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Long hole drilling for the platinum industry—a mining perspective

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Long blast hole drilling has been successfully implemented since the 1950s in caving operations, steeply dipping wide reefs and boxhole/raise developments worldwide. However, when applying long hole drilling to narrow tabular reefs, the challenges of achieving high economic drilling rates, combined with acceptable accuracy and appropriate powder factors have yet to be overcome. The focus in this paper is the selection, application and optimization of long hole drilling equipment at mining operations shallower than 2 000 meters. The drilling, blasting and cleaning operations have to be optimized as a process if maximum production at minimum per unit cost is to be achieved. For this to be possible, it is essential that the mine design and layout match the mining process being implemented.

The critical parameters affecting the sustainable implementation of long hole drilling in stoping applications for platinum mines are improved safety and higher production rates compared to conventional methods. In terms of drilling, both top hammer and down-the hole hammer configurations have been considered. For economic drilling, the benefits of monitoring and optimizing performance (i.e. drill bit rotation, thrust, and percussive power) for specific rock types are discussed.

While the drive is for longer, more accurate, smaller diameter holes to reduce drilling costs, the design of appropriate blast hole patterns and the corresponding charging and initiation systems is crucial for success. The effect of hole diameter and pattern on the powder factor, when comparing small diameter, shorter holes versus large diameter, longer holes, is a major factor in achieving good blasting results.

Once appropriate drilling and blasting have been implemented, the cleaning process becomes a potentially critical operation to ensure the production, cost and safety goals are met. Hence, the blast hole pattern and charge have to be integrated for optimal fragmentation to suit the cleaning cycle for maximum production output.

Providing that long blast hole drilling is applied as a process in a planned, committed and well-managed manner, the benefits in safety, production and cost over conventional methods should be achievable. However, if the implementation process is not planned and supported by all parties concerned, it will not be possible to succeed in the new operation. Hence, an implementation or roll-out plan of new technology is crucial for success.

Introduction

The operating cycle of in-stope and ledging long hole drilling is no different from conventional mining in the sense that both cycles have support, drill, blast and cleaning components. The differences occur within these components, however, and need to be understood if the long hole drilling method is to be successfully implemented in a sustainable economic manner. In-stope long hole drilling is the method whereby holes are drilled parallel to the face from a drill gully through to a toe gully. Figure 1 shows a typical in-stope long hole drilling layout utilizing trackless drill rigs[1]. Long hole ledging is the method whereby blind holes are drilled parallel to the advancing face from a centre gully. Typically, the ledging blind long holes are shorter than the in-stope through holes (i.e.: 8 m compared to 15 to 20 m).

By addressing potential bottlenecks (e.g. drilling rate, blasting advance rate, gully development rate) and constraints (e.g. size and mobility of rig, drill string life and costs, development ratio) of the in-stope long hole drilling method, the operation should be able to maintain similar (or better) production rates to ensure that the mine and plant capacities are kept at an optimum level.

The major factors directly affecting the sustainable economic implementation of long hole drilling in a stoping or ledging environment are site selection, support requirements (safety), drill rig selection/optimization, blasting and cleaning.

Site selection

The objective is to define the regional (large-scale) influences[2,3] of the orebody on the in-stope or ledging long hole drilling method and focuses on the geometrical properties of the orebody, such as its depth below surface, its dip and its geometry.

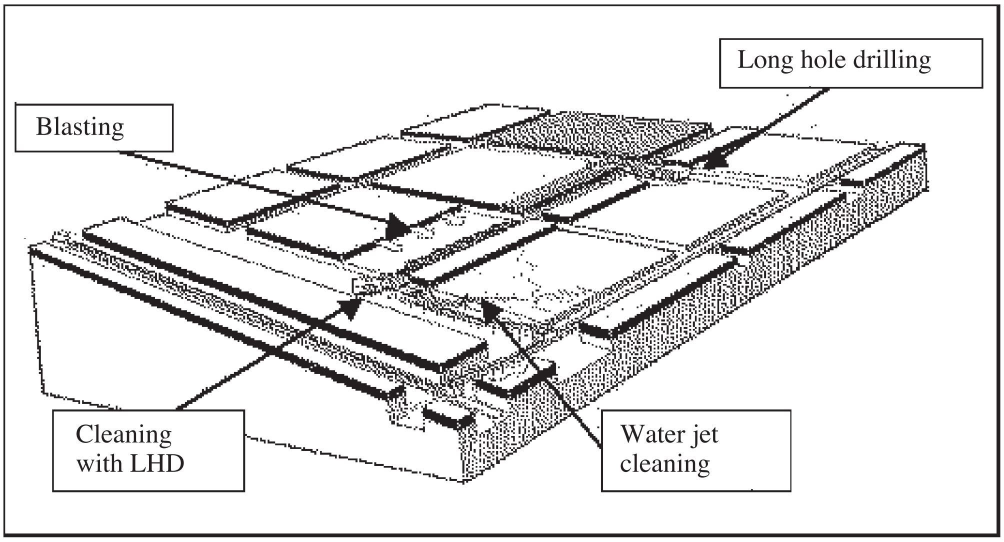
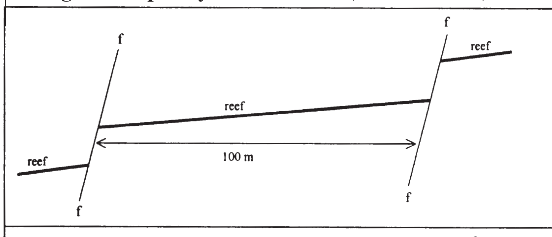
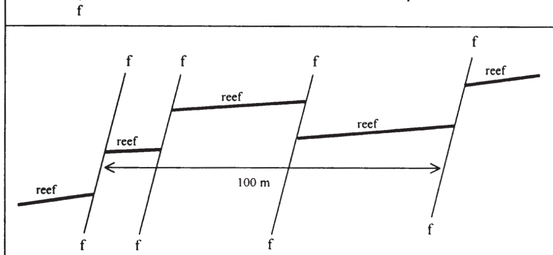
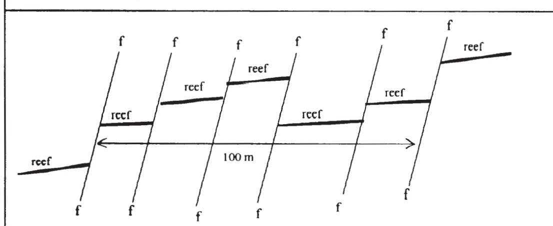


