection Tools

Originally, mining methods were selected based on the operating experience with similar types of deposits and on methods already in use in the area (Nicholas, 1981). However, in recent times, selection tools have been developed to aid engineers in determining appropriate mining methods for a deposit.

Although all the mining method selection tools reviewed are based on the characteristics of the target orebody, other factors also influence the determination of a preferred mining method. For example, in developed countries, significantly higher labour costs and regulatory requirements regarding worker safety favour increased mechanization of mining.

2.3.1 Morrison 1976

Morrison introduced a general classification of mining methods based on ground control approaches. Three categories of ground control approaches are defined as:

- Rigid Pillars
- Controlled Subsidence
- Caving

Morrison developed a chart (Figure 18) where the three ground control approaches are placed at the core of larger circles and the various mining methods that can be used to implement them surround these approaches, and vary depending on the ore width, stope support and strain energy accumulation conditions.

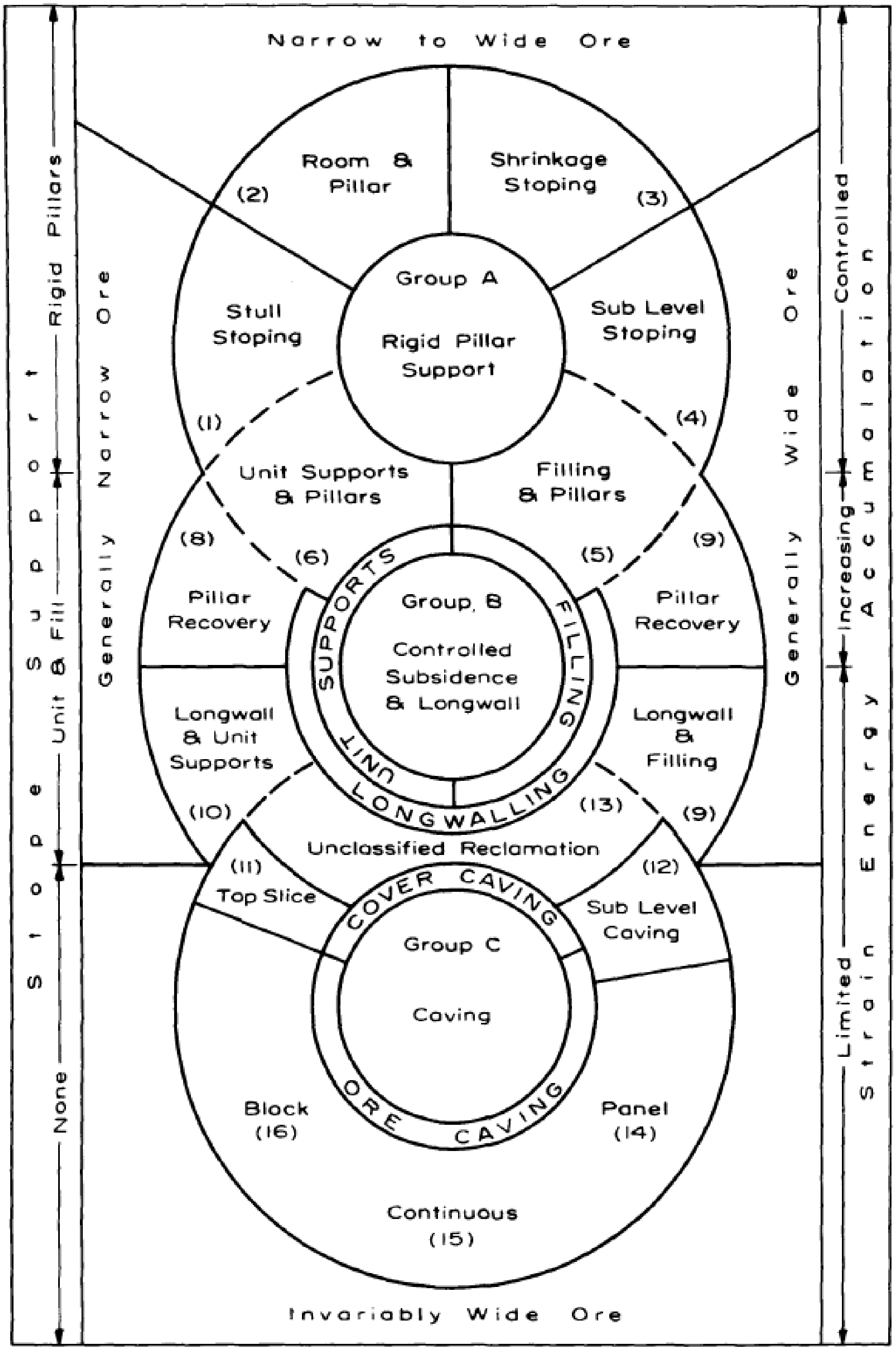

Figure 18: Mining methods-a classification (Morrison, 1976)

Group A — Rigid Pillars

These methods are self-supporting based on the pillar support in place and/or the limited mining spans. The support left in place minimizes subsidence and generally ensures that ground support is not an issue. These methods tend to be deployed in shallower depths with stronger rocks to ensure an adequate factor of safety. Methods include Room and Pillar, Shrinkage stoping and Sublevel stoping.

Group B — Controlled Subsidence/Longwall

The methods consider that the surrounding rockmass will eventually fail and that support is placed in order to control the rate and location of failure. Category B has a central circle around the core indicating that, if the support is a fill method or consists of unit type supports, or lies within a range suitable for longwall mining, controlled mining is optimal. The methods were historically made up of timbering methods but have evolved over time into cut and fill methods and sublevel stoping with backfilled stopes.

Group C — Caving

In caving methods, the development zones in which people are working are supported but the ore and waste rock must be allowed to cave in order to extract the ore. Category C consists of a central circle around the core indicating that a method is either cover caving or caving of the ore. Methods include sublevel caving (cover caving) and block caving (caving of the ore).

2.3.2 De Souza et al. 1987

De Souza and Archibald (1987) proposed a mining method selection tool based on rock mass classification. The method selection procedure consists of three steps:

1. Correlation between orebody geometry and mining methods is used as an initial screen to eliminate method extremes that are not suited for the orebody geometry (Figure 19).
2. Correlation between orebody geomechanic characteristics and mining methods (Figure 20).

3. Economic feasibility analysis is performed for the preferred methods selected from the second step. This reduces the number of methods in which a detailed analysis would be completed on for a feasibility study (Figure 21).

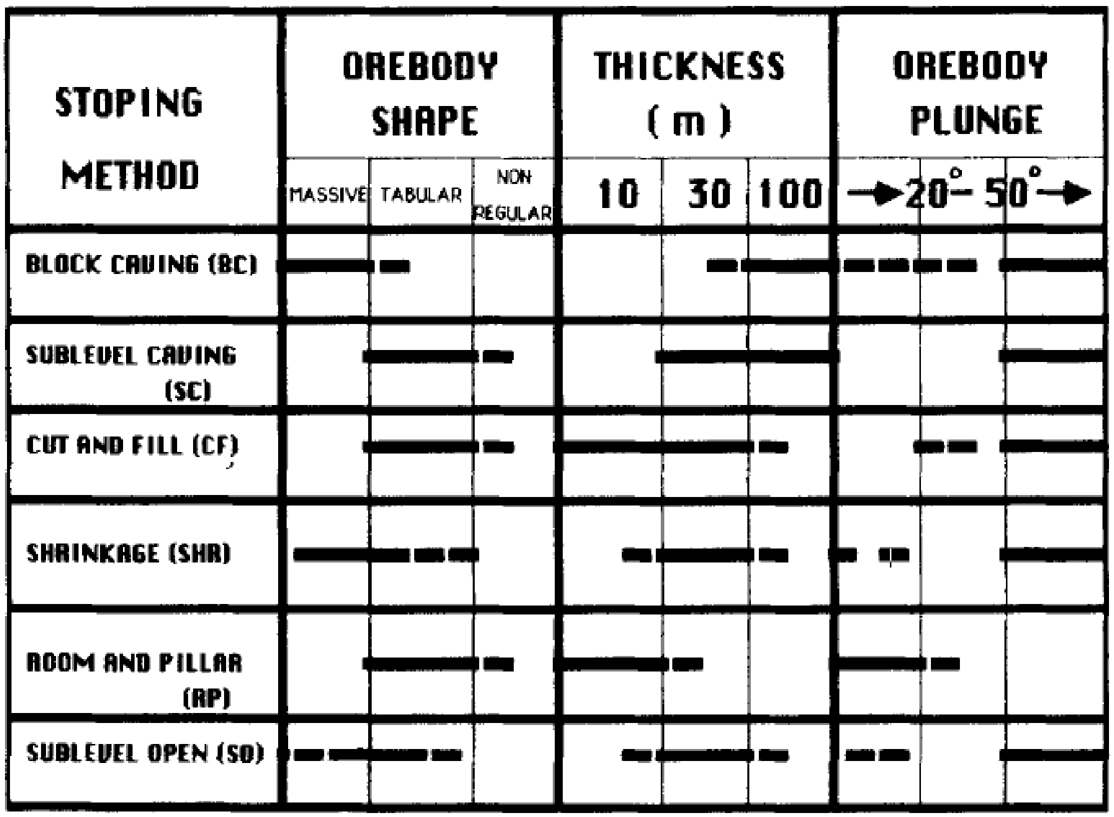

Figure 19: Orebody geometry vs. mining method selection chart (De Souza et. al., 1987)

STOPING METHOD	OREBODY	HANGING WALL	FOOTWALL
BLOCK CAVING	P1 - P4	P2 - P4	F - G2
SUBLEVEL CAVING	P3 - P4	P3 - P4	F - G2
CUT AND FILL	P4 - G1	P3 - P4	P3 - F
SHRINKAGE	F - G2	P4 - F	F - G2
ROOM AND PILLAR	G2 - G3	G1 - G3	G - G3
SUBLEVEL OPEN	G2 - G4	G2 - G4	G2 - G4

P1 – Exceptionally poor
P2 – Extremely poor
P3 – Very poor
P4 – Poor
F – Fair
G1 – Good
G2 – Very good
G3 – Extremely good
G4 – Exceptionally good

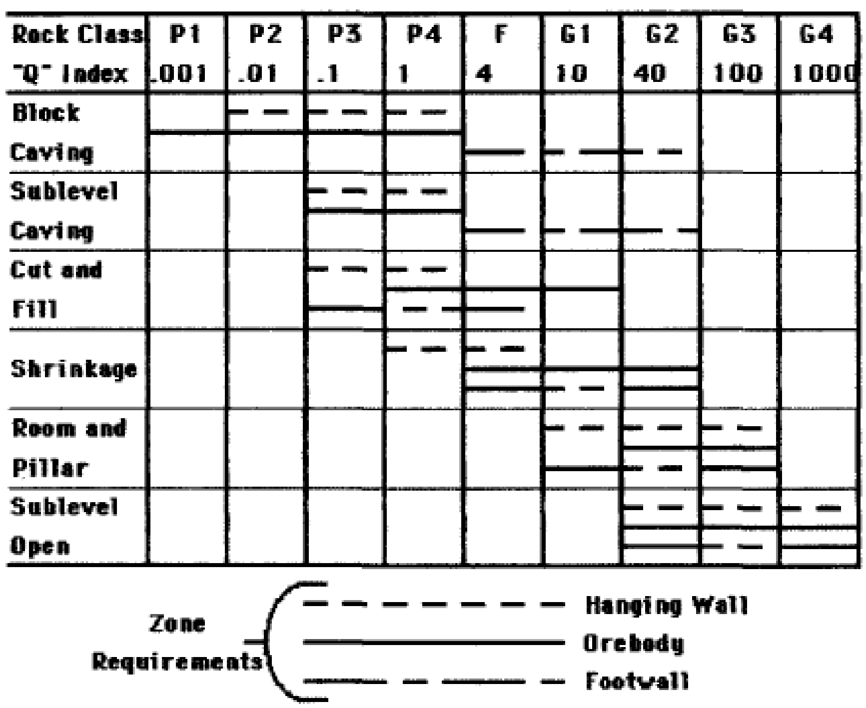

Figure 20: Rock mass classification vs. mining method requirements (De Souza et. al., 1987)

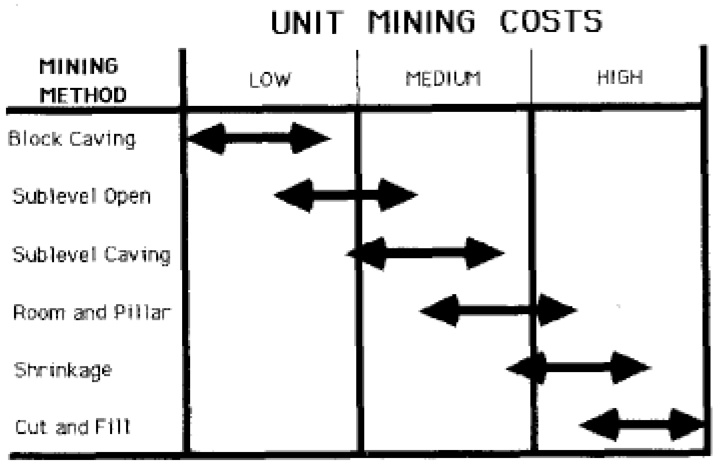

Figure 21: Unit mining cost comparison for selected mining methods (De Souza et. al., 1987)

2.3.3 Nicholas 1981/1992

Nicholas (1981, 1992) proposed a numerical process for selecting a mining method. He identified seven key parameters for choosing a mining method:

- Geometry and grade distribution of deposit,
- Rock mass strength of ore zone, hanging wall and footwall,
- Mining costs and capitalization requirements,
- Mining rate,
- Type and availability of labour,
- Environmental concerns,
- Other site-specific considerations.

Nicholas determined that the first two parameters along with mining costs had the greatest influence in selecting a mining method. His method assumes that significant reserves have been defined and little or no development has been completed. Based on these assumptions, Nicolas proposed a two-stage approach. First, the deposit's geometry, grade distribution and rock mechanics properties are determined. From these, the mining methods are ranked to determine the most applicable method. In stage two, the rock mechanics information is used to determine underground opening sizes, ground support requirements, orientation of drifts, and caving characteristics. Nicholas created five categories for the Geometry and Grade Distribution of a deposit:

- General Shape
  - Equi-dimensional: all dimensions are on the same order of magnitude
  - Platy-tabular: two dimensions are many times the thickness, which does not usually exceed 100m
  - Irregular: dimensions vary over short distances
- Ore Thickness
  - Narrow: <10m
  - Intermediate 10m – 30m
  - Thick: 20m – 100m
  - Very Thick: >100m
- Plunge
  - Flat: <20º
  - Intermediate: 20º - 55º
  - Steep: >55º
- Depth below surface
  - Provide actual depth
- Grade Distribution
  - Uniform: the grade at any point in the deposit does not vary significantly from the mean grade for the deposit
  - Gradational: grade values have zonal characteristics, and the grades change gradually from one to another
  - Erratic: grade values change radically over short distances and do not exhibit any discernible pattern in their changes

The rock mechanics characteristics used are:

1) Rock Substance Strength (Uniaxial strength (Pa)/overburden pressure (Pa))
    a) Weak: <8
    b) Moderate: 8 – 15
    c) Strong: >15
2) Fracture Spacing

Fractures/m	%ROD
i) Very Close: >16	0 – 20
ii) Close: 10-16	20 – 40
iii) Wide: 3 – 10	40 – 70
iv) Very Wide: <3	70 – 100

3) Fracture Shear Strength
    a) Weak: Clean joint with a smooth surface or fill with material whose strength is less than rock substance strength
    b) Moderate: Clean joint with a rough surface
    c) Joint is filled with a material that is equal to a stronger than rock substance strength

These parameters were applied to ten basic mining methods:

1) Open Pit
2) Block Caving
3) Sublevel Caving
4) Sublevel Stoping
5) Longwall
6) Room and Pillar
7) Shrinkage Stoping
8) Cut and Fill
9) Top Slicing

10) Square-Set

Each mining method was then ranked based on its suitability for the geometry and grade distribution of the deposit and the rock mechanics characteristics of the ore zone, footwall and hanging wall. Nicholas applied the following four ranks:

- *Preferred: characteristic is preferred for the mining method,*
- *Probable: if the characteristic exists, the mining method can be used,*
- *Unlikely: if the characteristic exists, it is unlikely that the mining method would be applied, but does not completely rule of the method, and*
- *Eliminated: if the characteristic exists, then the mining method cannot be used.*

The Rank values are listed in Table 12.

