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Geotechnical analyses for open pit mining in areas of large-scale slope instability

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ABSTRACT: Varying degrees of overall slope instability have developed in the copper porphyries at several mines in sectors with high pit slopes. This instability can occur at relatively flat slope angles (25 to 35 degrees); reducing the slope angle to improve overall slope stability would require a very large stripping program. Slope monitoring and operational experience have demonstrated that displacements of high pit slopes tend to exhibit a regressive character. An economic alternative to flattening the overall slope angle is to modify mine plans and operational procedures so that mining can continue despite these displacements. Mining an active slope requires reliable rock mechanics models to predict the response of the slope to mining. This paper discusses the geological characteristics, deformation behavior, and current stability analysis procedures for modeling of several large-scale open pit slope failures. Operational responses to slope displacement are examined in the context of case history data, and the concept of displacement-tolerant mine planning is presented.

1 INTRODUCTION

Since the 1980's ongoing development of open pit mines at several porphyry copper deposits has resulted in the construction of overall pit slopes in excess of 400 meters in height; long-range mine plans are currently being developed for slopes up to 900 meters in height. Many of these slopes, such as those at the Chuquicamata Mine, are now in excess of 600 meters, and are among the largest rock structures ever to be engineered by man. Because of the size of many of the operations, there is often an economic incentive to design working pit slopes near the optimum overall slope angle; however, experience has shown that a significant amount of slope instability can be expected at these economically optimum slope angles. Since most large open pit operations possess procedural flexibility, considerable slope displacement can be tolerated, provided that the slope "failures" can be adequately modeled, and predictions made as to their size and behavior.

The means of predicting the expected number and tonnage of multibench structurally-controlled failures (wedge, step-path, plane shear and step-wedge) are well-developed, and probabilistic models can be constructed for use in predicting the number, size, and displacement resulting from these types of rock slope failures (CANMET). However, for overall slope instability due to rock mass yield in weaker rock zones, models of rock mass strength, developed stress fields, and rock mass displacement are still in the developmental stages. Where weak zones are present in the pit slopes, overall slope instability and displacement can develop, which will impact mine operational procedures and mine planning. If reliable and predictive rock mechanics models of strength, stress, and displacement can be constructed for the overall pit slopes, then mine plans capable of tolerating overall slope instability can be formulated, which will improve operational safety, increase production efficiency, improve mine economics and extend the mine life.

The geotechnical analysis of large-scale overall slope instability involves these primary areas of study:

1. slope monitoring and kinematic displacement modeling,
2. geological, geotechnical and hydrological analysis, and
3. stress, displacement and stability modeling.

2 SLOPE DISPLACEMENT ANALYSIS

2.1 Regressive versus progressive slope failure

Most post-failure models are primarily based on empirical relationships derived from slope monitoring data (Cruden and Mazoumzadeh 1987, Voight, Orkan and Young 1989). In the past, examination of displacement data has led to the development of two general displacement models: a progressive failure model, for which slope displacement will continue to accelerate to a point of collapse (or greatly accelerated movement), and a regressive failure model, for which the slope will decelerate and stabilize. Zavodni and Broadbent (1982) defined regressive and progressive failure stages for several large-scale open pit slope failures, and related these stages to failure geometry. This work has been expanded by Savely and Call into a useful description of failure characteristics and expected slope behavior (Figure 1).

A slope failure which is in a progressive stage tends to become less stable with time, and has the potential for sudden, large movements. A review by Ryan and Call (1992) of slope failures which have proceeded to collapse indicates that a wide range of pre-collapse velocities exist for progressive slope failures. Savely (1982) and Kennedy (1970) discuss operational and monitoring procedures for minimizing the impact of progressive slope failure on mining.

Regressive slope failures can occur when rock mass yielding is the primary cause of displacement. In the U.S., case histories of regressive slope failures in open pit mining have been documented in the literature since the early 1950's (Bisbee, Butte). Many rock mass movements in high open pit slopes tend to decelerate with time provided that the ratio of driving to resisting forces decreases with displacement. Such large-scale slope instability is most often due to a complex yield condition in which shear and tension failures of both joints and intact rock lead to failure and plastic flow of the rock mass. Surface and subsurface monitoring data from several slope failures show that displacements are extremely variable in direction and in magnitude. Inevitably, this leads to the necessity of a sophisticated monitoring program for large slope areas.