Figure 1. Typical in-stope long hole drilling layout for a trackless mining operation[1]

Table I
The applicability of long hole drilling in geotechnical areas with varying complexity of geological structures

Geological complexity (Note: f = fault)	Applicability of long hole drilling
	• Block continuity between bounding geological structures • Little influence on layout of infrastructure • Easy application
	• Few faults between bounding geological structures • Selective application to possibly two largest blocks • Applicable with lower yield from long hole drilling
	• Many faults between bounding structure • Scattered small size blocks • Applicability unlikely

Geological structures

The number, persistence and character of major geological structures (e.g. faults and dykes) may influence the application of long hole drilling in that a capacity for flexibility and selectivity in stoping may be required to accommodate changes in the spatial distribution of the ore-bearing horizon.

Therefore, the applicability of the long hole drilling method is largely a function of the number of major geological structures within a predefined area (Table I). A simple approach could be to use the drill length capabilities of the long hole drill rig and convert this into a minimum inter-panel block dimension. If a rig can drill 20 m, then no major geological structure should occur within 20 m in the direction of drilling. Simply, this implies that a 100 m x 100 m block should ideally not contain more than 5 geological structures to ensure sufficient block sizes for optimized drilling.

Reef dip

The two aspects of reef dip that influence the application of the in-stope long hole drilling method are the ability to throw blast and the ride of the hangingwall and footwall, for the following reasons:

- The steeper the dip, the more efficient is blast and gravity-assisted ore movement into secondary cleaning points (gullies)

- The steeper the dip of the orebody, the more pronounced are the relative displacements of the hangingwall and footwall strata. The ride of the hosting rock mass influences the stability of in-stope pillars. Depending on the reef dip, dip orientated pillars may be more stable than strike pillars
- Since the choice of pillar orientation could be governed by the dip of the reef, it follows that the layout, in turn, is governed by the dip of the reef
- An investigation of the optimum reef dip for blast assisted cleaning and in-stope pillar stability, needs to be conducted. It is proposed that some compromise will be required to fulfil the requirements for both.

Reef planarity

The long hole drilling stoping method is most suited to orebodies with continuously planar hangingwall and footwall contacts. In reality, no orebody is continuously planar. The deviation from reef contact planarity is related to the quantity of ore dilution. The impact of dilution on the economic viability of the mining method will give an indication of the degree of deviation from planarity that could possibly be accommodated by the stoping method.

Channel widths

The primary aim of the stoping method is to increase ore recovery by reducing dilution. Long hole drilling and blasting could theoretically be tailored to ensure a selective minimum mining cut into the orebody. An engineered blast and the concept of man-free stopes could ensure that the stoping width is equivalent to the channel width. The reduction in stoping width and waste compared to that in conventional stopes is, however, dependent on whether the reef conglomerates are contained within single or multi-facies deposits. Multi-facies deposits are normally channelized, implying a complex assemblage of reef bearing horizons and internal waste. Whatever the mining method, the inclusion of waste into the milling system is unavoidable. Depending on the extent of channelization, grade distribution could be highly erratic.

The impact of channel widths on the long hole drilling method (Figure 2) is as follows:

- The stoping method is extremely sensitive to drill hole accuracy i.e. the target horizon must be located between the collar and toe of the hole. It follows that the smaller the pay channel width, the more probable that the hole will be misaligned from the target horizon. In turn, over-break and dilution will occur when stoping narrow (< 30 cm) channels
- Conversely, wider pay channel widths will accommodate hole inaccuracy and ensure that the bulk of the rock retrieved is ore. For channels exceeding a specific pay width (typically 1 m), there is a strong possibility that not all ore would be recovered i.e. reef will be left in footwall and/or hangingwall. Changing the blast hole layout could result in over-break akin to small channel mining. The economic impact of leaving reef in hangingwall or footwall, and the extent and quantity to which this is done, needs to be investigated for each proposed site.

Grade variability

The average grade within a large block of ground demarcated for long hole drilling may be above the required pay limit. However, individual grid blocks within the large block could fall below this cut-off. Selective mining of grid blocks within the larger block could improve the returns offered by this stoping method. The impact of unmined areas on the overall stability of the rock mass is dependent on the size and shape of abandoned blocks. Ideally, a long hole drilling site should have uniform grade distribution, but this would rarely be the case.