Table 12: Rank Value (Nicholas, 1981)

Ranking	Value
Preferred	3-4
Probable	1-2
Unlikely	0
Eliminated	-49

Each mining method has values assigned to the input parameters to determine the suitability of the mining method to the parameter. This is shown in Tables 13-16.

Table 13: Ranking of Geometry/Grade Distribution for Mining Methods (Nicholas, 1981)

Mining Method	General Shape			Ore Thickness				Ore Plunge			Grade Distribution		
	Massive	Tabular/Platey	Irregular	Narrow	Intermediate	Thick	Very Thick	Flat	Intermediate	Steep	Uniform	Gradational	Erratic
Open Pit	3	2	3	2	3	4	4	3	3	4	3	3	3
Block Caving	4	2	0	-49	0	2	4	3	2	4	4	2	0
Sublevel Stoping	2	2	1	1	2	4	3	2	1	4	3	3	1
Sublevel Caving	3	4	1	-49	0	4	4	1	1	4	4	2	0
Longwall	-49	4	-49	4	0	-49	-49	4	0	-49	4	2	0
Room and Pillar	0	4	2	4	2	-49	-49	4	1	0	3	3	3
Shrinkage Stopng	2	2	1	1	2	4	3	2	1	4	3	2	1
Cut and Fill	0	4	2	4	4	0	0	0	3	4	3	3	3
Top Slicing	3	3	0	-49	0	3	4	4	1	2	4	2	0
Square Set	0	2	4	4	4	1	1	2	3	3	3	3	3

Table 14: Ranking of Ore Zone Rock Mechanics Characteristics for Mining Methods (Nicholas, 1981)

Mining Method	Rock Substance Strength			Fracture Spacing				Fracture Strength		
	Weak	Moderate	Strong	Very Close	Close	Weak	Very Weak	Weak	Moderate	Strong
Open Pit	3	4	4	2	3	4	4	2	3	4
Block Caving	4	1	1	4	4	3	0	4	3	0
Sublevel Stoping	-49	3	4	0	0	1	4	0	2	4
Sublevel Caving	0	3	3	0	2	4	4	0	2	2
Longwall	4	1	0	4	4	0	0	4	3	0
Room and Pillar	0	3	4	0	1	2	4	0	2	4
Shrinkage Stopng	1	3	4	0	1	3	4	0	2	4
Cut and Fill	3	2	2	3	3	2	2	3	3	2
Top Slicing	2	3	3	1	1	2	4	1	2	4
Square Set	4	1	1	4	4	2	1	4	3	2

Table 15: Ranking of H/W Rock Mechanics Characteristics for Mining Methods (Nicholas, 1981)

Mining Method	Rock Substance Strength			Fracture Spacing				Fracture Strength		
	Weak	Moderate	Strong	Very Close	Close	Weak	Very Weak	Weak	Moderate	Strong
Open Pit	3	4	4	2	3	4	4	2	3	4
Block Caving	4	2	1	3	4	3	0	4	2	0
Sublevel Stoping	-49	3	4	-49	0	1	4	0	2	4
Sublevel Caving	3	2	1	3	4	3	1	4	2	0
Longwall	4	2	0	4	4	3	0	4	2	0
Room and Pillar	0	3	4	0	1	2	4	0	2	4
Shrinkage Stopng	4	2	1	4	4	3	0	4	2	0
Cut and Fill	3	2	2	3	3	2	2	4	3	2
Top Slicing	4	2	1	3	3	3	0	4	2	0
Square Set	3	2	2	3	3	2	2	4	3	2

Table 16: Ranking of F/W Rock Mechanics Characteristics for Mining Methods (Nicholas, 1981)

Mining Method	Rock Substance Strength			Fracture Spacing				Fracture Strength		
	Weak	Moderate	Strong	Very Close	Close	Weak	Very Weak	Weak	Moderate	Strong
Open Pit	3	4	4	2	3	4	4	2	3	4
Block Caving	2	3	3	1	3	3	3	1	3	3
Sublevel Stoping	0	2	4	0	0	2	4	0	1	4
Sublevel Caving	0	2	4	0	1	3	4	0	2	4
Longwall	2	3	3	1	2	4	3	1	3	3
Room and Pillar	0	2	4	0	1	3	3	0	3	3
Shrinkage Stopng	2	3	3	2	3	3	2	2	2	3
Cut and Fill	4	2	2	4	4	2	2	4	4	2
Top Slicing	2	3	3	1	3	3	3	1	2	3
Square Set	4	2	2	4	4	2	2	4	4	2

Once the values have been assigned to each parameter, the values are totaled up with the highest total representing the mining method with the most preferred characteristics for the deposit. Lower total values represent mining methods that may be used. Total that are below zero represent a mining method that is not appropriate for the deposit.

2.3.4 UBC Mining Method Selection, Miller-Tait et al., 1995

Miller-Tait et al. (1997) proposed an empirically derived modification to the Nicholas Method (Nicholas, 1981). The UBC method is similar to the Nicholas method in that both approaches rank the Geometry/Grade Distribution of the deposit and the rock mechanics characteristics. However, the UBC method modified each characteristic, except grade distribution and plunge, to better represent stope mining. This is done based on improved rock support techniques, more technical oversight of production and the increased size and sophistication of mechanical equipment. In one case, a value of -10 has been introduced to strongly discount a method rather than eliminating it by using the -49 value of the Nicholas Method.

For ore width, a very narrow category defined as 0-3m in thickness was added to account for narrow vein mining. The additional of a very narrow vein category applies a discount to the option of using an open stoping method due to dilution control issues. Preference is given to more manual mining methods using jacklegs or stopers.

The rock mechanics ratings were also adjusted to use Bieniawski's 1976 Rock Mass Rating system, instead of the Rock Mechanics characterization system proposed by Nicholas, for fracture spacing and fracture shear strength. The reason for this was to use a more general rating system for an initial assessment as well as to take advantage of the universal use of the Rock Mass Rating for consistency of data analysis.

The rock substance strength was also modified to account for the maximum in situ stress instead of the overburden pressure. This is based on many Canadian mines experiencing horizontal stresses two or more times the overburden pressure. Also, a very weak category was added to account for conditions where it would be unsafe for manned entry without ground support.

The UBC Mining Method Selection process is as follows:

1. General Shape
    a. Equi-dimensional: all dimensions are on the same order of magnitude
    b. Platety-tabular: two dimensions are many times the thickness, which does not usually exceed 35m
    c. Irregular: dimensions vary over short distances
2. Ore Thickness
    a. Very Narrow: <3m
    b. Narrow: 3-10m
    c. Intermediate 10m – 30m
    d. Thick: 20m – 100m
    e. Very Thick: >100m
3. Plunge
    a. Flat: <20º
    b. Intermediate: 20º - 55º
    c. Steep: >55º
4. Depth below surface
    a. Shallow: 0-100m
    b. Intermediate: 100-600m
    c. Deep: >600m
5. Grade Distribution

a. Uniform: the grade at any point in the deposit does not vary significantly from the mean grade for the deposit

b. Gradational: grade values have zonal characteristics, and the grades change gradually from one to another

c. Erratic: grade values change radically over short distances and do not exhibit any discernible pattern in their changes

6. Rock Mass Ratings

    a. Very Weak: 0-20

    b. Weak: 20-40

    c. Moderate: 40-60

    d. Strong: 60-80

    e. Very Strong: 80-100

7. Rock Substance Strength (Uniaxial strength/maximum principal stress)

    a. Very Weak: <5

    b. Weak: 5-10

    c. Moderate: 10-15

    d. Strong: >15

Tables 17-20 show the updated mining method parameter values.

Table 17: Geometry/Grade Distribution Rating for Mining Methods Part 1 (Miller-Tait et. al., 1995)

Mining Method	General Shape			Ore Thickness					Ore Plunge		
	Massive	Tabular or platy	Irregular	Very Narrow	Narrow	Intermediate	Thick	Very Thick	Flat	Intermediate	Steep
Open Pit	4	2	3	1	2	3	4	4	3	3	1
Block Caving	4	2	0	-49	-49	0	3	4	3	2	4
Sublevel Stoping	3	4	1	-10	1	3	4	3	2	1	4
Sublevel Caving	3	4	1	-49	-49	0	4	4	1	1	4
Longwall	-49	4	-49	4	3	0	-49	-49	4	0	-49
Room and Pillar	0	4	2	4	3	1	-49	-49	4	0	-49
Shrinkage Stopng	0	4	2	4	4	0	-49	-49	-49	0	4
Cut and Fill	1	4	4	3	4	4	1	0	1	3	4
Top Slicing	1	2	0	1	1	0	2	1	4	2	0
Square Set	0	1	4	4	3	2	0	0	2	3	2

Table 18: Geometry/Grade Distribution Rating for Mining Methods Part 2 (Miller-Tait et. al., 1995)

Mining Method	Grade Distribution			Depth		
	Uniform	Gradation	Erratic	Shallow	Intermediate	Deep
Open Pit	3	3	2	4	0	-49
Block Caving	3	2	2	2	3	3
Sublevel Stoping	4	4	3	3	4	2
Sublevel Caving	3	2	2	3	2	2
Longwall	4	1	0	2	2	3
Room and Pillar	4	2	0	3	3	2
Shrinkage Stopng	3	2	2	3	3	2
Cut and Fill	2	3	4	2	3	4
Top Slicing	2	1	1	2	1	1
Square Set	0	1	3	1	1	2

Table 19: Rock Mass Rating for Mining Methods (Miller-Tait et. al., 1995)

Mining Method	Ore Zone					Hanging Wall					Footwall				
	Very Weak	Weak	Moderate	Strong	Very Strong	Very Weak	Weak	Moderate	Strong	Very Strong	Very Weak	Weak	Moderate	Strong	Very Strong
Open Pit	3	3	3	3	3	2	3	4	4	4	2	3	4	4	4
Block Caving	4	3	2	0	-49	3	3	3	2	2	3	3	3	2	2
Sublevel Stoping	1	3	4	4	4	-49	0	3	4	4	0	0	2	3	3
Sublevel Caving	3	4	3	1	0	4	4	3	2	2	1	2	3	3	3
Longwall	6	6	4	2	2	6	5	4	3	3	-	-	-	-	-
Room and Pillar	-49	0	3	5	6	-49	0	3	5	6	-	-	-	-	-
Shrinkage Stopng	0	1	3	3	3	0	0	2	4	4	0	0	2	3	3
Cut and Fill	0	1	2	3	3	3	5	4	3	3	3	3	2	2	2
Top Slicing	3	2	1	1	0	0	0	2	3	3	0	0	1	2	2
Square Set	4	4	1	0	0	4	4	1	0	0	3	1	0	0	0

Table 20: Rock Substance Strength for Mining Methods (Miller-Tait et. al., 1995)

Mining Method	Ore Zone				Hanging Wall				Footwall			
	Very Weak	Weak	Moderate	Strong	Very Weak	Weak	Moderate	Strong	Very Weak	Weak	Moderate	Strong
Open Pit	4	3	3	3	3	3	4	4	3	3	4	4
Block Caving	4	2	1	0	4	3	2	0	4	3	2	1
Sublevel Stoping	0	2	4	4	0	1	4	5	0	1	3	3
Sublevel Caving	2	3	3	2	4	3	2	1	1	2	2	2
Longwall	6	5	2	1	6	5	2	2	-	-	-	-
Room and Pillar	0	0	3	6	3	0	2	6	-	-	-	-
Shrinkage Stopng	0	1	3	4	0	1	3	4	0	2	3	3
Cut and Fill	0	1	3	3	3	5	4	2	1	3	2	2
Top Slicing	3	2	1	0	3	2	2	2	2	2	1	1
Square Set	4	3	1	0	4	2	1	0	3	2	0	0

As with Nicholas' version, the values are totaled up with the highest total representing the mining method with the most preferred characteristics for the deposit. Lower total values represent mining methods that may be used. Total that are below zero represent a mining method that is not appropriate for the deposit.

2.4 Empirical Design Methods

Empirical methods are the most widely used design techniques employed within the mining industry largely due to their success in the design of mine structures (Pakalnis, 2015). Over the years empirical methods have been developed to support detailed mine design features, including ground support requirements, stope size and dilution, pillar size, maximum span widths, etc.

2.4.1 Grimstad and Barton Ground Support (Grimstad and Barton, 1993)

In relating the Q index to stability and support requirements of underground excavations, an additional parameter, the Equivalent Dimension, De, is defined in Equation 6:

Equation 6: Equivalent Dimension

$$De = \frac{Excavation\ span, diameter\ or\ height\ (m)}{Excavation\ Support\ Ratio\ ESR}$$

The Excavation Support Ratio (ESR) is determined by the intended use of the excavation and the degree of certainty in stability that the installed support system is to maintain. The ESR values are as follows (Table 21)

Table 21: Excavation Support Ratio (ESR) Categories (Barton et al, 1974)

Excavation Category		ESR
A	Temporary mine openings	3-5
B	Permanent mine openings, water tunnels for hydro power (Excluding high pressure penstocks), pilot tunnels, drifts and headings for large excavations	1.6
C	Storage rooms, water treatment plants, minor road and railway tunnels, surge chambers, access tunnels	1.3
D	Power stations, major road and railway tunnels, civil defense chambers, portal intersections	1.0
E	Underground nuclear power stations, railway stations, sports and public facilities, factories	0.8

Barton et al (1974) created several support categories that are defined by plotting Q against the Equivalent Dimension, De. The chart was updated by Grimstad and Barton (1993) **(**Figure 22) to account for the increasing use of steel fiber reinforced shotcrete in underground excavation support.

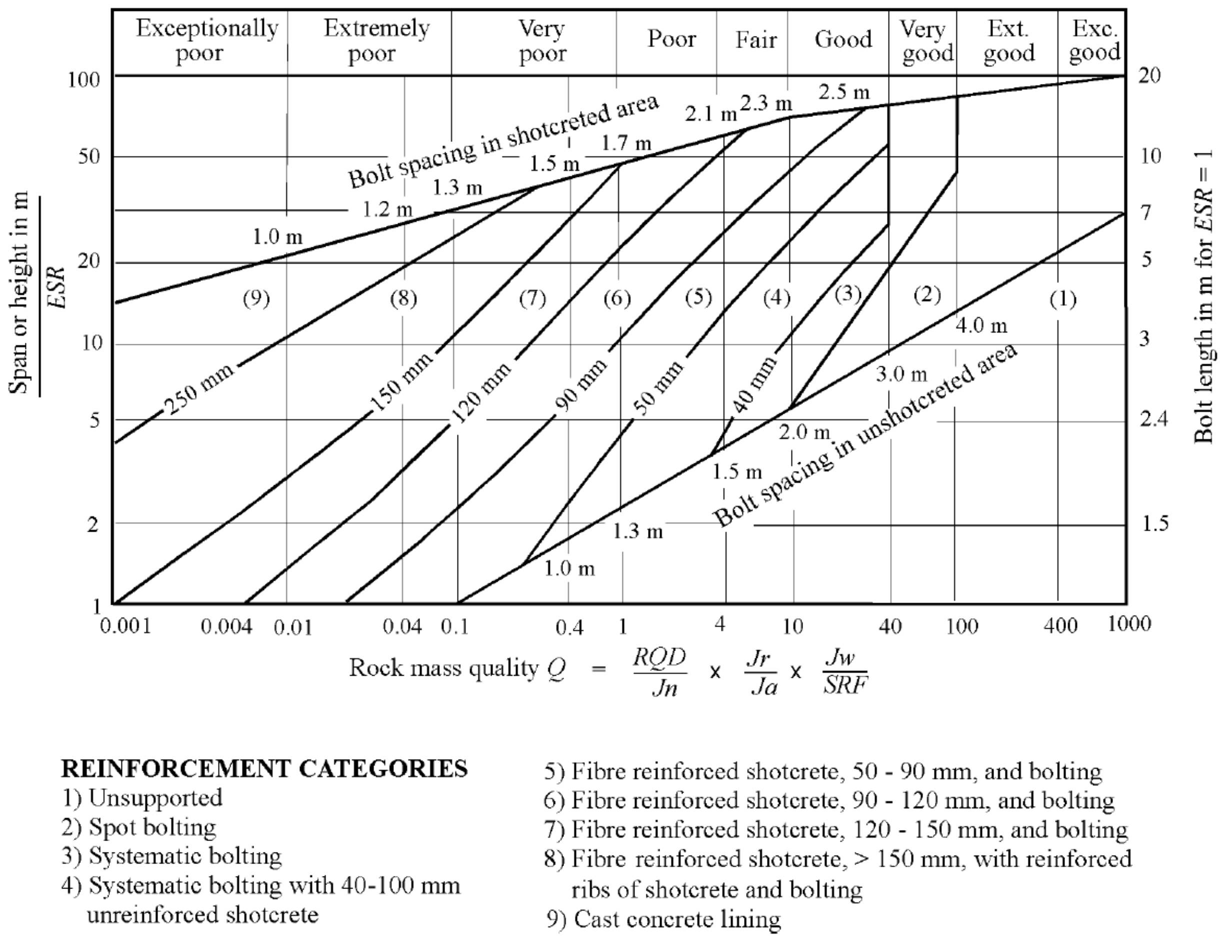

Figure 22: Estimated support categories based on Q (Grimstad and Barton, 1993)

2.4.2 Equivalent Linear Overbreak/Slough (ELOS)

Clark (1998) was able to quantify the potential degree of sloughage from a stope using the modified stability number, N' and the hydraulic radius (Figure 23).