For the remainder of this paper we will discuss the behavior and analysis of regressive, overall pit slope failures which tend to stabilize with displacement and time. Geotechnical analysis and experience with several regressive slope failures demonstrates that mining can continue in areas of large-scale displacement provided that the failure is regressive in character, the failure mechanism is well-defined, monitoring procedures are established and enforced, and an effective slope management program is in place.

2.2 Modeling regressive slope failures

Slope monitoring data consistently indicates that most regressive slope failures occur in response to mining activity. A typical response is shown in Figure 2. As mining proceeds along a level within a zone that is in active yield, confining stresses are reduced, excess strain energy is induced, and displacement of the rock face is produced. These slope displacements can be either elastic or plastic depending upon the state of stress and strength of the near field rock mass. Monitoring data often demonstrates that slope displacement initially develops in lower strength, highly fractured rocks near the toe of an active mine slope. Toe displacement directions are typically at very low angles (5 to 20 degrees from horizontal). Displacements higher in the slope do not develop instantaneously with displacement at the toe, and can have highly variable directions. Since the slope is not acting as a rigid block, displacements will tend to develop in a time-dependent manner, resulting in differing rates of movement from point to point within the slope (Figure 3). This time dependency can also be demonstrated in regard to surface and subsurface displacement.

3 GEOLOGICAL ANALYSIS

Several geologic characteristics common to the documented large-scale regressive slope failures include:

1. low rock mass strength in the toe,
2. clay alteration of fractures, faults and wall rock,
3. a ubiquitous joint set that dips into the pit,
4. high angle faults or continuous joints that form back and side releases for the slope movement, and
5. saturated toe and compartmentalized ground water conditions.

3.1 Low strength rock mass

Unfortunately, thick zones of low quality rock are associated with the processes of ore emplacement. Regional scale tectonics, contact metamorphism and the release of volatiles from the magmatic melt all contribute to the degradation of the rock within and surrounding porphyry deposits.

These zones of low rock mass strength generally exceed 100 meters in thickness and have an RMR value of less than 40 and an RQD of less than 30. Rock mass characteristics range from low RQD zones with clay-filled fractures and moderate wall rock alteration, to pervasively clay-altered zones in which discrete structure has been obliterated by mechanical and hydrothermal actions.

3.2 Clay alteration

Varying degrees of clay alteration are present in low rock mass strength zones, as well as in transition zones that border it. In the low rock mass strength zone, clay alteration varies from a pervasive fracture filling with weak to moderate wall rock alteration, to complete alteration of the rock to clay by mechanical and hydrothermal processes.

Clay alteration extends beyond the low rock mass strength zone, but is usually confined to larger structures. In some cases, a transition occurs between the weak toe rock and fresh rock. Transition zones are characterized by clay-filled fractures and faults; however, the wall rock alteration is diminished and the RQD improves.

Clay adversely affects the stability of slopes for two reasons. First, the shear strength of clay is less than the rock on rock shear strength of the fractures. Clay alteration of intact rock and clay infilling of fractures reduces rock mass strength. Second, clay impedes the natural or induced drainage of the slope, resulting in higher pore pressures within the rock mass.

3.3 Ubiquitous joint set

Ubiquitous joint sets which dip into the pit at 35 to 45 degrees are a common characteristic of these regressive failures. In general, such structures do not daylight in the overall slope, but provide a plane of weakness within the slope.

This ubiquitous plane of weakness reduces rock mass strength in that orientation, and principal stresses tend to realign parallel to the ubiquitous structures with mining (Amadei 1987). This stress realignment enhances slope movement along the ubiquitous joints, and may actually extend them.

3.4 High angle back and side release fault structures

High angle faults that strike parallel to subparallel to the trend of the pit wall form a back release for downward slope movement. These structures diminish the rigidity of the rock mass, form discrete kinematic blocks, and compartmentalize ground water. In addition, the maximum principal stress aligns parallel to these structures on the upslope blocks due to the large contrast in rigidity between the fault structures and the surrounding rock. Clay-filled fault structures or continuous joint systems that strike from perpendicular to oblique to the trend of the pit wall provide side releases for the slope movement. These structures increase the degree of freedom for movement, diminish the rigidity of the slope and compartmentalize ground water parallel to the slope.