Seismic vulnerability

The vulnerability of the layout to the influences of seismic activity will dictate the off-reef access way positions, on-reef access way positions and the extraction sequence. An examination of the susceptibility of any proposed in-stope long hole drilling layout to seismicity is imperative, and should be done to enable an estimate to be made of:

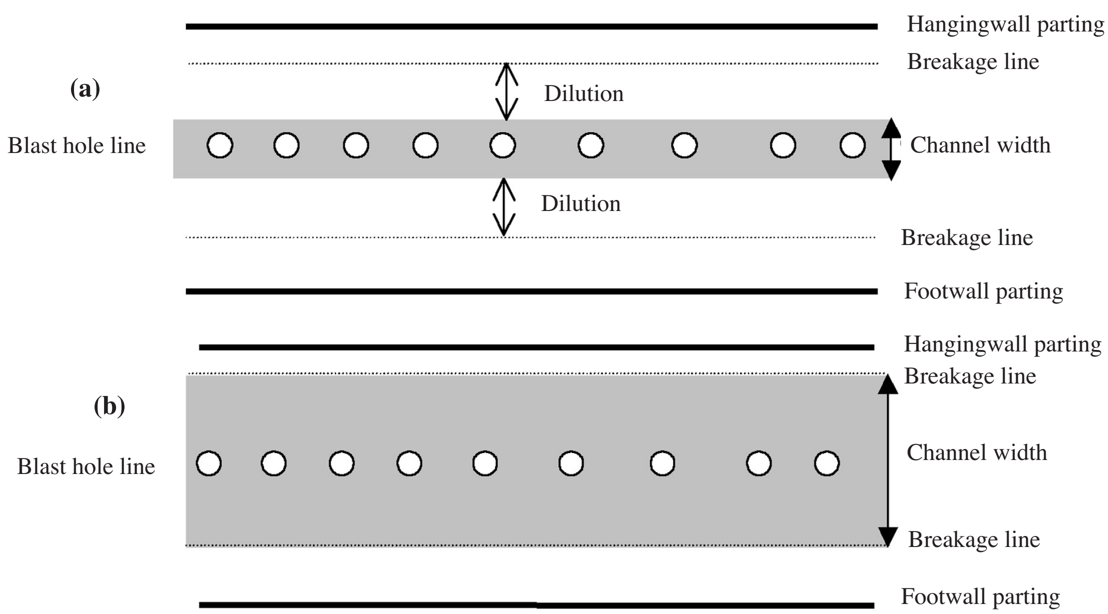


Figure 2. A schematic representation of the influence of pay channel width on breakage and dilution

- Geological structures most prone to instability based on historical datasets
- The ranking of geological structures with regard to risk, thus dictating the likely mining sequence.

Support

While the major benefit of in-stope long hole drilling is that that there are no longer any men at the face or in the stope, the unsupported span between pillars (whether on dip or strike) and the positioning of the access ways are critical to the safety of the workforce.

For shallow operations (<1 000 m), unsupported spans of up to 20 m separated by either dip or strike orientated in-stope pillars promotes the man-free stope concept. With increasing depth it is likely that unsupported spans between in-stope pillars will have to be reduced to less than 10 m, depending on the probable mode of failure of the hangingwall beam i.e. buckling, shear or rotational. Unsupported spans of less than 10 m at depth (say > 1 000 m), with the possibility of dynamic loading, will influence the design of the pillars.

To maintain pillar stability over the duration of the long hole drilling operation and possibly the duration of life of mine stoping operations, entails designing in-stope pillars that are slender enough (width < 1.6 x stoping width) to permit crushing, in both shallow and deep environments. It is perceived that in-stope pillars with greater widths than crush pillars (width > 1.6 x stoping width) will potentially be unstable under both quasi static and dynamic loading. Depending on the strength and condition of the hangingwall and footwall, seismicity emanating from foundation failure of pillars cannot be discounted.

The possible way of maintaining the man-free concept during long hole drilling for stopes at depth will be by integrating backfill with in-stope pillars. Once a stope is excavated, backfill can be placed in the void by sock filling from the up-dip gully to a position at the down-dip end of the stope that is delineated by backfill bulkheads. This concept is probably more feasible for dip orientated in-stope pillar layouts than strike pillar layouts.

The mine layout needs to reduce development ratios on reef, but at the same time allow for safe access gullys for men, equipment and material that are adequately supported and positioned. The support requirements for these excavations will exceed those normally applied in conventional stopes due to the high risks associated with increased personnel exposure. In order to determine the stope stability, preliminary geotechnical and geological mapping of the proposed site should be conducted to determine the rock mass rating in order to test the stability of the excavations to be made by the long hole mining method.

The stope stability, in turn, will determine the support required in the access gully. For example, if a site has a modified stability number[4] value N of 2.2 and hydraulic Radius[4] of 7, the stope is unstable and susceptible to caving. This site would therefore require a heavier gully support than a site that falls within the stable zone (Figure 3).

Production drilling

In order to achieve at least the same equivalent face advance rate as conventional mining, it is critical that the long hole drilling stope layout and the drilling equipment utilized are compatible and promote high rates of economic drilling. In addition, the blasting and cleaning operations must interface productively with the long hole drilling so that bottlenecks are minimized.

Whether the application is for shallow dipping (< 20 degrees) or steeply dipping (above 20 degrees to near vertical) reefs, the mining layout and the selection and performance of the drilling equipment are critical. While the mining layout should cater for all elements of the operation (i.e. development, drilling, blasting, cleaning, support, etc.), the whole operation hinges on the drilling performance.