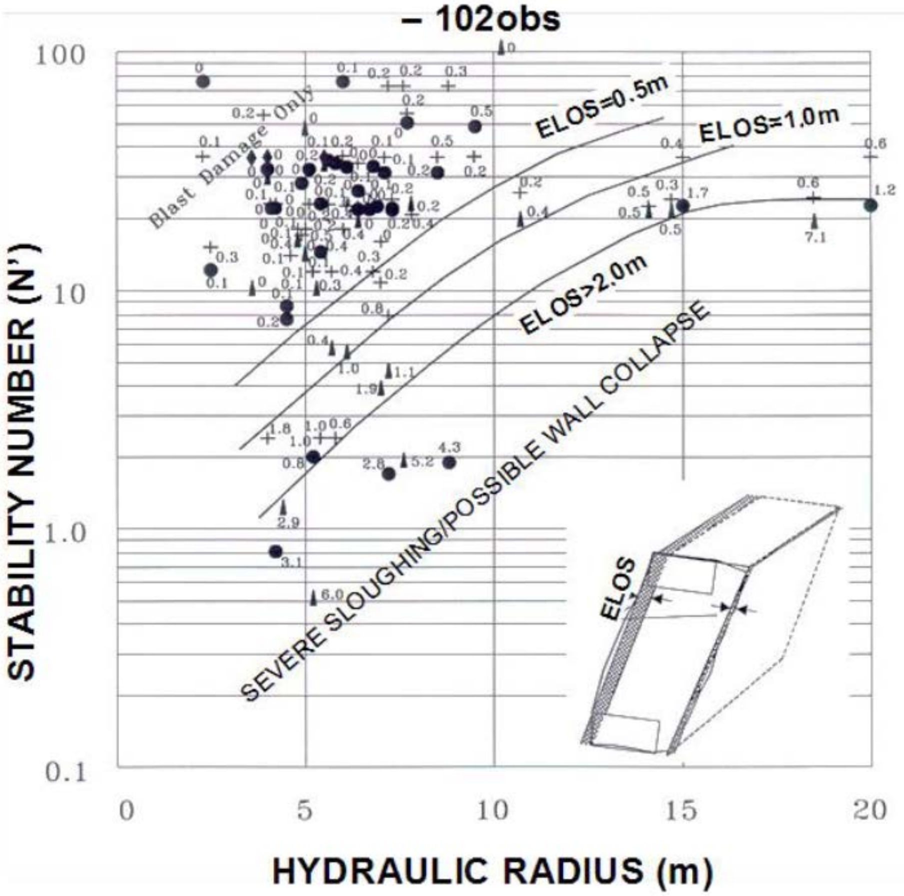

Figure 23: Empirical estimation of wall slough (ELOS) (Clark, 1998)

2.4.3 Pillar Stability Graph

The pillar stability graph was developed using 178 stability cases by Lunder (1994). Two main parameters are used to determine overall stability of the pillars:

- Geometric parameter that represents the pillar shape; pillar width/height ratio (Wp/Hp)
- Strength parameter that represents the pillar load versus the intact rock strength (Average pillar load/UCS)

Plotting these two parameters creates a chart (Figure 24) in which a pillar can be classified as failed, unstable or stable.

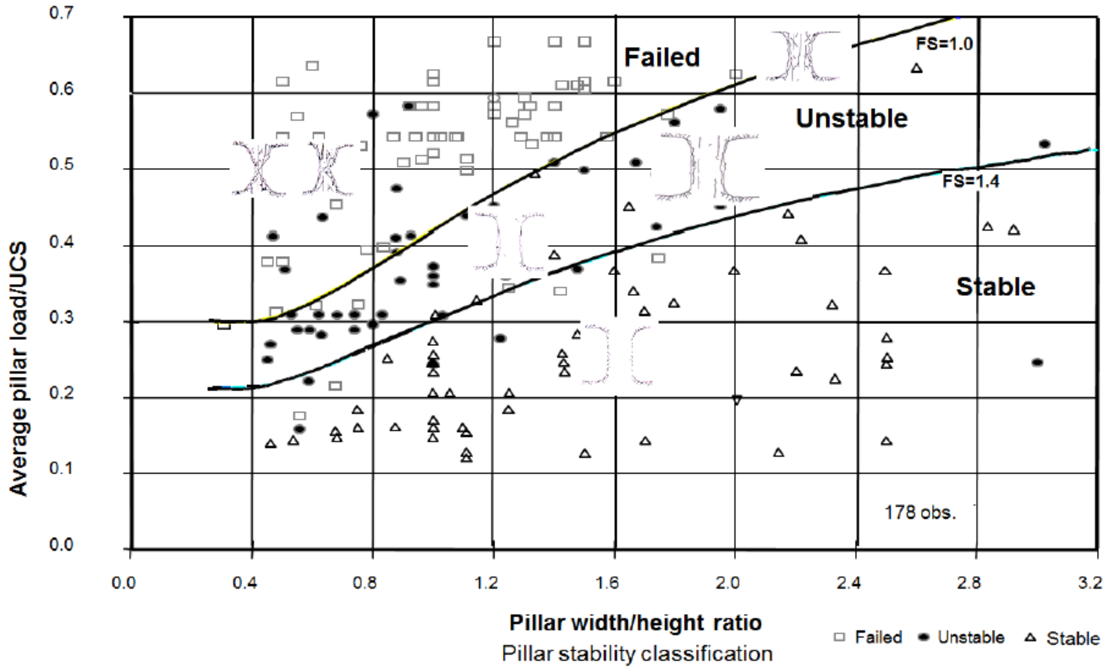

Figure 24: Pillar Stability Graph (Lunder, 1994)

2.4.4 Critical Span Curve

Originally developed by Lang (1994) and updated by Wang et al. (2002), the critical span curve (Figure 25) is an empirical chart used to evaluate back stability using the RMR classification system (Bieniawski, 1976) and the critical span. The critical span is defined as the diameter of the largest circle that can be drawn within the boundaries of an exposed back. The design span refers to backs with no support and/or localized pattern bolting. Local support is viewed as confining blocks that may loosen/fall during subsequent mining. Excavation stability is classified into three categories (Brady et. al., 2003):

- Stable excavation: (a) no uncontrolled falls of ground, (b) no movement of the back is observed, (c) no extraordinary support measures have been employed.
- Potentially unstable excavation: (a) extra ground support has been installed to prevent potential falls of ground, (b) movement has occurred in the back, and (c) increase frequency of ground movement has been observed.

- Unstable excavation: (a) area has collapsed. (b) depth of failure of the back is 0-5 times the span (in the absence of major structures). and (c) support was not effective in maintaining stability.

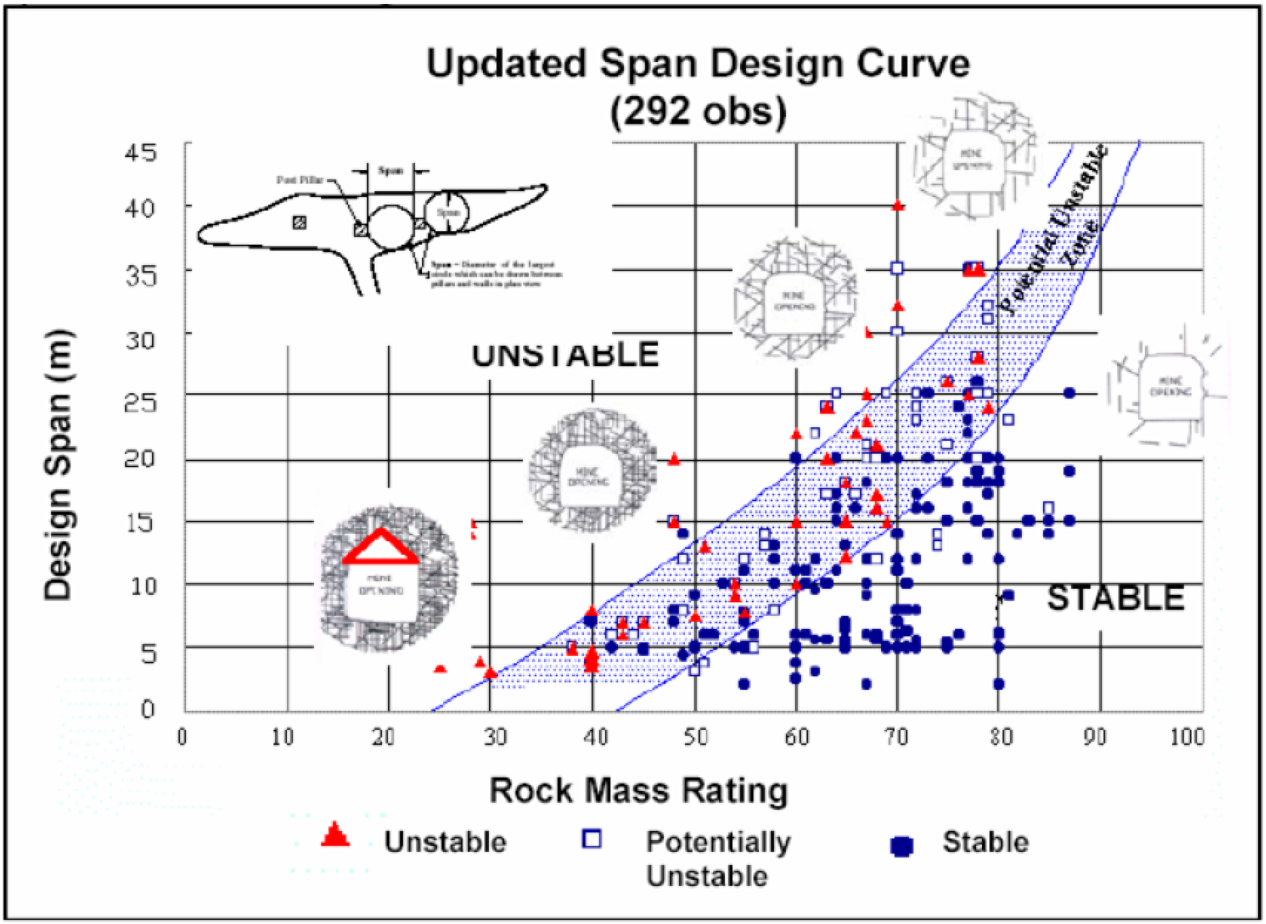

Figure 25: Updated Span Design Curve (Wang et. al, 2002)

Upon review it was felt that these empirical design tools provide a level of information more suited to detailed mine design than that which is required for a preliminary assessment of a potential orebody.

2.5 Production Rate and Cost Estimates

Empirical methods have been developed to estimate production rates and costs based on mining method and orebody characteristics.

2.5.1 Taylor's Rule

Developed in 1977 from 30 case studies, Taylor derived an empirical relationship between the overall reserve tonnage of a mine and the average daily production (Equation 7).

Equation 7: Taylor's Equation (Taylor, 1977)

$$Production\ \left(\frac{mt}{day}\right) = \ 0.0143\ x\ Tonnage^{0.75}$$

It is noted that Taylor's equation was not suitable for deposits with more than 200 Mt reserve tonnage, very deep, or flat ore bodies. The equation also assumes a 350-day operating year.

Singer, Menzie and Long (2000) reviewed 28 case studies of underground mines with massive sulphide deposits and developed alternative coefficients for Taylor's equation (Equation 8).

Equation 8: Taylor's Equation (Singer, et. al., 2000)

$$Production\ \left(\frac{mt}{day}\right) = \ 0.0248\ x\ Tonnage^{0.704}$$

Long (2009) completed an extensive review of mines to further refine the coefficients of Taylor's equation to reflect both open pit/block cave (Equation 9) and underground operations (Equation 10).

Equation 9: Taylor's Equation-Open Pit/Block Cave Mines (342 cases) (Long, 2009)

$$\boldsymbol{Production\ \left(\frac{mt}{day}\right) = \ 0.123\ x\ Tonnage^{0.649}}$$

Equation 10: Taylor's Equation - Underground Mines (197 cases) (Long, 2009)

$$Production\ \left(\frac{mt}{day}\right) = \ 0.297\ x\ Tonnage^{0.562}$$

Table 22 summarizes the various coefficients for Taylor's Equation.

Table 22: Taylor's Rule Coefficients (Dominski et. al., 2014)

Mine Type	a	b	Source	# Mines
Unknown	0.0143	0.75	Taylor (1986)	~30
Underground - Massive Sulfide	0.0248	0.704	Singer, Menzie, Long (2000)	28
Open Pit/Block Caving - Other	0.123	0.649	Long (2009)	342
Underground - Other	0.297	0.562	Long (2009)	197

2.5.2 Benchmarking

Benchmarking is a method of estimating costs by comparing a proposed development to existing mines using similar mining methods and within similar operating jurisdictions. In Table 23, a summary of relative cost factors for various underground mining methods is presented. The minimum cost for Block Caving is assigned a relative cost factor of 1.0, as it represents the lowest cost underground mining method. The relative cost factors were generated from Atlas Copco's experience with operating mines across the globe.

Table 23: Relative Operating Cost Per Mining Method (Atlas Copco, 2014)

Mining Method	Relative Cost Factor (Min)	Relative Cost Factor (Max)
Room and Pillar	7	20
Sublevel Stoping	7	25
Cut and Fill	20	70
Shrinkage Stoping	20	50
Vertical Crater Retreat	7	25
Longwall mining	7	20
Sublevel Caving	5	15
Block Caving	1	2.5

The System for Electronic Document Analysis and Retrieval (SEDAR) is a filing system developed for the Canadian Securities Administrators to:

- facilitate the electronic filing of securities information as required by Canadian Securities Administrator,
- allow for the public dissemination of Canadian securities information collected in the securities filing process, and
- provide electronic communication between electronic filers, agents and the Canadian Securities Administrator.

Technical filings from the SEDAR database were extracted to generate Table 24. The information in Table 24 provides recent and relevant production rate and cost information from existing mines using different mining methods.