3.5 Ground water conditions

Ground water causes a destabilizing effect in all of these large-scale slope failures. In a freedraining slope, the hydraulic gradients are generally low, as evidenced by a gentle drawdown cone and a dry slope. However, clay-filled faults and fractures greatly reduce the permeability of the rock mass comprising the slope. High pressure gradients can occur as evidenced by seepage faces observed at significant levels above the pit bottom. In these cases, the drawdown cone is very steep, resulting in ground water being close to the pit face. This pressure creates a significant buoyant force on potential failure surfaces, which reduces the ability of the slope to resist shear. Driving forces increase further if the water pressure acts along a vertical back release.

Slope movements are quite sensitive to changes in water pressures. The onset of slope failure dilates the rock mass, creating additional storage. This added storage results in a lowering of the water level and a temporary increase in stability. In very dry regions, the lack of recharge may stabilize the slope or cause diminished slope movements for several pushback cycles. However, in areas of high recharge, the increase in stability is short-lived and the slopes should continue to be monitored for increased pore pressure.

4 FAILURE MODELING AND STABILITY ANALYSIS

The geologic characteristics listed in Section 3 of this paper often combine to cause the large-scale regressive slope failures observed. A suggested sequence of events that explains the slope displacements includes:

1. initiation of movement by active mining in the weak toe rock,
2. propagation of movement upslope due to successive release of kinematic blocks, and
3. deceleration to a more stable state.

4.1 Initiation of movement

Mining through low strength rock in the toe reduces the confining stress and creates an overstressed condition. The excess stress is relieved by movement in the toe of the slope and by stress transfer into rock that can carry additional load. The maximum principal stress is commonly horizontal at the toe, resulting in strong horizontal displacement.

Due to the pervasive clay fillings and wall rock alteration in the weak toe rock, natural or induced drainage is limited. Pore water pressures in this low permeability saturated rock can be high due to stress redistribution. The high pore water pressure reduces the ability of this weak rock mass to resist shear. In some cases, this material becomes amenable to drainage if discrete failure surfaces are developed.

4.2 Propagation of movement upslope

Displacements propagate upslope as confinement is reduced in successive upslope kinematic blocks. Upslope movements occur parallel to the non-daylighted joint sets and along high angle release faults.

A commonly observed characteristic of these failures is a differential shear along the high angle faults that form the back releases of identifiable kinematic blocks. Along a particular fault structure, the upslope block drops relative to the downslope block (Figure 4). This differential movement results from a significant vertical stress gradient across the faults. Due to the low strength and modulus of the fault structure, the maximum principal stresses on the upslope block align with the fault. This, coupled with the relief of lateral confinement due to downslope displacements, can result in a stress condition analogous to active earth pressure. The additive effects of the deep shear often result in the observed differential displacement along the release faults.

Clay-filled fault structures provide side releases for the upslope kinematic blocks. The shapes of the kinematic blocks range from rectangular to trapezoidal, depending on the obliquity of the side release faults. Trapezoidal blocks formed by non-daylighted wedges can be responsible for high stress concentrations in the toe. These stress concentrations may contribute to plastic failure of fair to good quality rock in the toe of the slope. Two-dimensional stability analyses do not adequately model this three-dimensional stress concentration.

The high angle release faults that form the kinematic blocks also tend to compartmentalize ground water. Ground water lowers the resistance of the rock mass to shear, resulting in a deeper zone of shearing along ubiquitous joint sets. In addition, water pressure acting on the back planes increases the driving forces.

Lowering pore water pressure in the upslope kinematic blocks greatly reduces overall slope displacements by reducing the driving forces and increasing the shear resistance. Additionally, stabilizing the upslope blocks imparts less stress on the weak toe material. Installation of dewatering systems usually pays dividends because slightly steeper slopes can be achieved and delays in mining operations caused by a moving slope can be reduced.