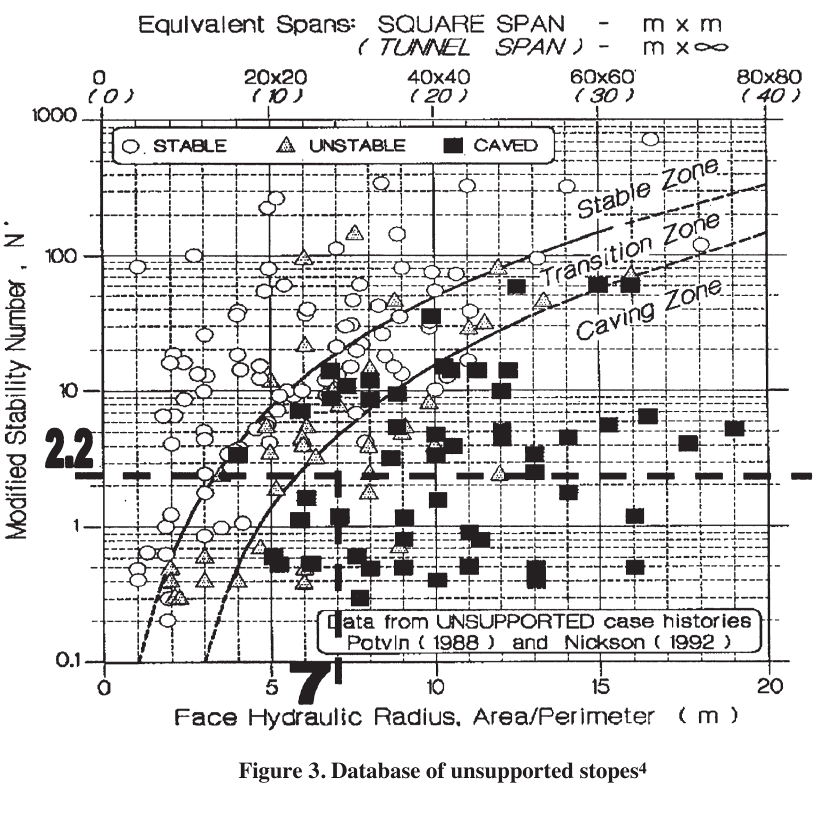


Figure 3. Database of unsupported stopes[4]

Drilling accuracy[5] determines the stope width that can be mined and the length of the hole that can be drilled, and has a significant effect on the efficiency and economics of the mining operation. Hole deviation is a multi-dimensional problem dependent on factors such as collaring, alignment and trajectory deviations. Drilling can be further influenced by factors such as the fracture condition of the rock, the geological structure, and mechanical aspects of the drilling equipment, for instance: type of hammer, in-hole guidance, thrust, bit type, and wear of the bit, and wear of the drill-string guides.

Drilling represents the most critical aspect to the mining cycle in that it is the highest cost component and represents the critical component of the mining cycle, as blocks need to be predrilled before blasting can take place. The hole diameter, drill pattern and burden spacing directly influence the stoping width.

The drillability and abrasivity properties of the rock to be drilled are the two most important criteria to consider when selecting drilling equipment. The drillability property of a rock encompasses brittleness (B_3), compressive (*qc*) and tensile strength (*qt*), the coefficient of rock strength (CRS), rock impact hardness number (RIHN), Vickers microhardness and rock texture, as discussed by Altindag[6,7]. The relationship between brittleness and drillability has been studied and the best correlation was found to be the B_3 value. To date there is no universally accepted rock brittleness concept, but the B_3 concept and its direct relation to specific energy in rock cutting is being suggested as a definition for rock cutting efficiency.

In the long hole drilling research[8], a simple method of measuring the rock drillability and the correlation to percussive drilling equipment selection has been formulated. This entails monitoring an existing drill rig's performance drilling the same rock type in which long hole drilling is to be carried out. For example, a development end drill rig could be monitored to obtain the required data for analysis. Figure 4 shows the comparison of typical rates of penetration (RoP) for the first drill rod versus percussive blow energy for a range of drill bit diameters and drifter types, but for the same *in situ* rock type (i.e. Norite).

From the information collected (i.e. bit diameter, rod penetration rate, drill supply pressure, drill settings and drill type) the specific rock removal rate (SRRR) for the rock type is calculated. This value is a property of the rock type, regardless of the drill or bit diameter used.

The SRRR (mm³/kJ) relationship is:

$$\text{SRRR} = \frac{\text{amount of material removed by percussive drilling per second (mm}^3\text{/s) (mm}^3\text{/kJ)}}{\text{percussive energy required (kJ/s)}}$$

Whether a pneumatic, hydropowered or electro-hydraulic drill is being considered, the same methodology can be applied for the drill selection. A comparison of different drills and drifters for a specific rock type (i.e. norite) is shown in Figure 5 for different bit sizes (41 mm, 51 mm and 65 mm diameter), plotted against percussive blow energy.

If the intention is to drill long holes longer than 10 m in length, then only the drifters on the right-hand side of the 10 m line would be considered. If holes of up to 10 m were being considered, then one could select the smaller drills and drifters on the left-hand side of the 10 m line. A rule of thumb is that for a top hammer drill or drifter, a minimum of 6 kW power is required per metre length of hole you intend to drill if high rock cutting efficiency is to be achieved.

Another important factor to consider is the length of drill rods that will be used. For each joint in the drill string, between 6 to 10% of percussive energy will be lost, depending on the coupling type (i.e. taper thread or parallel thread, male female rod, etc.). Therefore, the longer the drill rod, the more energy at the bit for a given hole length and the more efficient the drilling, in the case of top hammer drifters. The minimum useable energy available at the bit may be decided by the unacceptably long drilling time, rather than by a failure to actually break rock (Figure 6). Both the COP1238 hydraulic drill and the S36IR pneumatic unit have been successfully used to drill holes of in excess of 30 m. These long holes are normally drilled using 3.2 m drill steels, whereas 1.2 m long rods are used for in stope long hole drilling due to space constraints in the drilling gullies. In addition, the rod handling would increase so the overall hole time would increase dramatically. Down-the hole (DTH) hammers do not have as serious a limitation with energy loss, but percussive pressure loss is more critical, and they do have a minimum diameter limitation (currently 89 mm diameter for the Wassara 80).