Table 24: Comparative Mining Costs (SEDAR, 2019)

Mine	**Location**	**Method**	**Tonnes Per Day**	**Mining Cost (per tonne)**	**Milling (cost per tonne)**	**G&A (cost per tonne)**	**Other (cost per tonne)**	**Total (cost per tonne)**	**Ratio (Method to OP)**
Harte - Sugar Zone	Ontario	Sublevel Stoping	783	$ 100.00	$ 34.00	$ 33.00	$ 17.00	**$ 184.00**	10
Pretium - Brucejack	BC	Sublevel Stoping	3800	$ 98.23	$ 28.87	$ 46.91	$ 47.77	**$ 221.79**	12
TMAC - Hope Bay	Nunavut	Sublevel Stoping	2000	$ 89.04	$ 33.60	$ 35.10	$ 9.94	**$ 167.68**	9
New Gold - New Afton	BC	Block Cave	13000	$ 7.92	$ 11.37	$ 3.56	$ 0.05	**$ 22.90**	1
SSR Mining - Seabee	Sask	Sublevel Stoping	1050	$ 89.76	$ 27.72	$ 79.20		**$ 196.68**	10
IAMGOLD - Cote Gold Project	Ontario	Open Pit	36000	$ 8.88	$ 8.34	$ 1.94		**$ 19.17**	1
IAMGOLD - Westwood	Quebec	Sublevel Stoping	2425	$ 143.13	$ 24.48	$ 18.51		**$ 186.12**	10
Hecla - Casa Berardi	Quebec	Sublevel Stoping	2000	$136.80	$20.70			**$ 157.50**	8
Hecla - Green Creeks	Alaska	Cut and Fill	2000	$ 106.22	$ 49.57	$ 41.82	$ 40.85	**$ 238.46**	12
Hecla - Fire Creek	Nevada	Cut and Fill	160	$ 275.88	$ 148.10			**$ 423.98**	22
Hecla - Fire Creek	Nevada	Sublevel Stoping	100	$ 254.10	$ 148.10			**$ 402.20**	21

Chapter 3

Proposed Deposit Evaluation Procedure

The proposed deposit evaluation procedure consists of three components, illustrated in Figure 26:

1. characterization of the orebody and selection of technically feasible mining methods,
2. determination of production rates, costs (operating), and mine life based on technically feasible mining methods, and
3. a simplified cashflow analysis for each technically feasible mining method and selection of a preferred mining method for detailed design.

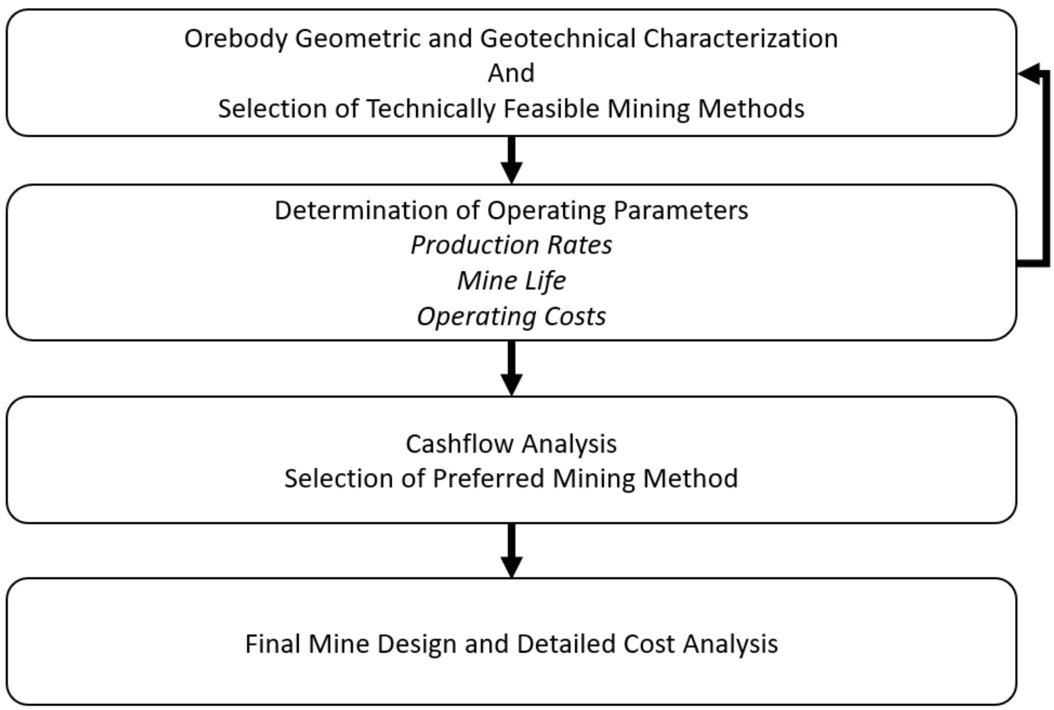

Figure 26: Schematic of Proposed Deposit Evaluation Procedure

3.1 Characterization of Orebody and Selection of Preferred Mining Methods

The first step of the procedure is to define the geometric and geotechnical characteristics of the deposit. The characteristics are entered into the UBC Mining Method Selection Tool to aid the user in identifying technical feasible mining methods.

The UBC Mining Method Selection Tool, introduced in Section 2.3.4, is used based on the following considerations:

- Improved consideration for mechanized mining methods utilized in the developed world
- Use of readily available data
- More detailed factors affecting the selection of mining methods
- Use of numerical ranking values applied to various parameters for different mining methods
- Use of a recognized rock mass classification system (RMR)
- Availability of online software

Section 2.3.4 details the values assigned from each component to a particular mining method. For ease of use, the online tool (http://www.edumine.com/xtoolkit/xmethod/miningmethodgraphic.htm) is used for this analysis.

3.1.1 Estimation of Production Rates

With the technically feasible mining methods identified, the next step is to determine the potential economics of the deposit. The first step is to define a potential reserve of the deposit using recovery and dilution factors related to the selected mining methods. Table 25 summarizes recoveries and dilution factors for various underground mining methods.

Table 25: Mining Method Parameters Chart (Hartman, 1992)

Mining Method	Recovery	Dilution
Room and Pillar	60-80%	10-20%
Sublevel Stoping	60-80%	10-20%
Cut and Fill	90-100%	5-10%
Shrinkage Stoping	70-85%	<10%
Vertical Crater Retreat	60-80%	10-20%
Longwall mining	70-90%	10-20%
Sublevel Caving	80-90%	10-35%
Block Caving	90-125%	10-20%

Reserves are calculated as follows:

$$Reserves = (Resource\ x\ Recovery)\ x\ (1 + Dilution)$$

Grade for reserves is calculated as follows:

$$Grade = Grade_{Resource}\ x\ \frac{(Resource\ x\ Recovery)}{Reserves}$$

For simplicity, it is assumed that any dilution will have no grade. However, this will depend on the grade distribution of the individual deposit.

Once reserves have been calculated for the selected mining methods, production rates are estimated using Taylor's Equation (Section 2.5). Taylor's equation and mining method coefficients are outlined below.

$$Production\ \left(\frac{mt}{day}\right) = \ a\ x\ Tonnage^{b}$$

Table 26: Updated Coefficients for Taylors Equation (Long, 2009)

Mine Type	a	b
Open Pit/Block Caving - Other	0.123	0.649
Underground - Other	0.297	0.563

Mine life is estimated by the following:

$$Mine\ Life\ (years) = Reserves(mt)\ /\ \ (Production\ (tpd)\ x\ Operating\ Days\ per\ year)$$

3.1.2 Estimation of Operating Costs

Using the SEDAR database (2.5.2), 11 mines were reviewed for mining method and corresponding operating cost (Table 24). A base operating cost of $21/tonne is used which was derived from the average costs associated with the block cave and open pit mines reviewed in Table 24.

Using the base operating cost, a range of operating costs is derived for each technically feasible mining method using the relative costs reported in Table 23.

3.4 Preliminary Cash Flow Analysis

Using the production estimates from Section 3.2 and preliminary production costs from Section 3.3 annual cashflows are estimated for each mining method judged to be technically feasible (Section 3.1). Analysis of these annual cashflows allows identification of a preferred mining method for use in preparing a final mine design and detailed cost estimates.

3.5 Applicability and Limitations

3.5.1 Applicability

This is meant as a tool for prefeasibility assessment of the development of deposits within an active mine site (brownfield development). As such, assumptions have been made regarding capital requirements. As many of the capital costs such of construction of mill and support infrastructure, equipment purchases and initial employee training has already been accrued, the sustaining capital is assumed to be captured in the overall mining cost.

3.5.2 Limitations

3.5.2.1 Characterization of the Orebody

Nicholas Mining Method Selection Tool ranks methods using a qualitative ranking of various ore body characteristics based on pre-1981 mining practices. The UBC Mining Method Selection updates Nicholas' qualitative rankings based on observed mining practices in Canada pre-1995. This data is now 20 plus years old and does not fully account for the mining industries shift towards increased daily production rates though the use of larger, automated mining equipment. This increase in daily production rates has allowed the development of lower grade deposits be reducing unit operating cost.

There have also been changes to the regulatory environment in which current mines need to operate including more stringent environmental conditions and the need to actively engage First Nations and other third party stakeholders in the planning process. The UBC Method does not consider non-technical considerations in determining the feasibility of a particular mining method. The UBC Mining Selection

Tool adjusts the rankings from Nicholas to increase focus on hard rock stoping methods (i.e. sublevel stoping, cut and fill, shrinkage and room and pillar.). The authors of the UBC Mining Method Selection Tool felt that caving methods required significantly more detailed analysis due to the complexity of the mining method and hence would not be appropriate to select caving methods using the tool alone.

The selection method also assumes there is sufficient geological and geotechnical information to define characteristic of the deposit. This includes sufficient drill holes to define grade, shape and geotechnical inputs.

Both selection tools make use of Bieniawski's Rock Mass Rating which database is largely (>90%) focused on depths below 500m though there are cases at depths of 2000m or more. As such, the use of RMR at lower depths has less confidence.

The selection tools are set up to provide a range of potentially appropriate mining methods. "It is intended to indicate those methods that will be most effective given the geometry/grade distribution and rock mechanics characteristics, and which will require more detailed study" (Nicholas, 1981).

3.5.2.2 Production Rates and Costs

Production rates in the evaluation procedure make use of Taylor's equation which was initially developed in 1986 with refinements in 2000 and 2009. As such, the data used with the most recent coefficients are 10 years old. The data used to determine these coefficients will be biased towards historical mining practices and unlikely accounts for a larger and automated mining equipment.

The use of the relative cost table (Table 23) provides a wide range of possible mining costs associated with a mining method. The relative cost table does not stratify for the size of the operation or the value of the ore produced. However, larger operations will tend to have a lower per unit operating cost but a greater capital investment. A small number of mines where used for the benchmarking with only an open it and block cave mine sued to determine the base cost of $21/tonne. Using unit costs from the SEDAR database may have inconsistencies due to how mines report their costs. A larger database stratified by mining cost and capacity would allow for refinement of the mining cost range. Also, as detailed designs have not been done at this stage, a wide range of potential costs needs to be evaluated to determine the potential risk of the deposit.

3.5.2.3 Cashflow

In the evaluation procedure capital costs have not been looked at as the deposit is considered to be within a brownfield operation with the initial capital costs being sunk. Operating costs are assumed to include all remaining incremental capital costs to support the development as well as production, haulage, milling, technical and administrative costs. The costs are assumed to be directly proportional to production rates. It is assumed that production is constant year to year and prorated for part years. Revenues assume a constant price of gold and a fixed foreign exchange rate. The grade of the deposit is also assumed to be constant throughout the mine life and that the predicted grade is indeed the realized grade. These assumptions allow for constant revenue and cost projection. In reality these factors can fluctuate from year to year. Inconsistent grade and rock type can also impact mill recoveries which is not taken into account for this evaluation procedure as the level of detailed analysis would be covered in a feasibility study. The wide variety in relative cost may not select a clear winner but allows for the identification of the most promising methods for more detailed analysis.

Chapter 4

Case Study – Bonanza Ledge Mine

4.1 Introduction

Barkerville Gold Mines Limited is a gold focused exploration and mining company based in Wells, British Columbia (Figure 27). Brownfield exploration activities are focused on further defining the resource for the Cariboo Gold project on Cow and Island Mountains. The operations team is focused on the Bonanza Ledge and BC Vein Deposits on Barkerville Mountain (Figure 28). Mining on the Bonanza Ledge Deposit was completed in 2018 with development for the BC Vein Deposit currently under construction. Production from the BC Vein Deposit is planned to start in 2020.

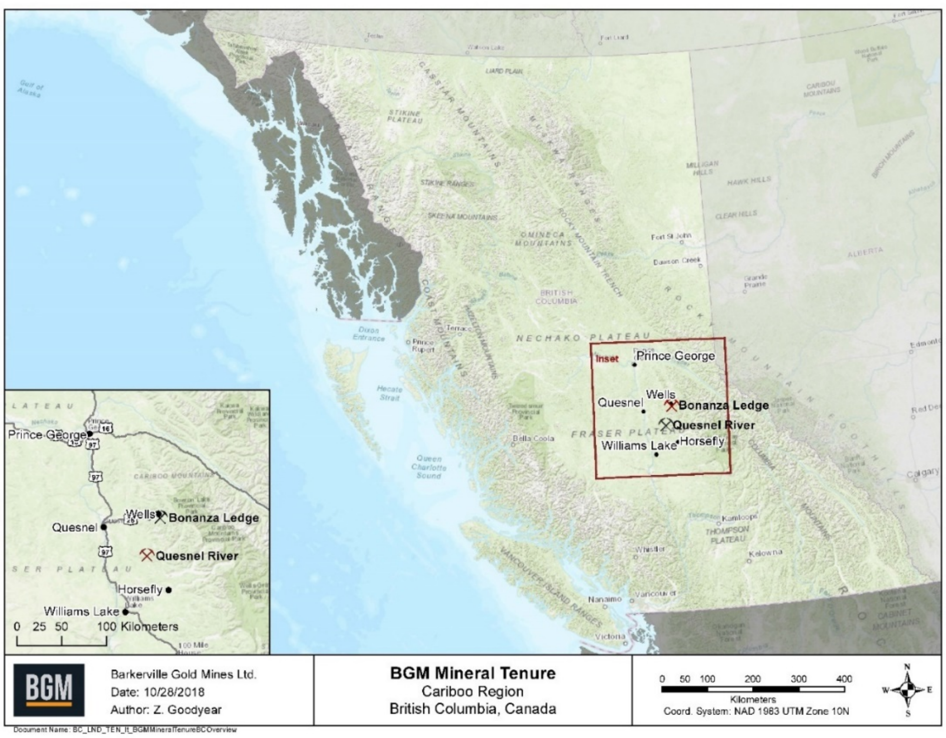

Figure 27: BGM Location Map (BGM, 2018)

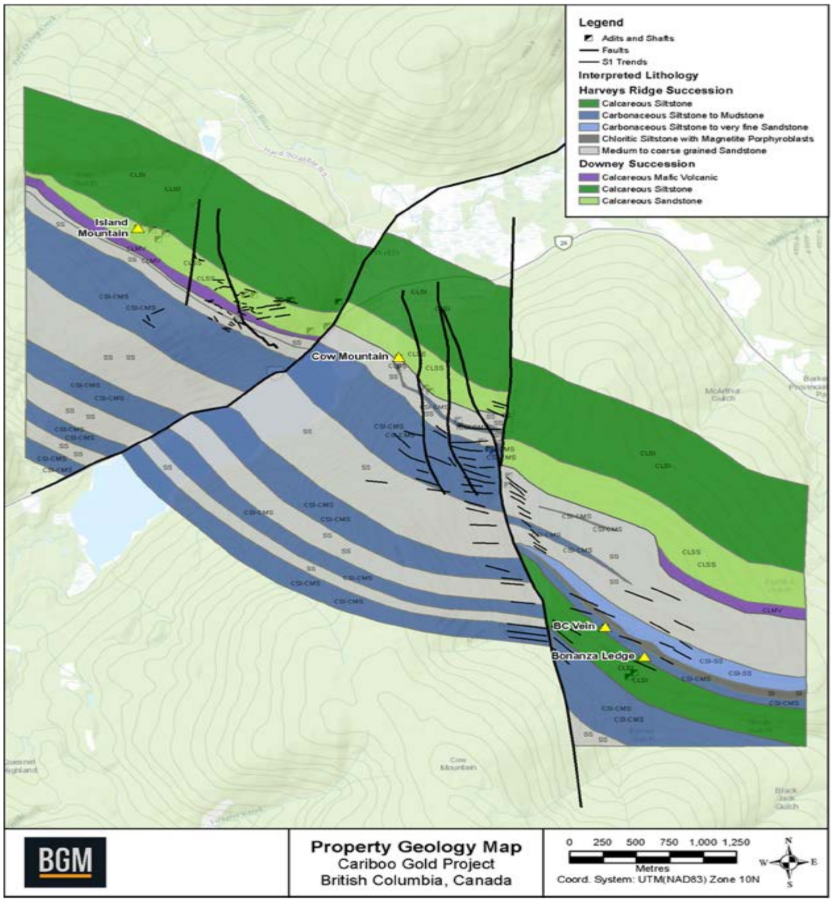

Figure 28: BGM Property Geology Map

Similar to the case for the Bonanza Ledge deposit, ore from the BC Vein Deposit will be processed at the QR mill, located approximately 110 kilometers from Wells.