4.3 Reduction of slope movement

Slope displacements decrease as a result of:

1. halting or lowering the production rate in the weak toe,
2. lowering of the pore water pressure due to dilation of the displacing rock mass, and/or
3. displacing to a more stable geometry.

4.4 Stability modeling

Simple rigid block models are not appropriate for analyzing rock slope stability when the primary mechanism of failure is plastic (toe) to pseudo-plastic (shear along ubiquitous joints) yielding of the rock mass. Limiting equilibrium methods have limited application because they cannot predict strain or the extent of slope deformation. For these reasons, the combination of discrete element and continuum numerical methods are finding widespread application for the overall stability analysis of these high open pit slopes. The sophistication and efficiency of these models has improved rapidly. In the last decade, several software packages have been on the market that are valuable tools for analyzing slope behavior.

When using these numerical methods, it is critical to understand the mechanisms producing the slope failure, the strength characteristics of the rock mass and the pit hydrology. Once the model is defined, it is refined to correspond to the historical slope displacements and measured in-situ stresses.

Parameters that affect the model response include in-situ stress, rock mass strength, elastic properties, structure and hydrology. Because displacements are often not elastic, superposition is not valid, and a careful simulation of historical mining is required for stability analysis. A history match between the model and the slope can often be obtained through several stress paths. Therefore, a large geomechanical database must be developed to define the parameters used in the model. This database should include the following:

1. rock types observed from surface and subsurface geologic mapping,
2. block sizes:
   a. cell mapping,
   b. RQD logging of core, and
   c. surface classification mapping of weak rock masses,
3. intact shear strength:
   a. uniaxial compression tests,
   b. Brazilian disk tension tests,
   c. triaxial compression tests, and
   d. point load tests,
4. fracture shear strength:
   a. small-scale direct shear tests of core, and
   b. large-scale direct shear tests of rock blocks,
5. orientation of geologic structure:
   a. rock fabric mapping,
   b. oriented core, and
   c. geologic mapping of major structures,
6. hydrological characteristics:
   a. geologic mapping and drilling to define character of water-bearing rock,
   b. water level measurement in exploration and geomechanical drill holes,
   c. reporting wet blast holes,
   d. reporting seeps observed on the slope,
   e. slug testing of exploration and geomechanical holes to provide a rough estimate of transmissivity, and
   f. pump tests with observation holes for definition of transmissivity and storage in more homogeneous aquifers, and
7. in-situ stress measurements.

Rock mass strength and elastic properties, and hydrologic parameters are defined from the data mentioned above. These data are used to zone the rock into discrete domains that possess similar engineering characteristics. Quantitative methods have been developed to classify rock masses according to rock type, block size distribution, intact rock shear strength, fracture shear strength, orientation of the geologic structures, pore water pressure, and in-situ stresses. The RMR (Bieniawski 1993) and Q-system (Barton et al. 1974) rock mass classifications have been used extensively to choose appropriate underground mining methods and to estimate support requirements for underground openings, settlements in rock mass foundations and the rock mass shear strength.

Of particular importance in constructing numerical models of pit slopes is the definition of the rock mass shear strength and the elastic modulus. Several empirical classification schemes have been developed to estimate the rock mass strength and elastic properties. Numerical slope modeling experience has shown that the rock mass strength and elastic properties derived from these classification schemes do not always provide model responses that match the measured slope displacements.

More rigorous methods of determining the rock mass strength characteristics are needed. The use of three-dimensional numerical methods to model the mechanical response of the rock mass to various loading conditions show some promise. To be effective, these models must accurately portray the intact rock properties, the orientation of stress fields, the orientation and spacings of discontinuities, and the strength and elastic properties of the discontinuities and/or discontinuity fillings.

Although numerical methods have proven valuable in defining the failure mechanism of these large-scale slope failures and predicting the response of the slope to future mine plans, they are not adequate for modeling the combined responses of mechanical loading, deformation, and water flow. The large size of these slope models places a constraint on the number of sensitivity analyses for various static water conditions which can be performed in a reasonable period of time. Future versions of the numerical methods must represent the mechanical-hydrologic interaction of large pit models efficiently. Topics that updated numerical methods should address include:

1. consolidation,
2. efficient establishment of hydraulic gradients in complex hydrological conditions, and
3. change in hydraulic gradients resulting from rock mass dilation during failure.