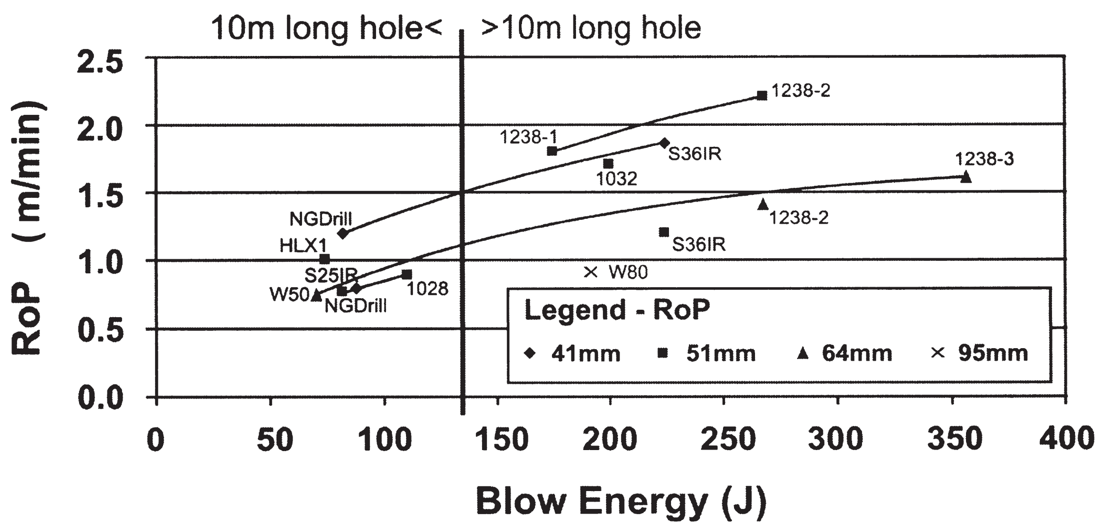


Figure 4. First drill rod rate of penetration versus blow energy for commercial drills