4.2 Background

Mining activity in the area dates to the Cariboo Gold Rush of the early 1860s and includes extensive placer mining and lode (hard rock) mining (BGM, 2018).

4.2.1 Placer and Hardrock Mining

The Bonanza Ledge (BL) Deposit is located on the ridge that divides Lowhee Creek and Stouts Gulch. Extensive placer mining occurred in both valleys from the early 1890s to the early 1920s.

From the late 1930s to the early 1940s, hard rock mining of the BC Vein Deposit continued near the Bonanza Ledge Deposit and, in the late 1940s, the Canusa Mine was developed.

4.2.2 Bonanza Ledge Deposit Discovery

From the mid to late 1990s, BGM (and its predecessors) continued exploration drilling in the area and in March 2000, mineralization (Bonanza Ledge) was discovered below the BC Vein Deposit. From 2003 to 2004, a 7000-t bulk sample was extracted.

4.2.3 Bonanza Ledge Mine – 2010 to 2015

After reviewing the bulk sample results, BGM initiated baseline studies for a permit application for an open pit mine and subsequently submitted a joint application in late 2010. The 2010 application proposed extraction of approximately 200 tonnes per day (t/d), with an annual mining rate of 74 000 t/y over a four-year mine life (for a total of 296,000 t), as a two-phase build out: a starter pit and an ultimate pit. On December 5, 2011, the Ministry of Energy, Mines and Petroleum Resources (MEMPR) approved the application and issued the first mine permit for Bonanza Ledge.

Approximately 101 000 t of ore was extracted between June 2014 and March 2015.

4.2.4 Bonanza Ledge Mine - 2015 to Present

In 2015, BGM reviewed the approach for the mine and developed an underground mining strategy. The revised underground strategy was submitted to MEMPR in the Mines Act Permit Amendment (MAPA) 2016 Application. The revised strategy reduced the Mine Footprint Area (MFA) from that proposed in the 2010 Application, increased the permitted volume mined per year to 150,000 t and changed the method of extraction from open pit to underground mining.

The amended permit was issued to BGM on March 29, 2017 and construction began in May 2017. Production on BL Phase I began in March 2018 and was completed in December 2018. Approximately 150,000 t of ore was extracted from BL Phase 1.

Currently, access is being developed to the second deposit within the Bonanza Ledge Mine. This second deposit is referred to as the BC Vein or the Phase 2 development. Meanwhile, drilling continues to further define the extent of the much larger, third deposit, Cariboo Gold (BGM, 2018).

The evaluation procedure developed in Section 3 is applied to the physical geology and geotechnical data for the recently completed Bonanza Ledge development. The results obtained from the evaluation procedure are then compared to actual results. This comparison allows confirmation of the appropriateness of the proposed evaluation procedure. Variations between predicted results and actual results are discussed.

4.3 Geology

Bonanza Ledge is a replacement mineralization hosted within calcareous siltstone. It consists of fine-grained pyrite ore only. Sulphide content in the replacement ore is generally high, from 10% replacing thin calcareous bands to massive, largely replacing entire beds completely. The Bonanza Ledge replacement is fault-bounded in the footwall of the BC Vein structure (Figure 29).

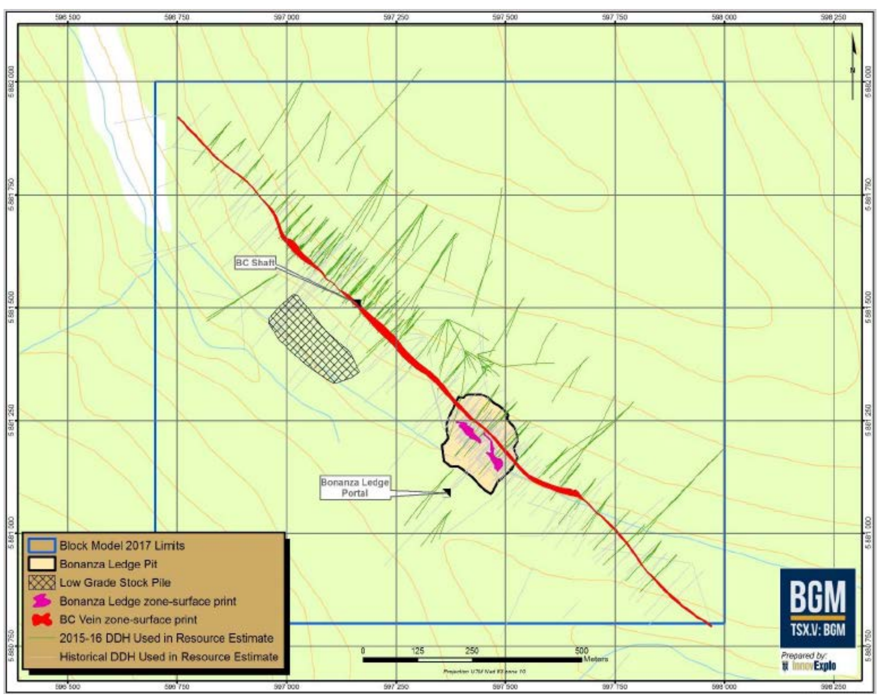

Figure 29: BC Vein and Bonanza Ledge Deposits (Barkerville Gold, 2018)

4.3.1 Bonanza Ledge Resource

Initial resource modeling for the Bonanza Ledge Deposit estimated that the deposit contained 264,000t with an average grade of 7.30 g/t (Figure 30). A 3g/t cut-off was used as part of the resource estimate.

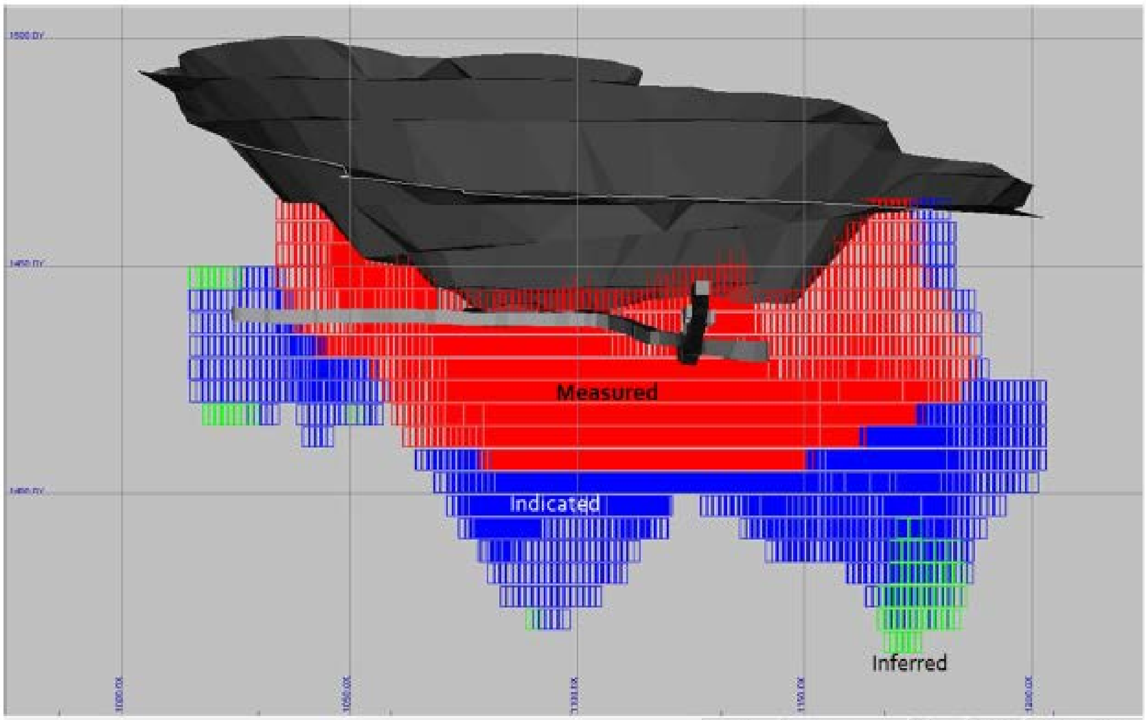

Figure 30: Bonanza Ledge Resource (Barkerville Gold, 2018)

4.3.2 Geometry

Using the UBC Mining Method Selection definitions for geometry, the Bonanza Ledge Deposit was determined to have the following characteristics:

- General Shape: Irregular. Deposit is ~120m in strike length and 40m in depth below the previous pit,
- Ore Thickness: Ranges from 10m up to 30m,
- Plunge: Between 70° and 90°,
- The mine plan included the removal of the crown pillar between the pit and the lower extents of the deposit,
- Grade Distribution: Erratic. Grade fluctuates throughout deposit.

4.4 Geotechnical

Geotechnical drillholes were completed and logged by Golder Associates in 2016 and reported in 2017. The results are displayed in Table 27. In general, the Bonanza Ledge Deposit rock mass quality is classified as fair (Bieniawski's, 1989). Lab testing included two uniaxial compressive strength tests and two triaxial compressive strength tests performed on each lithologic unit. The overall intact rock strength is weak, however, given the low stress environment in which mining will take place, pillar stability was not be a significant concern.

Table 27: Bonanza Ledge Geotechnical Properties (Golder, 2017b)

Parameter	HW	ORE	FW
RMR_{76} 25th Percentile	44	44	44
RMR_{76} 75th Percentile	61	61	61
RMR_{76} Average	52	52	52
Q 25th Percentile	4	4	4
Q 75th Percentile	20.7	20.7	20.7
Q Average	15	15	15
UCS (MPa)	34	34	34
E (GPa)	50.2	50.2	50.2
ν	0.29	0.29	0.29
Design Unit Weight (tonnes/m^3)	3.0	3.0	3.0
Rock Substance Strength (UCS/σ_v)	38.5	38.5	38.5

In situ stresses are assumed to be hydrostatic ($\sigma_v = \sigma_H$) in the area based on Golder's previous experience and knowledge of the Cariboo Area (Golder, 2017b). This has not been confirmed with local stress measurements.

4.5 Mining Method Selection

Using the UBC Mining Method selection tool (Edumine, 2019), the geometric, grade distribution and geotechnical data was analyzed (Figure 31: Bonanza Ledge Mining Method Selection (Edumine, 2019)).

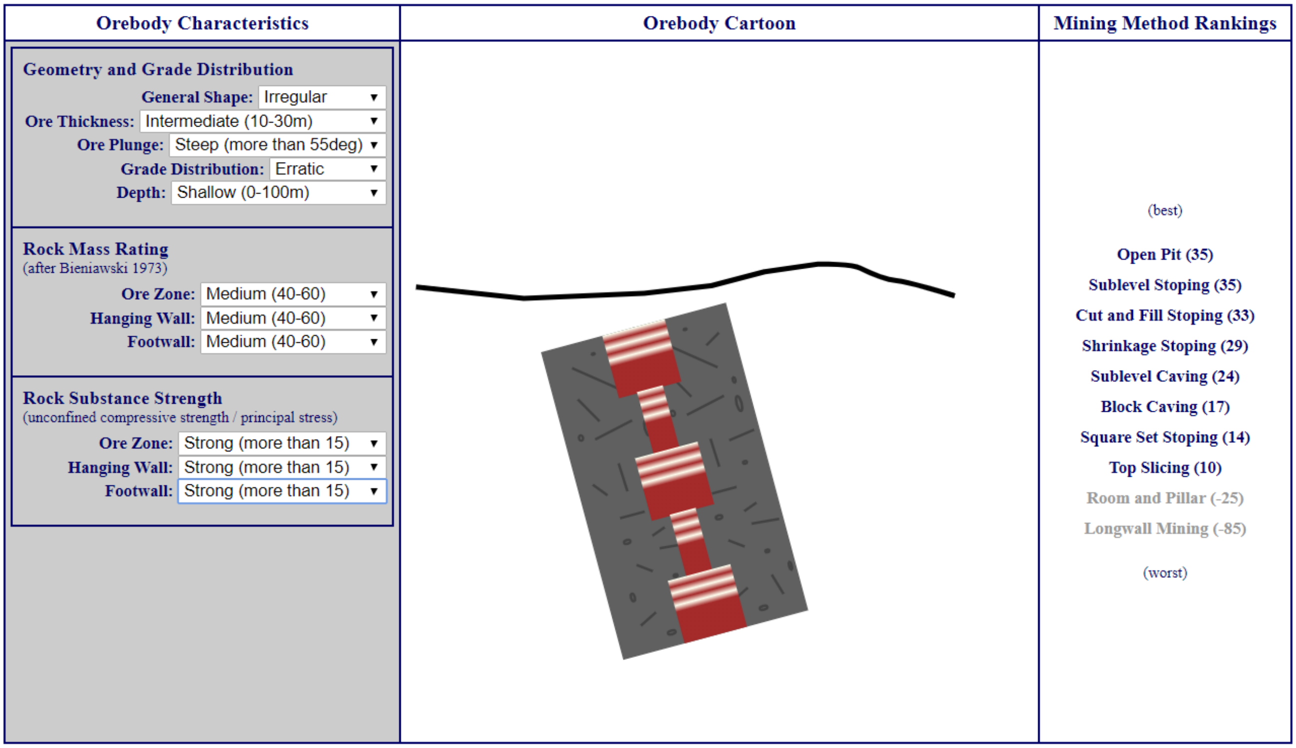

Figure 31: Bonanza Ledge Mining Method Selection (Edumine, 2019)

Based on application of the UBC model, the preferred methods are as follows:

1. Open Pit (35)
2. Sublevel (35)
3. Cut and Fill (33)
4. Shrinkage Stoping (29)

The open pit option was eliminated as the current mine permit is for underground mining only. At this point in time, the company was not willing to make a significant change to the mine permit. This is due to a number of reasons including significant public/first nations consultations, additional time-consuming environmental studies and the distraction from the strategic focus of building knowledge to for flagship Cariboo Gold deposits.

4.6 Production Rates and Costs

Applying the variables for 'Underground-Other' from Table 22 to Taylor's Equation and using the Table 25 inputs, the estimated reserves and production rates are as follows:

Table 28: Bonanza Ledge Production Estimates

Mining Method	Estimated Recovery	Estimated Dilution	Estimated Tonnage	Estimated Grade (g/t)	Estimated Production Rate (tonnes/day)	Estimated Mine Life (years)
Sublevel Stoping	60-80%	10-20%	176,000 – 264,000	5.84-6.57	263 – 331	1.8 – 2.2
Cut and Fill	90-100%	5-10%	250,105 – 293,333	6.57-6.94	321 – 351	2.1 – 2.3
Shrinkage Stoping	70-85%	0-10%	184,800 – 249,333	6.57-7.30	271 - 320	1.9 – 2.1

4.7 Estimated Cashflow

Using the base operating cost of $21/tonne and the relative cost data from Table 23, a range is calculated for the overall operating cost per mining method in the following table. Details are presented in Appendix A.

Table 29: Projected Operating Costs

Mining Method	Min Relative Cost Factor	Max Relative Cost Factor	Min Projected Operating Cost ($/tonne)	Max Projected Operating Cost ($/tonne)
Room and Pillar	7	20	147	420
Sublevel Stoping	7	25	147	525
Cut and Fill	20	70	420	1470
Shrinkage Stoping	20	50	420	1050
Vertical Crater Retreat	7	25	147	525
Longwall mining	7	20	147	420
Sublevel caving	5	15	105	315
Block Caving	1	2.5	21	52.5

Using the project operating costs from Table 29, a cashflow table was generated for the selected mining methods to determine the optimal cashflow for the deposit.