4.5 Displacement rates and overall slope angle

A range of overall slope angles can be excavated in regressive slope failures. Steeper slopes experience greater velocities and larger overall displacements. However, there is a maximum limit to the angle that can be excavated before a progressive accelerating slope failure will occur. Steepening the upslope geometry beyond this critical angle results in driving forces that are greater than the resisting forces in the upslope kinematic blocks. These upslope blocks can drive the toe to an accelerated failure condition. One of the fundamental roles of stability modeling is to determine the critical slope angle and identify whether this accelerated condition is beginning to develop. Figure 5, a plot of model velocity histories for three slope geometries, illustrates the use of modeling in defining the critical slope angle.

5 OPERATIONAL PROCEDURES TO MINIMIZE THE IMPACT OF DISPLACEMENT ON MINING

Because of the procedural flexibility in most large open pit mines, slope displacement does not necessarily constitute "failure" from a mine management standpoint. This relationship between theoretical and operational slope failure in mining has been discussed (Munn 1985). In particular, the real hazard to mine operations is often the potential for greatly accelerated movement occurring near equipment and personnel. If this can be mitigated, a significant amount of slope displacement can often be tolerated with routine mine operations such as dozing and additional shovel shifts for cleanup. Mining in areas of large-scale slope instability generally results in unsteady production. Provided that a mine slope failure is regressive in character, slope displacement can be controlled using specific operational procedures: dewatering, additional stripping, control of the excavation geometry, and control of the excavation rate. The effect of these controlling measures can be assessed and predicted with geotechnical analysis.

5.1 Step-out versus unload

At Twin Buttes during the past five years, it has been demonstrated that periodic, small (less than 20 feet) step-outs into the pit could effectively decrease slope displacement in an area of historical slope failure (Ness 1992). There is often a trade-off between the cost of cleanup or additional stripping and the value of ore lost due to a step-out. There also may be an optimum location for a step-out in a pushback, and this can be defined by the geotechnical engineer with stability analysis.

5.2 Controlling excavation rates

Control of the mining rate is another means of maintaining displacement velocity to minimize its impact on mine operations. This technique is difficult to implement, and requires significant flexibility from mine operations. Once displacement velocity of a slope region reaches a limit defined by the rock mechanics staff (through analysis and experience), mining is discontinued until an acceptable relaxation limit is achieved. Savely (1993) discusses how this approach is optimized by maximizing operational efficiency in relation to the moving slope area. At several properties, short mining periods and shorter delays have been found to be more cost-effective than long mining periods and longer delays. This can also be demonstrated mathematically with displacement modeling (Figure 6). Since the vast majority of accelerating and decelerating displacement curves for rock slopes approximate either an exponential or a power function, the relationship between the time spent in excavation and the time required for relaxation is not linear. The higher displacement rates associated with longer mining periods require a substantially greater proportion of time for deceleration. Stability analysis and slope monitoring data can be used to assess optimum extraction rates.

5.3 Pushback width

Optimum pushback widths can be defined from a geotechnical perspective as well as a mining perspective. Practical mining experience suggests that narrow pushbacks within failed slopes are difficult to maintain. This can also be demonstrated numerically with detailed stress and energy analyses of rock slopes. The role of the geotechnical engineer should be to determine whether there is an advantage in changing the pushback geometry due to the state of existing slope instability, or the location of a major geologic structure.

Stability analysis of overall slope failures using continuum models demonstrates that stress concentration occurs at the toe of slopes as they are excavated; this has been validated by in-situ stress measurements at several mines. In the case of the excavation occurring within a plastic zone, a change in the stress and energy state occurs, which results in slope displacement. The stress path for excavation is a combination of lateral extension and axial compression. Strain energy for underground excavations has been analyzed in detail by others (Salamon, Farmer), and similar methods can be applied to pit slopes which are at yield. In general, smaller excavations result in less energy changes and displacement than larger excavations. However, because of the stress concentrations that develop in the toe of a slope, strain energy potential within the slope is not uniform. This leads to a nonlinear relationship between pushback width and strain energy (or maximum shear stress) for varying sizes of incremental mining cuts. When narrow pushbacks are mined, excavation takes place within the zone of stress concentration and the resulting displacements are typically a high percentage of the overall pushback width. This renders narrow pushbacks difficult to maintain, and results in either excessive time spent in additional slide cleanup, or frequent unplanned step-outs into the pit. If step-outs must be taken too often, it may not be possible to complete the pushback, and ore production may be lost. Additionally, if strain softening of the rock mass occurs with displacement, there is considerable geotechnical incentive to "mine out" the existing failed rock as much as possible in order to take advantage of the higher strength associated with a less disturbed rock mass.