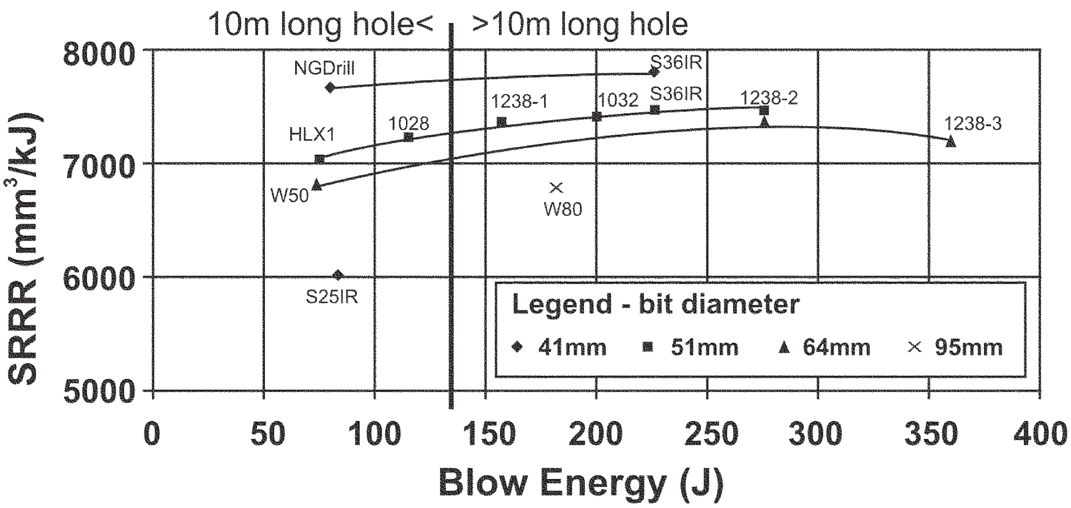


Figure 5. Specific rock removal rate versus blow energy for commercial drills

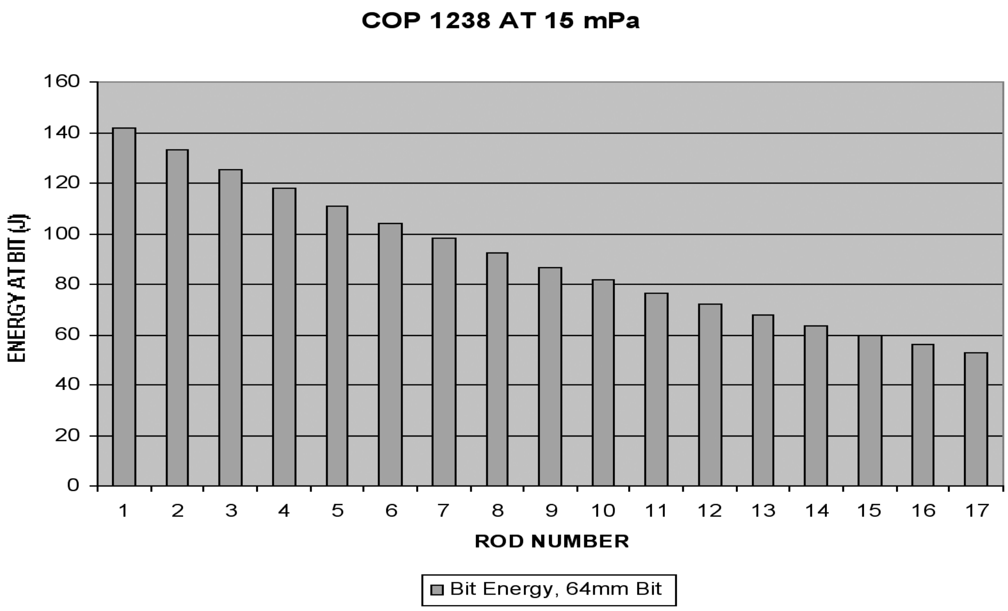


Figure 6. Graph showing the relation between bit energy and the number of drill rods

The degrees of freedom designed into the rig must match the required drilling method and pattern. Too many degrees of freedom will result in a rig that is difficult to set up and maintain stability. Hence, as few as possible pivot and slide mechanisms must be allowed for. The ultimate in manoeuverability and positioning will result in high complexity for set-up and accuracy.

It is impossible to recover a poorly set-up or misaligned hole during drilling. The position and direction of the drill feed must be able to be finely adjusted and kept perfectly steady for the duration of drilling and rod extraction. Further development and monitoring are required on the lower end drill rigs to accurately assess productivities and costs.

Optimizing rotational speed and percussive frequency is another important parameter for efficient rock drilling. This can be achieved by monitoring penetration rates for different rotational speeds at a set percussive frequency, and further proven by collecting chip samples direct from the flushing discharge while drilling. The collected samples can be analysed by particle size distribution (PSD) as shown in Figure 7. Too slow a speed will result in finer chips and slower penetration rates. Too high a speed will give larger chips, slower penetration, as well as high circumferential bit wear. The optimum is specific to each rock type and will give a balance of small and large chips at the ideal penetration rate.

The drill rig and drifter should be able to achieve drilling rates of 150 metres per shift, on a par with our Australian counterparts, in order to blast and clean at rates that compare with conventional mining.

Achieving high drilling accuracy has been one of the major downfalls for long hole drilling in the stope environment. The rig set-up, collaring and thrusting control is critical in this respect, particularly as the drive is to drill smaller diameter holes (41 mm diameter) with a resulting less stiff drill string. The choice of bits also plays a major role. In the 51 mm size the choice is between blade or insert bits and button bits. Blade bits give greater accuracy, being less susceptible to deviation than button bits. Blade bits

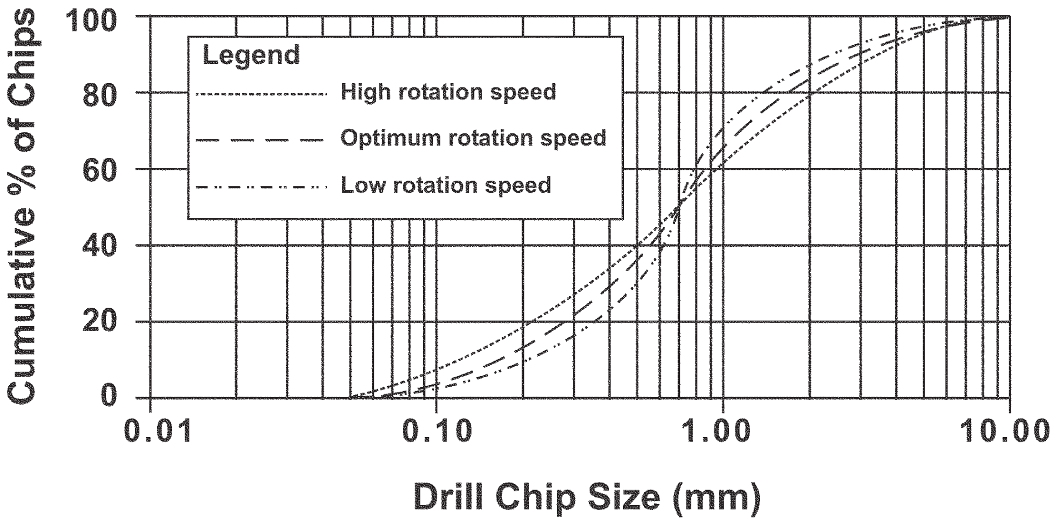


Figure 7. Comparison of particle size distribution

normally require 25 to 50% more rotation speed than button bits. Faster rotation and the lowest thrust possible usually improve hole accuracy. For example, the COP1238, set to operate at 60 Hz on a 51 mm button bit, would require a rotational speed of 225 rpm.

The abrasivity of the rock being drilled has a direct impact on the life of the drill bit and drill string, and its effect varies in direct proportion to the rotational speed (i.e. high rotational speed in a highly abrasive rock type will result in high gauge wear of the bit and rods). Typically, in quartzitic rock, a 51 mm button bit could achieve a 40 m life, while in norite the same bit could achieve 120 m life. The higher the wear rate, the higher the operating cost of the drilling operation.

Blasting

The rock breaking mechanism in long hole stoping is very different from conventional stoping, as are the objectives of the long hole blast[3,5,8]. For example, long hole stoping desires a worker-free stope panel, hence there is a need to throw the majority of the rock out of the panel area and into the gully. Stemming becomes less of an issue in long holes, and decoupling/air gaps are advantageous for achieving a softer break (through a lower powder factor) necessary to maintain the narrow stope width (Figure 2). Care must be taken to ensure subsequent unblasted holes are not damaged or are protected by a sacrificial insert (e.g. PVC tubes). Larger diameter holes (i.e. 65mm diameter) charged with 38 mm diameter explosive cartridges benefit from a lower effective powder factor due to the large air gaps between the cartridge and hole, but drilling costs are higher than a smaller diameter hole (51 mm or 41 mm diameter).