Table 30: Estimated Cashflows for Bonanza Ledge

Min Cashflow			
	Y1	**Y2**	**Y3**
Sublevel Stoping	$(36,195,906.75)	$(36,195,906.75)	$(6,725,706.75)
Cut and Fill	$(149,396,697.48)	$(149,396,697.48)	$(43,182,885.43)
Shrinkage Stoping	$(90,255,382.09)	$(90,255,382.09)	$(11,893,009.45)
Max Cashflow			
	Y1	**Y2**	**Y3**
Sublevel Stoping	$29,198,782.39	$24,239,190.60	
Cut and Fill	$(2,767,067.06)	$(2,767,067.06)	$(372,610.18)
Shrinkage Stoping	$3,557,220.31	$3,093,625.99	

*Gold price of $1600 CAD/ Au. Oz used based on 2018 realized gold price.

Based on the estimated cashflows from Table 30, only the sublevel stoping and shrinkage stoping methods would generate positive cashflow under optimal conditions. This allows for the elimination of Cut and Fill for further consideration. Sublevel stoping appears to offer the most attractive returns and should be considered for more detailed analysis.

4.8 Comparison of Projections to Actuals

In general, the projections from the evaluation procedure were comparable to the realized values (Table 31).

The predicted mining method of sublevel stoping from the evaluation procedure was the actual implemented mining method for development of the Bonanza Ledge Mine as recommended to BGM by external consultants. Recovery of the ore body, dilution and operating costs were within the predicted ranges of the evaluation procedure. Though the deposit had an erratic grade distribution, material at the periphery of the deposit generally carried grade, albeit below the desired cut-off, resulting in no dilution coming from outside the resource. Dilution generally resulted from backfill from adjoining stopes.

Table 31: Bonanza Ledge Mine Projected vs Actual

Parameter	Projected	Actual
Mining Method	Sublevel Stoping	Sublevel Stoping
Total Tonnes	176,000 – 264,000	158,173
Grade (g/t)	5.84 – 6.57	5.03
Dilution	10-20%	19%
Recovery	60-80%	60%
Production Rate (tonnes/day)	263 - 331	433
Mine Life	1.8 - 2.2	1
Operating Cost (CAD/t)	147 - 420	260

Total tonnes, grade, production rate and mine life were outside of the predicted values assessed by the evaluation procedure. It is believed that these discrepancies can be explained from some of the decisions that were made at the mine site. During production, the decision to increase daily tonnage was made to ensure a consistent hauling rate from the mine site to the QR mill. This was achieved through a reduction of curing time of the cement backfill from 28 days to 10 days, allowing stopes to become available sooner. While allowing increased rates of production, the reduced curing time resulted in increased dilution and reduced recovery of the deposit due to backfill failures into the adjoining stopes.

During production, it was identified that the deposit had greater variability of grade distribution than assumed in the initial model with certain lithologies being barren rather than having some mineralization. This may have resulted in an overestimation of the initial resource.

Overall, the evaluation procedure has been validated for this deposit with similar geometric and geotechnical inputs.

Chapter 5

BC Vein Evaluation

Development of access to the BC Vein Deposit is currently underway. The orebody evaluation procedure developed in Section 3 was applied to the BC Vein Deposit to determine the preferred mining method and to assess its economic feasibility. Mining of the BC Vein Deposit is considered a brownfield development as it relies on the infrastructure already in place, including surface facilities, the mine portal and the mine mill.

5.1 BC Vein Geology

The gold-bearing BC Vein is the most continuous and voluminous vein of a set of shear veins that run northwest and dip steeply (70°) to the northeast (Figure 32). The BC Vein is controlled by a brittle fault zone with evidence of multiple movements and mineralization pulses within the fault zone.

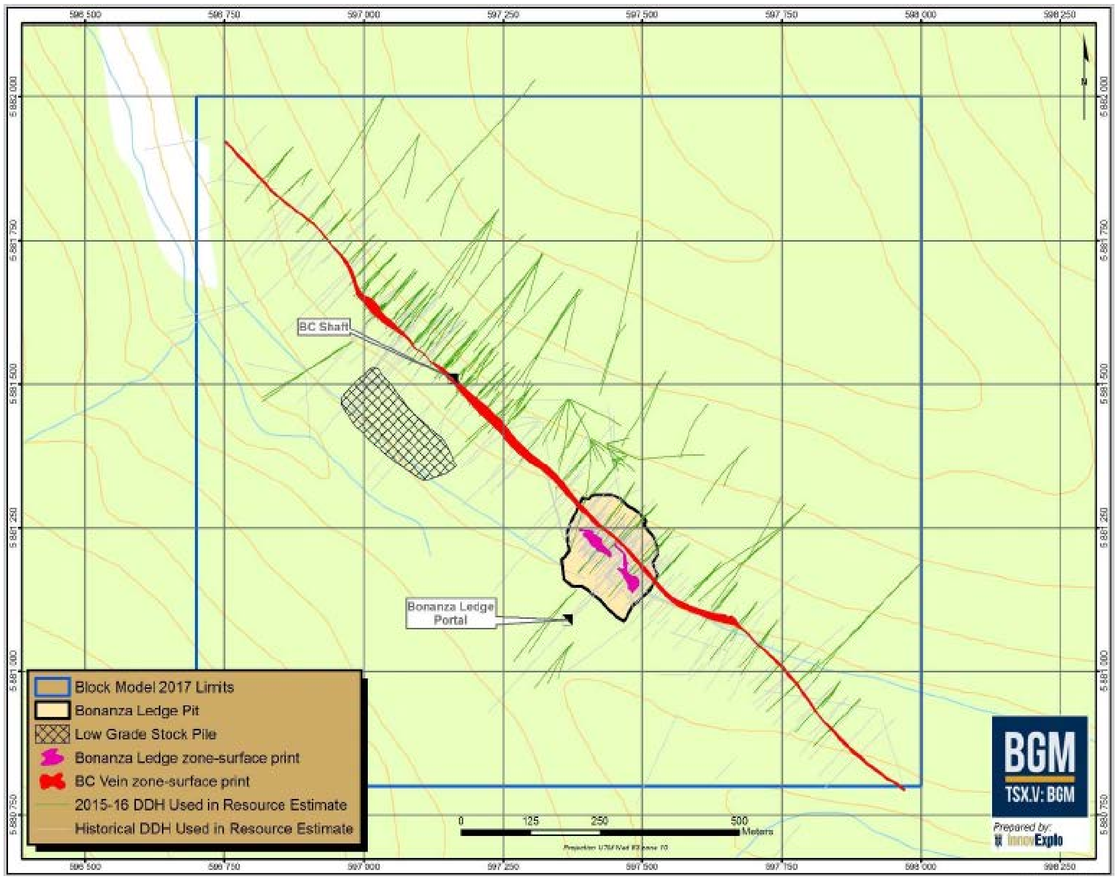

Figure 32: BC Vein Deposit (BGM, 2018)

5.1.1 BC Vein Resource

Initial resource modeling for the BC Vein estimates that the deposit contains 596,903t with a grade of 5.87 g/t (Figure 33). A 3g/t cut-off was used as part of the resource estimate.

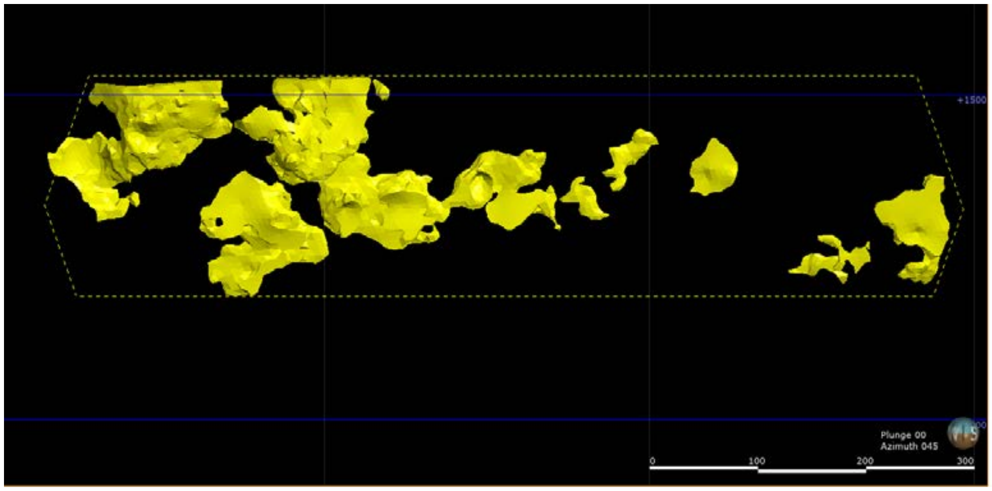

Figure 33: BC Vein Resource-596,903t at 5.87 g/t (3g/t cut-off) (MacQueen, 2019)

5.1.2 Geometry

Using the UBC Mining Method Selection definitions for geometry, the BC Vein Deposit was determined to have the following characteristics (Figure 34):

- General Shape: Platy/Tabular. Deposit is ~800m in strike length and 200m in depth (with current drilling)
- Ore Thickness: Ranges from 3m up to 20m. Average = 8m
- Plunge: Between 65° to 75°
- Depth Below Surface: Deposit outcrops at surface. Mining would be within 100m from surface
- Grade Distribution: Erratic. Grade fluctuates throughout deposit

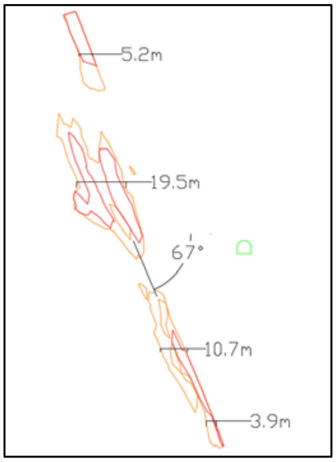

Figure 34: BC Vein Geometry (MacQueen, 2019)

5.2 Geotechnical

Geotechnical logging was completed by BGM geologists in 2017 and interpreted in 2019. The results are displayed in Table 32. As with the Bonanza Ledge Deposit, the rock mass for the BC Vein Deposit is considered to be fair (Bieniawski, 1989) with the intact rock strength of the footwall, hanging wall and ore being weak. With the low stress environment, there are not likely to be any stress related issues until mining reaches the lower depths of the deposit. At this point in time, it is possible that induced stresses could be greater than the intact rock strength of the ore.

Table 32: BC Vein Geotechnical Parameters (Golder, 2017b)

Parameter	HW	ORE	FW
RMR_{76} 25^{th} Percentile	53	30	44
RMR_{76} 75^{th} Percentile	64	57	61
RMR_{76} Average	57	45	52
Q 25^{th} Percentile	4.3	1	4
Q 75^{th} Percentile	20.7	9.7	20.7
Q Average	18	9	15
UCS (MPa)	43	24	34
E (GPa)	32.8	52.8	50.2
v	0.35	0.13	0.29
Design Unit Weight (tonnes/m^3)	2.8	2.8	3.0
Rock Substance Strength (UCS/σ_v)	7.8 - 54	4.4 - 30	5.8 - 37

In situ stresses are assumed to be hydrostatic ($\sigma_v = \sigma_H$) in the area based on Golder's previous experience and knowledge of the Cariboo Area (Golder, 2017b). This has not been confirmed with local stress measurements.

5.3 Mining Method Selection

Using the UBC Mining Method selection tool (Edumine, 2019), the geometric, grade distribution and geotechnical inputs were analyzed (Figure 35).

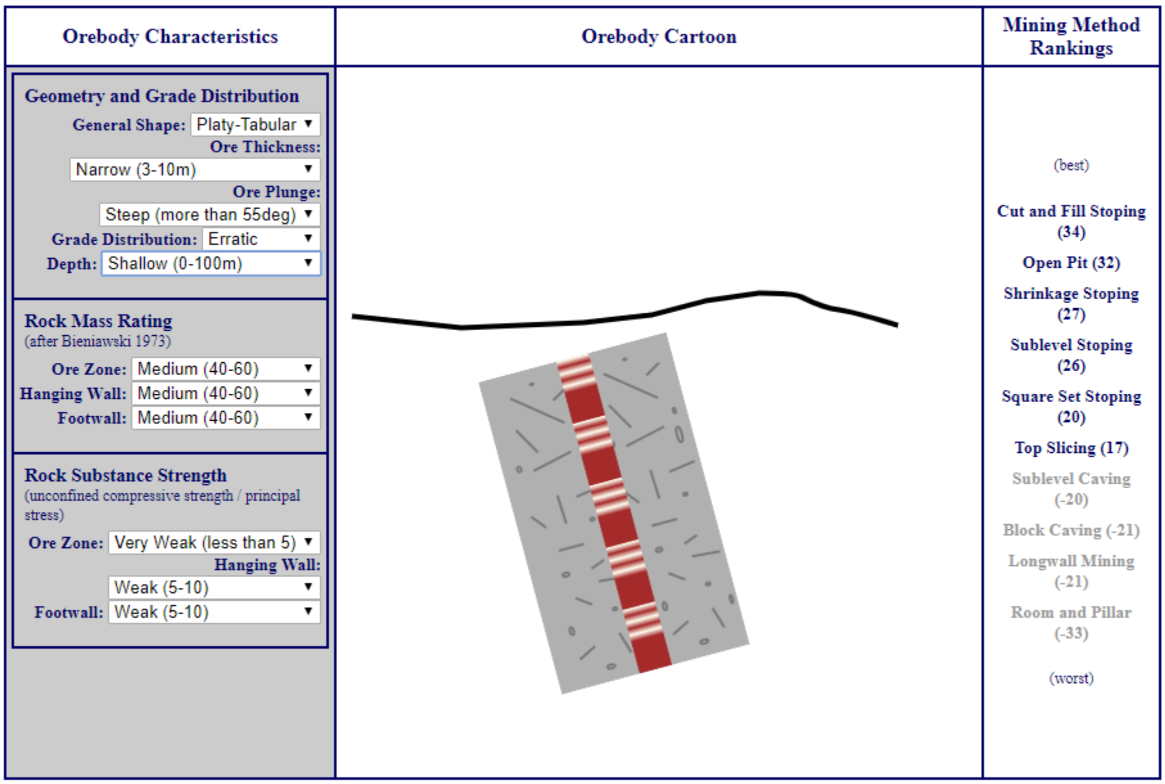

Figure 35: BC Vein Mining Methods Selection (Edumine, 2019)

Based on the inputs, the technically feasible mining methods are:

1. Cut and Fill (34)
2. Open Pit (32)
3. Shrinkage Stoping (27)
4. Sublevel Stoping (26)

The open pit option was eliminated as the current mine permit is for underground mining only. Although application for a revised permit, allowing open pit mining is possible, current public, government agency and First Nations concerns could easily result in significant time delays and costs without granting of the request being a foregone conclusion.

The geometric and geotechnical characteristics are similar to those of Bonanza Ledge Deposit (Figure 31) and hence the procedure is considered to be appropriate for this deposit.

5.4 Production Rates and Costs

Applying the variables for 'Underground-Other' from Table 22 to Taylor's Equation and using the Table 25 inputs, the estimated reserves and production rates are as follows:

Table 33: BC Vein Production Estimates

Mining Method	Estimated Recovery	Estimated Dilution	Estimated Tonnage	Estimated Grade (g/t)	Estimated Production Rate (tonnes/day)	Estimated Mine Life (years)
Sublevel Stoping	60-80%	10-20%	358,142 – 477,522	4.70 - 5.28	416 – 523	2.6 – 3.1
Cut and Fill	90-100%	5-10%	537,213 – 596,903	5.28 - 5.58	507 - 555	3.1 – 3.3
Shrinkage Stoping	70-85%	0-10%	417,832 – 507,368	5.28 – 5.87	428 - 507	2.7 – 3.1

5.5 Estimated Cashflow

Using the project operating costs from Table 33, a cashflow table was generated for the selected mining methods to determine the optimal cashflow for the deposit (Table 34). Details presented in Appendix B.

Table 34: Estimated Cashflows for BC Vein

Min Cashflow				
	Y1	**Y2**	**Y3**	**Y4**
Sublevel Stoping	$(57,028,956.74)	$(57,028,956.74)	$(57,028,956.74)	$(7,106,671.86)
Cut and Fill	$(235,979,915.59)	$(235,979,915.59)	$(235,979,915.59)	$(64,231,020.49)
Shrinkage Stoping	$(142,489,940.50)	$(142,489,940.50)	$(142,489,940.50)	$(6,748,365.08)
Max Cashflow				
	Y1	**Y2**	**Y3**	**Y4**
Sublevel Stoping	$46,535,006.29	$46,535,006.29	$28,674,203.99	
Cut and Fill	$(3,999,252.09)	$(3,999,252.09)	$(3,999,252.09)	$(205,917.63)
Shrinkage Stoping	$5,992,554.42	$5,992,554.42	$4,031,164.13	

*Gold price of $2000 CAD/Au. Oz was used based on projections for 2020.