Most problems in blasts originate from inaccurate drilling, hence it is critical that long hole stoping crews are aware of the fundamental importance that drilling has on the overall blast results. The basic difference between conventional narrow reef stoping and long hole stoping is the need to reduce the stoping width from 100 cm–150 cm down to 50 cm–80 cm. The decision to reduce stoping width in long hole stoping requires a reconsideration of the blast hole pattern. The need for two rows (commonly applied in conventional stoping) is questionable as the explosive energy released by a single hole is sufficient to break the rock out to the required stoping width.

Long holes break the rock in a different method from short holes. Long holes break the rock by tensile fracture (compared to compressive fractures for conventional stoping), and the tensile stress splits the rock nearly vertically to the first bedding plane above and below the drill hole (see Figure 4). Thus a measure of control is lost in the stope width control, re-emphasizing the importance of site selection. Generally, more than one bedding plane exists in the immediate proximity of the drill hole and inevitably, a compromise must be found between the burden and the explosive energy, to optimize the stope width. The interhole delay between individual blasts (sequentially fired) also influences the breaking out of the bedding planes during blasts. If sequential shots are out of time, then negative pressure waves are created, which can lead to the breaking of the hangingwall beam. Ultimately the success of long hole stoping depends on the blast design and its performance.

Cleaning

Cleaning plays a fundamental role in long hole stoping as the method depends on the removal of the blasted ore from the panel without requiring workers to enter the panel. The tonnage handling requirement for cleaning is a function of the hole length, stoping width achieved, and the effective burden. The maximum tonnage for long hole stoping is lower than the requirements for conventional stoping (i.e. 80 tons). During blasting, between a single and four blast holes should be initiated at any one time, and the blast should not exceed 50 tons. Based on these production parameters, the cleaning cycle should not pose a problem.

There are two basic methods of blasting a stope panel, defined by the drilling direction. The first long hole stoping layout utilizes dip drilling whereby the rock is blasted into the panel and subsequently removed either by scrapers or by water jets. Steeply dipping reefs call for drilling on dip rather than strike, in a retreating drill and blast pattern using gravity–assisted cleaning. The second method, involving strike drilling, relies on the explosives to perform most of the work with a large percentage of the ore being 'thrown"'into the strike gully.

As the tonnage of rock broken by the long hole drilling method is less than that by conventional stoping, due to the smaller stope width, cleaning should not pose a problem.

The mine layout needs to be conducive for the cleaning operation by ensuring the optimum number of scraper winches or load haul dump trucks are employed (Figure 1).

Based on stope widths of 50 cm, 60 cm and 100 cm with 30 cm, 45 cm and 60 cm blast hole burdens, the following calculated tonnages and cleaning times would be required, based on the average cleaning distance and the number of holes fired per blast (Table II).

From Table III, it can be seen that a maximum strike gully cleaning distancee of 83.5 m, (when 2 shots are fired), equates to a cleaning period of 184 minutes. Thus, two blast holes fired at the most extreme panel can be cleaned in one shift.

However, if the stoping width is not controlled (i.e. the stope width increases) then the cleaning period could increase to over 6 hours, which would equate to more than one cleaning shift. Therefore, when mining at a stoping width of 50 cm to 60 cm, cleaning does not represent a constraint.

Monitoring

The implementation strategy and plan for in-stope long hole drilling must include accurate daily monitoring of the drill rig and blasting results. The drill rig monitoring can be done through the use of a data recorder fitted on to the rig to record all vital parameters on a 24-hour basis[9] if necessary, and downloaded regularly for analysis on a Palm Top or PC. The benefits include cycle times and power consumption measurements, in addition to the pressures (percussive, thrust, rotation) and frequencies that can be measured. A typical profile of results obtained through data collection is shown in Figure 8 for a hole length of 8.4 m using 1.2 m long drill rods.

The results show the activity and cycle time for two holes, including drilling, drill rod extraction and rig movement/set up (i.e. 22 minutes to drill, 10 minutes to extract the rods, and 13 minutes to move the rig and set up for drilling the next hole). The total cycle time for collar to collar is therefore 44 minutes per hole, giving a maximum number of eight holes per eight hour shift (i.e. six-hour drilling time per shift). By using a data logger mounted on the drill rig, the rig utilization and availability can be measured and monitored on a weekly and monthly basis, without the need for monitoring staff.

The blasting results, however, require personal inspection and logging on a daily or per shift basis, depending on the blast cycle. The hole burden, hole accuracy, number of holes blasted at a time, the charge type and configuration, as well as the blast advance, fragmentation and resultant stope width need to be captured for analysis of the current operation and the selection of future blasting strategies.