Based on the estimated cashflows from Table 34, both sublevel stoping and shrinkage stoping would generate positive cashflow under optimal conditions. Sublevel stoping generates the greatest potential cashflow.

Based on the estimated cashflows from Table 34, only the sublevel stoping and shrinkage stoping methods would generate positive cashflow under optimal conditions. This allows for the elimination of Cut and Fill for further consideration. Sublevel stoping appears to offer the most attractive returns and should be considered for more detailed analysis.

5.6 Summary

Applying the proposed evaluation procedure to the Bonanza Ledge Deposit resulted in identification of sublevel stoping as the preferred mining method. This was the method utilized in mining the Bonanza Ledge Deposit. In addition, the predicted recovery, dilution and operating costs were within the predicted ranges of the evaluation. The grade, production rate and mine life were slightly outside the values predicted by the evaluation procedure. These discrepancies are explained by decisions made at the mine site to increase the production rate to provide a steady supply of ore for shipping to the mill. This was achieved through a reduction in curing time of the backfill. The grade model also predicted grade in portions of the orebody that ultimately did not carry any gold as determined by sampling. This also resulted in increased dilution and may have inflated the initial grade of the deposit.

The results of the analysis for the Bonanza Ledge case study provides confidence to the use of this procedure for evaluation of the BC Vein Deposit.

Applying the procedure developed in this thesis results in sublevel stoping being identified as the preferred mining method for the extraction of the BC Vein Deposit. As the deposit is similar in geometric and geotechnical parameters to Bonanza Ledge, selection of sublevel stoping is consistent with the results obtained by applying the procedure to the Bonanza Ledge Deposit.

A detailed technical design is required to refine operating cost estimates and required capital for the mine.

While the Bonanza Ledge case study supports the validity of the proposed evaluation procedure, the examination of additional existing mines would increase confidence in the procedure.

Further refinement of the benchmark data (Table 24) to confirm the values used in this procedure can be obtained from mining operations with similar geometric and geotechnical parameters to Bonanza Ledge. Information obtained from mines that operate under similar regulatory and economic conditions will further enhance the accuracy of the predicted operating costs.

For the BC Vein Deposit, the use of actual costs from Bonanza Ledge falls within the range of the predicted operating costs based on benchmark data from other mines in similar jurisdictions.

Additional opportunities for enhancing benchmark data will result from the completion of the detailed mine design and the ongoing monitoring of the development of the BC Vein Deposit.

In addition, non-technical constraints, including regulatory, environmental, and First Nations concerns must be considered in the final decision to proceed with the project.

Chapter 6

Conclusions and Recommendations

6.1 Conclusions

The objective of this thesis was to develop a procedure for the evaluation of new deposits within a brownfield mining operation. A literature review of underground mining methods, rock mass classification systems and mining method selection tools was conducted to understand which mining methods are applicable to different geometrical and geotechnical characteristics of an orebody. Empirical methods for estimating production rates and mining costs were reviewed to assist in determining potential productivity and operating costs of technically feasible mining methods.

An evaluation procedure was then developed and calibrated based on the information obtained from the literature review. The result of this evaluation procedure was then tested against a recently mined deposit to assess the procedure's predicted results against results obtained from the mined deposit. The results from mined deposit where found to be within the range of the values predicted by the literature review-based procedure for mining cost, recovery and dilution. Grade, total tonnes and mining rate were outside the predicted values, but the minor discrepancies were a result of over aggressive mining and an overly optimistic geological model.

The evaluation procedure developed in this thesis is a three-step process. Each step within the procedure are subject to specific constraints and limitations. The first step is the characterization of the orebody and the selection of technically feasible mining methods. The limitations for this step include:

- UBC Mining Method Selection Tool relies on Nicholas' methodology using qualitative rankings based on observed mining practices in Canada pre 1995,
- UBC Mining Method Selection Tool focuses on hard rock stoping methods (i.e. sublevel stoping, cut and fill, shrinkage and room and pillar). Authors feel that caving methods required significantly

more detailed analysis and hence would not be appropriate to select caving methods using the tool alone.

- Selection method assumes there is sufficient geological and geotechnical information to define characteristic of the deposit. This includes sufficient drill holes to define grade, shape and geotechnical inputs.
- The use of Bieniawski's Rock Mass Rating which database is largely (>90%) focused on depths below 500m though there are cases at depths of 2000m or more. As such, the use of RMR at lower depths has less confidence.
- The selection tools are set up to provide a range of potentially appropriate mining methods. "It is intended to indicate those methods that will be most effective given the geometry/grade distribution and rock mechanics characteristics, and which will require more detailed study" (Nicholas, 1981).

These limitations in the step constrain the application to deposits which are suited for hard rock stoping methods with depths less than 500m. Application of the procedure to deposits in depth greater than 500m must be done with caution as the data used on developing the Rock Mass Rating is limited for depths greater than 500m.

Step two is the determination of the reserves of the deposit (recovery and dilution factors), mining rate and cost based on potential mining methods. The limitations for this step include:

- Taylor's equation was initially developed in 1986 with refinements in 2000 and 2009. As such, the data used with the current coefficients is 10 years old. This likely does not account for a larger, automated and mechanized mining equipment and as such, the coefficients would benefit from additional updating,
- The use of the relative cost table (Table 23) provides a wide range of possible mining costs associated with a mining method. The relative cost table does not stratify for the size of the operation or the value of the ore produced. However, larger operations will tend to have a lower per unit operating cost but a greater capital investment,

- A small number of mines where used for the benchmarking with only an open pit mine and block cave mine used to determine the base cost of $21/tonne,
- Using unit costs from the *SEDAR* database may have inconsistencies due to how mines report their costs. A larger database stratified by mining cost and capacity would allow for refinement of the mining cost range,
- Detailed designs have not been done at this stage; a wide range of potential costs needs to be evaluated to determine the potential risk of the deposit,

These limitations in the step are that the mining rates are biased to historical practices and do not likely account for larger, automated and mechanized mining equipment. The large range in relative costs is possibly a result of combining highly mechanized mines with more manual intensive mines.

Step three is the cashflow analysis for each potentially feasible mining method identified in step one and the corresponding factors determined from step two. Limitations in step three include:

- Capital costs not included as the deposit is within a brownfield operation with the cost being sunk,
- Mining cost is assumed to include all remaining incremental capital costs to support the development of the new deposit,
- The costs are assumed to be directly proportional to production rates,
- It is assumed that production is constant year to year and prorated for part years. This allows for constant revenue and costs,
- Operating costs are all in, including development, production, haulage, milling and technical and administrative costs,
- The wide variety in relative cost may not select a clear winner but allows for the identification of the most promising methods for more detailed analysis,
- Revenues assumes a constant price of gold and a fixed foreign exchange rate. In reality, both these factors are volatile,

- Grade is also assumed to be constant throughout the life of the deposit where in reality grade can fluctuate from year to year,
- Inconsistent grade and rock type can also impact mill recoveries which is not taken into account for this evaluation procedure as the level of detailed analysis would be covered in a feasibility study, and
- The predicted grade is assumed to be realized.

As a result of these limitations, the application of the proposed procedure should be limited to the scoping phase of a feasibility study. More detailed feasibility assessments are required to build the business case to commit the financial resources required to develop the deposit. As part of the detailed analysis, an optimize grade and corresponding tonnage would be determined to maximize the profit.

6.1.1 Bonanza Ledge Case Study

The evaluation procedure was calibrated using the values from the literature review. The evaluation procedure was then tested using a case study from the Bonanza Ledge Mine comparing projected to the actual results. Mining method, recoveries, dilution and mining costs were within ranges projected by the procedure. The actual rate of production was greater than the rate projected by the procedure. A review of initial design assumptions and actual practices at the Bonanza Ledge Mine provides explanations for many of the differences observed between predicted and actual mining rates. This provides a level of confidence in the predictive capabilities of the procedure, particularly because the Bonanza Ledge data was for 2018.

6.1.2 BC Vein Evaluation

Testing of the evaluation procedure against the actual results obtained from the Bonanza Ledge deposit provided a level of confidence to use the evaluation to assess the potential economic feasibility of mining of the BC Vein Deposit. Three technically feasible mining methods were identified for economic analysis based on the geometric and geotechnical properties of the deposit. Based on the economic analysis, it was determined that only two mining methods had the ability to generate positive cashflows.

The two methods with positive cashflows were sublevel stoping and shrinkage stoping with sublevel stoping generating the greatest cashflow.

This thesis has successfully developed an objective procedure for evaluation of brownfield mineral deposits.

The evaluation procedure uses geometric and geotechnical information typically available to the mine technical staff and thus allows the in-house evaluation of additional ore bodies within the vicinity of the mine.

Although this procedure is based on information from the Bonanza Ledge deposit, it is likely applicable to mines operating in similar jurisdictions.

6.2 Recommendations

Further case studies comparing values predicted from the evaluation procedure compared to the actual values obtained during mining operations would allow refinement of the production rate parameters (Taylor's Equation) and operating costs. Case studies should include deposits with similar and different geotechnical and geometric conditions to the initial Bonanza Ledge case study.

Case studies in different regulatory jurisdictions would allow the development of location specific parameters for production rates and operating costs.

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Appendix A: Productivity and Cash Flow Calculations for Bonanza Ledge Case Study

INPUTS

Resource Tonnes	264000
Resource Grade (gpt)	7.3
Gold Price (CAD/Oz)	1600
Relative Cost	21
Mill & GA Cost	

DILUTION AND RECOVERY

Mining Method	Min Recovery	Max Recovery	Min Ore Tonnes	Max Ore Tonnes	Min Dilution	Max Dilution
Room and Pillar	60%	80%	158,400	211,200	10%	20%
Sublevel Stoping	60%	80%	158,400	211,200	10%	20%
Cut and Fill	90%	100%	237,600	264,000	5%	10%
Shrinkage Stoping	70%	85%	184,800	224,400	0%	10%
Vertical Crater Retrea	60%	80%	158,400	211,200	10%	20%
Longwall mining	70%	90%	184,800	237,600	10%	20%
Sublevel Caving	80%	90%	211,200	237,600	10%	35%
Block Caving	90%	100%	237,600	264,000	10%	20%

TONNES AND GRADE

	MinR/MinD Tonnes	MinR/MaxD Tonnes	MaxR/MinD Tonnes	MaxR/MaxD Tonnes	MinR/MinD Grade	MinR/MaxD Grade	MaxR/MinD Grade	MaxR/MaxD Grade	MinR/MinD Ounces	MinR/MaxD Ounces	MaxR/MinD Ounces	MaxR/MaxD Ounces
Room and Pillar	176,000	198,000	234,667	264,000	6.57	5.84	6.57	5.84	37,177	37,177	49,569	49,569
Sublevel Stoping	176,000	198,000	234,667	264,000	6.57	5.84	6.57	5.84	37,177	37,177	49,569	49,569
Cut and Fill	250,105	264,000	277,895	293,333	6.94	6.57	6.94	6.57	55,765	55,765	61,961	61,961
Shrinkage Stoping	184,800	205,333	224,400	249,333	7.30	6.57	7.30	6.57	43,373	43,373	52,667	52,667
Vertical Crater Retrea	176,000	198,000	234,667	264,000	6.57	5.84	6.57	5.84	37,177	37,177	49,569	49,569
Longwall mining	205,333	231,000	264,000	297,000	6.57	5.84	6.57	5.84	43,373	43,373	55,765	55,765
Sublevel Caving	234,667	324,923	264,000	365,538	6.57	4.75	6.57	4.75	49,569	49,569	55,765	55,765
Block Caving	264,000	297,000	293,333	330,000	6.57	5.84	6.57	5.84	55,765	55,765	61,961	61,961

COST

	Min Relative Cost	Max Relative Cost	Min Cost 1	Max Cost 1	Min Cost 2	Max Cost 2	Min Cost 3	Max Cost 3	Min Cost 4	Max Cost 4
Room and Pillar	7	20	$25,872,000.00	$73,920,000.00	$29,106,000.00	$83,160,000.00	$34,496,000.00	$98,560,000.00	$38,808,000.00	$110,880,000.00
Sublevel Stoping	7	25	$25,872,000.00	$92,400,000.00	$29,106,000.00	$103,950,000.00	$34,496,000.00	$123,200,000.00	$38,808,000.00	$138,600,000.00
Cut and Fill	20	70	$105,044,210.53	$367,654,736.84	$110,880,000.00	$388,080,000.00	$116,715,789.47	$408,505,263.16	$123,200,000.00	$431,200,000.00
Shrinkage Stoping	20	50	$77,616,000.00	$194,040,000.00	$86,240,000.00	$215,600,000.00	$94,248,000.00	$235,620,000.00	$104,720,000.00	$261,800,000.00
Vertical Crater Retrea	7	25	$25,872,000.00	$92,400,000.00	$29,106,000.00	$103,950,000.00	$34,496,000.00	$123,200,000.00	$38,808,000.00	$138,600,000.00
Longwall mining	7	20	$30,184,000.00	$86,240,000.00	$33,957,000.00	$97,020,000.00	$38,808,000.00	$110,880,000.00	$43,659,000.00	$124,740,000.00
Sublevel Caving	5	15	$24,640,000.00	$73,920,000.00	$34,116,923.08	$102,350,769.23	$27,720,000.00	$83,160,000.00	$38,381,538.46	$115,144,615.38
Block Caving	1	2.5	$5,544,000.00	$13,860,000.00	$6,237,000.00	$15,592,500.00	$6,160,000.00	$15,400,000.00	$6,930,000.00	$17,325,000.00

REVENUE

	a Revenue Low	b Revenue High
Room and Pillar	$59,482,479.74	$79,309,972.99
Sublevel Stoping	$59,482,479.74	$79,309,972.99
Cut and Fill	$89,223,719.61	$99,137,466.23
Shrinkage Stoping	$69,396,226.36	$84,266,846.30
Vertical Crater Retrea	$59,482,479.74	$79,309,972.99
Longwall mining	$69,396,226.36	$89,223,719.61
Sublevel Caving	$79,309,972.99	$89,223,719.61
Block Caving	$89,223,719.61	$99,137,466.23