Table II
Tonnages for a single 20 m hole based on the various stoping widths and burdens

Burden	Tonnage @ 30 cm burden	Tonnage @ 45 cm burden	Tonnage @ 60 cm burden
Stoping width 1 (50 cm)	8.2	124	16.5
Stoping width 2 (60 cm)	9.9	14.8	19.8
Stoping width 3 (100 cm)	16.5	24.8	33.0

Table III
Calculated cleaning times based on number of blast holes and cleaning distance

Number of holes	Cleaning distance (m)	Cleaning time (min)	Clean dist. (m)	Cleaning time (min)	Clean dist. (m)	Cleaning time (min)	Clean dist. (m)	Cleaning time (min)
1	10	11	34.5	38	59	65	83.5	92
2	10	22	34.5	76	59	130	83.5	184
3	10	33	34.5	114	59	195	83.5	276
4	10	44	34.5	156	59	260	83.5	368

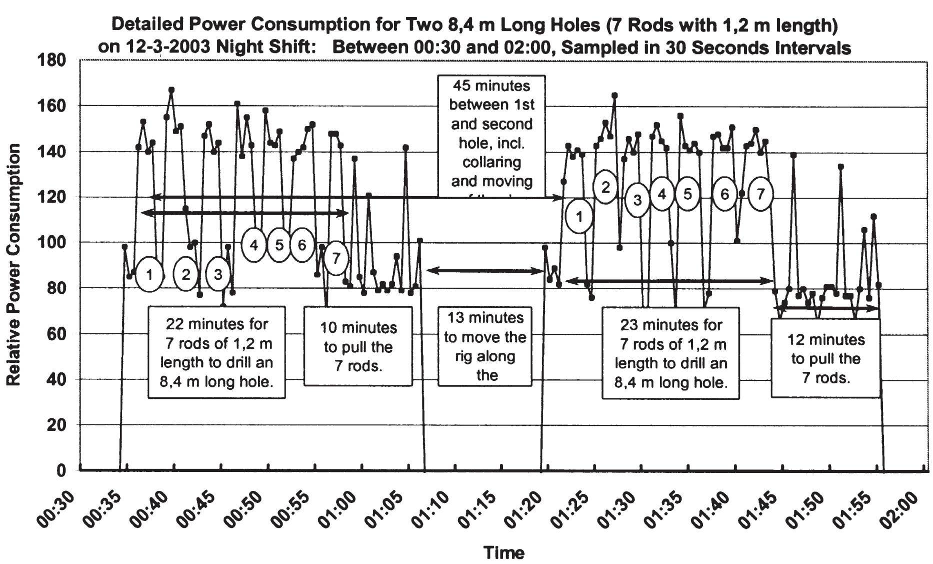


Figure 8. Typical profile of results obtained through data monitoring[9]

Conclusion

Site selection, support, drilling, blasting and cleaning are integral components of long hole drilling, yet each requires careful consideration, planning and optimization for successful long hole drilling implementation.

By following a selection process based on the parameters highlighted above, a practical long hole drilling mining method should be able to be implemented. For example, a favourable geotechnical condition for a potential long hole drilling site may show that 30 m long holes (i.e. 30 m long face) are possible, but the analysis of the unsupported span could reduce the practical hole length to say 20 m to prevent caving and the need for expensive gully support.

Further analysis of the rock properties (abrasivity and rock removal rate), hole diameter and drill/drifter capabilities would determine if the hole length of 20 m and the planned production rates can be achieved. If not, then alternative configurations need to be evaluated (i.e. change hole burdens, explosive type, reduce hole diameter and length, multiple booms/rigs, smaller drills, etc.) before the final mine layout can be determined. In addition, the requirement for drilling accuracy makes it essential that the rock be competent and uniform, with minimum stratification. The ability to drill accurate holes with lengths in the region of 10 m to 20 m is critical and it must be stressed that good set-up and alignment practices are required. Drilling accuracy affects the blasting results (and hence stope width) and the length of hole that can be drilled, and thus has a significant effect on the efficiency and economics of the mining operations.

As an indication of the potential economic benefit of in-stope long hole drilling, a comparison was made between a typical South African platinum mine's conventional mining production and costs, and the scenario if 10% of the area mined operated in-stope long hole drilling (Table IV). The results show a profit of R208.5 million for an integrated long hole and conventional stoping operation compared to a profit of R198.5 million for conventional mining only, giving a R10 million (5%) increase in profit. This potential benefit could justify the implementation of in-stope long hole drilling, providing the implementation programme was successful in ensuring stope width control (i.e. extracting reef with minimal dilution) and the required drilling rates were achieved.

To date this method of mining has not yet been implemented on a mine-wide or large-scale, full cost operation in South Africa. The operations currently in progress are on a small-scale-marginal mining basis, and the success of these operations will determine the future use of this yield-enhancing mining method.

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Table IV
Increase in profit due to integration of long hole stoping and conventional mining

Monthly Basis	Current	Proposed			
	Conventional	LHS	Conventional	Total	
Reef width	0.6	0.6	0.6	0.6	m
Stope width	1.2	0.6	1.2	1.2	m
Area mined	113463	11346	102116	113463	m^2/mth
Tons milled	569500	34170	511000	545170	t/mth
Mill yield	5.6	9.4	5.6	5.8	g/t
Pt Refined	101 891.4	10 261.9	91 629.5	101891	ounces
Pt Price	800.00	800.00	800.00	800.00	$/oz
Exchange rate	6.67	6.67	6.67	6.67	R/$
Revenue	543.69	54.76	488.93	543.69	R (m)
Stoping	1742.00	1429.79	1742.00	1722.43	R/ounce
Development	603.0	359.23	603.0	587.72	R/ounce
Other	1043.0	621.36	1 456.0	1403.69	R/ounce
Total cost per ounce	3 388.0	2410.38	3 388.0	3326.73	R/ounce
Total monthly costs	345.2	24.7	310.4	339.0	R (m)
Profit per month	198.5	30.0	178.5	208.5	R (m)
Return on costs	57.5%	121.4%	57.5%	61.5%	%

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