CASH FLOWS

	Min CF 1a	Max CF 1a	Min CF 1b	Max CF 1b	Min CF 2a	Max CF 2a	Min CF 2b	Max CF 2b
Room and Pillar	$33,610,479.74	($14,437,520.26)	$53,437,972.99	$5,389,972.99	$30,376,479.74	($23,677,520.26)	$50,203,972.99	($3,850,027.01)
Sublevel Stoping	$33,610,479.74	($32,917,520.26)	$53,437,972.99	($13,090,027.01)	$30,376,479.74	($44,467,520.26)	$50,203,972.99	($24,640,027.01)
Cut and Fill	($15,820,490.92)	($278,431,017.23)	($5,906,744.29)	($268,517,270.61)	($21,656,280.39)	($298,856,280.39)	($11,742,533.77)	($288,942,533.77)
Shrinkage Stoping	($8,219,773.64)	($124,643,773.64)	$6,650,846.30	($109,773,153.70)	($16,843,773.64)	($146,203,773.64)	($1,973,153.70)	($131,333,153.70)
Vertical Crater Retrea	$33,610,479.74	($32,917,520.26)	$53,437,972.99	($13,090,027.01)	$30,376,479.74	($44,467,520.26)	$50,203,972.99	($24,640,027.01)
Longwall mining	$39,212,226.36	($16,843,773.64)	$59,039,719.61	$2,983,719.61	$35,439,226.36	($27,623,773.64)	$55,266,719.61	($7,796,280.39)
Sublevel Caving	$54,669,972.99	$5,389,972.99	$64,583,719.61	$15,303,719.61	$45,193,049.91	($23,040,796.24)	$55,106,796.53	($13,127,049.62)
Block Caving	$83,679,719.61	$75,363,719.61	$93,593,466.23	$85,277,466.23	$82,986,719.61	$73,631,219.61	$92,900,466.23	$83,544,966.23
	Min CF 3a	Max CF 3a	Min CF 3b	Max CF 3b	Min CF 4a	Max CF 4a	Min CF 4b	Max CF 4b
Room and Pillar	$24,986,479.74	($39,077,520.26)	$44,813,972.99	($19,250,027.01)	$20,674,479.74	($51,397,520.26)	$40,501,972.99	($31,570,027.01)
Sublevel Stoping	$24,986,479.74	($63,717,520.26)	$44,813,972.99	($43,890,027.01)	$20,674,479.74	($79,117,520.26)	$40,501,972.99	($59,290,027.01)
Cut and Fill	($27,492,069.86)	($319,281,543.55)	($17,578,323.24)	($309,367,796.92)	($33,976,280.39)	($341,976,280.39)	($24,062,533.77)	($332,062,533.77)
Shrinkage Stoping	($24,851,773.64)	($166,223,773.64)	($9,981,153.70)	($151,353,153.70)	($35,323,773.64)	($192,403,773.64)	($20,453,153.70)	($177,533,153.70)
Vertical Crater Retrea	$24,986,479.74	($63,717,520.26)	$44,813,972.99	($43,890,027.01)	$20,674,479.74	($79,117,520.26)	$40,501,972.99	($59,290,027.01)
Longwall mining	$30,588,226.36	($41,483,773.64)	$50,415,719.61	($21,656,280.39)	$25,737,226.36	($55,343,773.64)	$45,564,719.61	($35,516,280.39)
Sublevel Caving	$51,589,972.99	($3,850,027.01)	$61,503,719.61	$6,063,719.61	$40,928,434.53	($35,834,642.40)	$50,842,181.15	($25,920,895.77)
Block Caving	$83,063,719.61	$73,823,719.61	$92,977,466.23	$83,737,466.23	$82,293,719.61	$71,898,719.61	$92,207,466.23	$81,812,466.23

MINING RATE AND MINE LIFE

Mining Method	a	b	Mining Rate 1	Mining Rate 2	Mining Rate 3	Mining Rate 4	Mine Life 1	Mine Life 2	Mine Life 3	Mine Life 4
Room and Pillar	0.297	0.562	263.47	281.50	309.71	330.90	1.83	1.93	2.08	2.19
Sublevel Stoping	0.297	0.562	263.47	281.50	309.71	330.90	1.83	1.93	2.08	2.19
Cut and Fill	0.297	0.562	321.00	330.90	340.58	351.09	2.13	2.19	2.24	2.29
Shrinkage Stoping	0.297	0.562	270.80	287.32	302.02	320.44	1.87	1.96	2.04	2.13
Vertical Crater Retrea	0.297	0.562	263.47	281.50	309.71	330.90	1.83	1.93	2.08	2.19
Longwall mining	0.297	0.562	287.32	306.98	330.90	353.55	1.96	2.06	2.19	2.30
Sublevel Caving	0.297	0.562	309.71	371.86	330.90	397.31	2.08	2.39	2.19	2.52
Block Caving	0.123	0.649	406.00	438.25	434.73	469.27	1.78	1.86	1.85	1.93

CASH FLOW AND MINE LIFE SUMMARY

	Min CF	Max CF	Min CF Mine Life	Max CF Mine Life
Room and Pillar	($23,677,520.26)	$53,437,972.99	2.19	1.83
Sublevel Stoping	($44,467,520.26)	$53,437,972.99	2.19	1.83
Cut and Fill	($298,856,280.39)	($5,906,744.29)	2.29	2.13
Shrinkage Stoping	($146,203,773.64)	$6,650,846.30	2.13	1.87
Vertical Crater Retrea	($44,467,520.26)	$53,437,972.99	2.19	1.83
Longwall mining	($27,623,773.64)	$59,039,719.61	2.30	1.96
Sublevel Caving	($23,040,796.24)	$64,583,719.61	2.52	2.08
Block Caving	$73,631,219.61	$93,593,466.23	1.93	1.78

MIN YEARLY CASH FLOW

Min Cashflow	Y1	Y2	Y3
Sublevel Stoping	($20,343,688.88)	($20,343,688.88)	($3,780,142.51)
Cut and Fill	($130,559,175.80)	($130,559,175.80)	($37,737,928.78)
Shrinkage Stoping	($68,583,256.99)	($68,583,256.99)	($9,037,259.66)

MAX YEARLY CASH FLOW

Max Cashflow			
Mining Method	Y1	Y2	Y3
Sublevel Stoping	$29,198,782.39	$24,239,190.60	
Cut and Fill	($2,767,067.06)	($2,767,067.06)	($372,610.18)
Shrinkage Stoping	$3,557,220.31	$3,093,625.99	

Appendix B: Productivity and Cash Flow Calculations for BC Vein Case Study

INPUTS

Resource Tonnes	596903
Resource Grade (gpt)	5.87
Gold Price (CAD/Oz)	2000
Relative Cost	21

DILUTION AND RECOVERY

Mining Method	Min Recovery	Max Recovery	Min Ore Tonnes	Max Ore Tonnes	Min Dilution	Max Dilution
Room and Pillar	60%	80%	358,142	477,522	10%	20%
Sublevel Stoping	60%	80%	358,142	477,522	10%	20%
Cut and Fill	90%	100%	537,213	596,903	5%	10%
Shrinkage Stoping	70%	85%	417,832	507,368	0%	10%
Vertical Crater Retreat	60%	80%	358,142	477,522	10%	20%
Longwall mining	70%	90%	417,832	537,213	10%	20%
Sublevel Caving	80%	90%	477,522	537,213	10%	35%
Block Caving	90%	100%	537,213	596,903	10%	20%

TONNES AND GRADE

	MinR/MinD Tonnes	MinR/MaxD Tonnes	MaxR/MinD Tonnes	MaxR/MaxD Tonnes	MinR/MinD Grade	MinR/MaxD Grade	MaxR/MinD Grade	MaxR/MaxD Grade	MinR/MinD Ounces	MinR/MaxD Ounces	MaxR/MinD Ounces	MaxR/MaxD Ounces
Room and Pillar	397,935	447,677	530,580	596,903	5.28	4.70	5.28	4.70	67,590	67,590	90,120	90,120
Sublevel Stoping	397,935	447,677	530,580	596,903	5.28	4.70	5.28	4.70	67,590	67,590	90,120	90,120
Cut and Fill	565,487	596,903	628,319	663,226	5.58	5.28	5.58	5.28	101,385	101,385	112,650	112,650
Shrinkage Stoping	417,832	464,258	507,368	563,742	5.87	5.28	5.87	5.28	78,855	78,855	95,753	95,753
Vertical Crater Retreat	397,935	447,677	530,580	596,903	5.28	4.70	5.28	4.70	67,590	67,590	90,120	90,120
Longwall mining	464,258	522,290	596,903	671,516	5.28	4.70	5.28	4.70	78,855	78,855	101,385	101,385
Sublevel Caving	530,580	734,650	596,903	826,481	5.28	3.82	5.28	3.82	90,120	90,120	101,385	101,385
Block Caving	596,903	671,516	663,226	746,129	5.28	4.70	5.28	4.70	101,385	101,385	112,650	112,650

COST

	Min Relative Cost	Max Relative Cost	Min Cost 1	Max Cost 1	Min Cost 2	Max Cost 2	Min Cost 3	Max Cost 3	Min Cost 4	Max Cost 4
Room and Pillar	7	20	$58,496,494.00	$167,132,840.00	$65,808,555.75	$188,024,445.00	$77,995,325.33	$222,843,786.67	$87,744,741.00	$250,699,260.00
Sublevel Stoping	7	25	$58,496,494.00	$208,916,050.00	$65,808,555.75	$235,030,556.25	$77,995,325.33	$278,554,733.33	$87,744,741.00	$313,374,075.00
Cut and Fill	20	70	$237,504,562.11	$831,265,967.37	$250,699,260.00	$877,447,410.00	$263,893,957.89	$923,628,852.63	$278,554,733.33	$974,941,566.67
Shrinkage Stoping	20	50	$175,489,482.00	$438,723,705.00	$194,988,313.33	$487,470,783.33	$213,094,371.00	$532,735,927.50	$236,771,523.33	$591,928,808.33
Vertical Crater Retreat	7	25	$58,496,494.00	$208,916,050.00	$65,808,555.75	$235,030,556.25	$77,995,325.33	$278,554,733.33	$87,744,741.00	$313,374,075.00
Longwall mining	7	20	$68,245,909.67	$194,988,313.33	$76,776,648.38	$219,361,852.50	$87,744,741.00	$250,699,260.00	$98,712,833.63	$282,036,667.50
Sublevel Caving	5	15	$55,710,946.67	$167,132,840.00	$77,138,233.85	$231,414,701.54	$62,674,815.00	$188,024,445.00	$86,780,513.08	$260,341,539.23
Block Caving	1	2.5	$12,534,963.00	$31,337,407.50	$14,101,833.38	$35,254,583.44	$13,927,736.67	$34,819,341.67	$15,668,703.75	$39,171,759.38

REVENUE

	a	b
	Revenue Low	Revenue High
Room and Pillar	$135,180,532.93	$180,240,710.57
Sublevel Stoping	$135,180,532.93	$180,240,710.57
Cut and Fill	$202,770,799.39	$225,300,888.22
Shrinkage Stoping	$157,710,621.75	$191,505,754.98
Vertical Crater Retreat	$135,180,532.93	$180,240,710.57
Longwall mining	$157,710,621.75	$202,770,799.39
Sublevel Caving	$180,240,710.57	$202,770,799.39
Block Caving	$202,770,799.39	$225,300,888.22

CASH FLOWS

	Min CF 1a	Max CF 1a	Min CF 1b	Max CF 1b	Min CF 2a	Max CF 2a	Min CF 2b	Max CF 2b
Room and Pillar	$76,684,038.93	($31,952,307.07)	$121,744,216.57	$13,107,870.57	$69,371,977.18	($52,843,912.07)	$114,432,154.82	($7,783,734.43)
Sublevel Stoping	$76,684,038.93	($73,735,517.07)	$121,744,216.57	($28,675,339.43)	$69,371,977.18	($99,850,023.32)	$114,432,154.82	($54,789,845.68)
Cut and Fill	($34,733,762.71)	($628,495,167.97)	($12,203,673.89)	($605,965,079.15)	($47,928,460.61)	($674,676,610.61)	($25,398,371.78)	($652,146,521.78)
Shrinkage Stoping	($17,778,860.25)	($281,013,083.25)	$16,016,272.98	($247,217,950.02)	($37,277,691.58)	($329,760,161.58)	($3,482,558.35)	($295,965,028.35)
Vertical Crater Retreat	$76,684,038.93	($73,735,517.07)	$121,744,216.57	($28,675,339.43)	$69,371,977.18	($99,850,023.32)	$114,432,154.82	($54,789,845.68)
Longwall mining	$89,464,712.08	($37,277,691.58)	$134,524,889.73	$7,782,486.06	$80,933,973.38	($61,651,230.75)	$125,994,151.02	($16,591,053.11)
Sublevel Caving	$124,529,763.91	$13,107,870.57	$147,059,852.73	$35,637,959.39	$103,102,476.73	($51,173,990.97)	$125,632,565.55	($28,643,902.14)
Block Caving	$190,235,836.39	$171,433,391.89	$212,765,925.22	$193,963,480.72	$188,668,966.02	$167,516,215.96	$211,199,054.84	$190,046,304.78
	Min CF 3a	Max CF 3a	Min CF 3b	Max CF 3b	Min CF 4a	Max CF 4a	Min CF 4b	Max CF 4b
Room and Pillar	$57,185,207.60	($87,663,253.74)	$102,245,385.24	($42,603,076.09)	$47,435,791.93	($115,518,727.07)	$92,495,969.57	($70,458,549.43)
Sublevel Stoping	$57,185,207.60	($143,374,200.40)	$102,245,385.24	($98,314,022.76)	$47,435,791.93	($178,193,542.07)	$92,495,969.57	($133,133,364.43)
Cut and Fill	($61,123,158.50)	($720,858,053.24)	($38,593,069.68)	($698,327,964.42)	($75,783,933.94)	($772,170,767.27)	($53,253,845.12)	($749,640,678.45)
Shrinkage Stoping	($55,383,749.25)	($375,025,305.75)	($21,588,616.02)	($341,230,172.52)	($79,060,901.58)	($434,218,186.58)	($45,265,768.35)	($400,423,053.35)
Vertical Crater Retreat	$57,185,207.60	($143,374,200.40)	$102,245,385.24	($98,314,022.76)	$47,435,791.93	($178,193,542.07)	$92,495,969.57	($133,133,364.43)
Longwall mining	$69,965,880.75	($92,988,638.25)	$115,026,058.39	($47,928,460.61)	$58,997,788.13	($124,326,045.75)	$104,057,965.77	($79,265,868.11)
Sublevel Caving	$117,565,895.57	($7,783,734.43)	$140,095,984.39	$14,746,354.39	$93,460,197.50	($80,100,828.66)	$115,990,286.32	($57,570,739.84)
Block Caving	$188,843,062.73	$167,951,457.73	$211,373,151.55	$190,481,546.55	$187,102,095.64	$163,599,040.02	$209,632,184.47	$186,129,128.84

MINING RATE AND MINE LIFE

Mining Method	a	b	Mining Rate 1	Mining Rate 2	Mining Rate 3	Mining Rate 4	Mine Life 1	Mine Life 2	Mine Life 3	Mine Life 4
Room and Pillar	0.297	0.562	416.73	445.24	489.85	523.38	2.62	2.75	2.97	3.12
Sublevel Stoping	0.297	0.562	416.73	445.24	489.85	523.38	2.62	2.75	2.97	3.12
Cut and Fill	0.297	0.562	507.71	523.38	538.68	555.30	3.05	3.12	3.20	3.27
Shrinkage Stoping	0.297	0.562	428.31	454.44	477.69	506.83	2.67	2.80	2.91	3.05
Vertical Crater Retreat	0.297	0.562	416.73	445.24	489.85	523.38	2.62	2.75	2.97	3.12
Longwall mining	0.297	0.562	454.44	485.54	523.38	559.19	2.80	2.95	3.12	3.29
Sublevel Caving	0.297	0.562	489.85	588.16	523.38	628.41	2.97	3.42	3.12	3.60
Block Caving	0.123	0.649	689.39	744.16	738.18	796.82	2.37	2.47	2.46	2.57

CASH FLOW AND MINE LIFE SUMMARY

	Min CF	Max CF	Min CF Mine Life	Max CF Mine Life
Room and Pillar	($52,843,912.07)	$121,744,216.57	3.12	2.62
Sublevel Stoping	($99,850,023.32)	$121,744,216.57	3.12	2.62
Cut and Fill	($674,676,610.61)	($12,203,673.89)	3.27	3.05
Shrinkage Stoping	($329,760,161.58)	$16,016,272.98	3.05	2.67
Vertical Crater Retreat	($99,850,023.32)	$121,744,216.57	3.12	2.62
Longwall mining	($61,651,230.75)	$134,524,889.73	3.29	2.80
Sublevel Caving	($51,173,990.97)	$147,059,852.73	3.60	2.97
Block Caving	$167,516,215.96	$212,765,925.22	2.57	2.37

YEARLY MIN CASHFLOWS

Min Cashflow	Y1	Y2	Y3	Y4
Sublevel Stoping	($31,955,942.93)	($31,955,942.93)	($31,955,942.93)	($3,982,194.55)
Cut and Fill	($206,185,129.47)	($206,185,129.47)	($206,185,129.47)	($56,121,222.19)
Shrinkage Stoping	($108,211,740.68)	($108,211,740.68)	($108,211,740.68)	($5,124,939.55)

YEARLY MAX CASHFLOWS

Max Cashflow				
Mining Method	Y1	Y2	Y3	Y4
Sublevel Stoping	$46,535,006.29	$46,535,006.29	$28,674,203.99	
Cut and Fill	($3,999,252.09)	($3,999,252.09)	($3,999,252.09)	($205,917.63)
Shrinkage Stoping	$5,992,554.42	$5,992,554.42	$4,031,164.13	