For maximum production efficiency, a pushback must be wide enough for double spotting trucks and three lanes of traffic. For a normal rectangular cut with the digging face at right angles to the pit wall, the inside corner is relatively inefficient because a wider swing or single spotting is required. The outer edge is similarly inefficient, in addition to having a less than complete digging face. Thus a narrow pushback, with the majority of the digging time in the inside corner and the outer edge, will produce fewer tons per shovel shift (Figure 7).

Where there is overall slope displacement, the effective pushback width will be reduced by bench sluffing. For example, if a 15 meter high bench dug at 63 degrees sluffs to a 36 degree angle-of-repose, 6.5 meters of the pushback width will be lost. If sluffing of the working bench and the bench below occurs in the same section, the effective width of the pushback would be reduced by 13 meters. Thus, in this case, pushbacks should be designed at least 13 meters wider for a displacing slope than for a stable slope.

In particular, the following sequence is to be avoided:

1. A production schedule is planned with a minimum width pushback and full shovel and truck efficiency.
2. Slope displacement reduces the effective width of the pushback and the tons per shovel shift decreases.
3. At the lower production rate, ore will not be uncovered in time so the pushback width is decreased in an attempt to uncover ore.
4. The reduction in pushback width results in one-way traffic, more single side loading and delays from tension crack offsets. This further reduces the tons per shovel shift.
5. Steps 3 and 4 are repeated until the pushback is not minable.

Another factor that must be considered in production scheduling is the horizontal displacement. If the slope experiences horizontal displacement on some level between one pushback and the next, additional material will have to be mined to achieve the design toe.

5.4 Slope dewatering

Dewatering has been demonstrated in several cases as an effective control for displacing slopes (Argall and Brawner 1979). Cost-benefit analysis indicates that dewatering programs are one of the most cost-effective mechanisms available for improving slope stability for both stable and unstable ground. In many low permeability rock masses, reduction of pore pressure prior to failure is difficult with conventional methods, but is readily achievable after failure. Dewatering below the pit bottom is often required to achieve an acceptable pore pressure condition for the overall slope, and an analysis of aquitards and compartmentalization of ground water must be completed by the geotechnical engineer to ensure that the proper level of effort is focussed on dewatering.

6 CONCLUSIONS

Improvements in continuum and discrete element models have enhanced the capability of engineers to design high overall pit slopes which often experience significant slope instability. These models can be used to predict the development of rock mass movements which previously could only be evaluated empirically. As the sophistication of the stability models improves, more data is required to extend the model; particularly high quality data is necessary for its calibration. Better estimates of the state of stress and pore pressure are needed in addition to improvements in rock mass strength prediction techniques. In particular, stability models should continually be developed which are capable of modeling dynamic processes.

Provided that enough data can be gathered to acquire a firm understanding of the failure mechanism, stability models can be developed which will enable engineers to provide operations personnel with reliable predictions of the impact of mine excavation on the stability of the overall pit slopes. If pit slope analysis can be developed in a spirit of cooperation between the geology, engineering and planning staffs, geotechnical analysis can be run in concert with mine planning to create an optimum slope design.

Experience with several large open pit slope failures indicates that mining can be successful in areas of large-scale slope stability if unsteady production can be accepted in the unstable pit sectors. If such fluctuating production can be tolerated, specific operational procedures can be used to minimize the impact of slope displacement on mine operations. Geotechnical analysis can be used to identify the procedures required to allow mine operations to work within or near areas of large-scale slope instability. Such analyses can greatly improve the possibility of successful mining in these areas.

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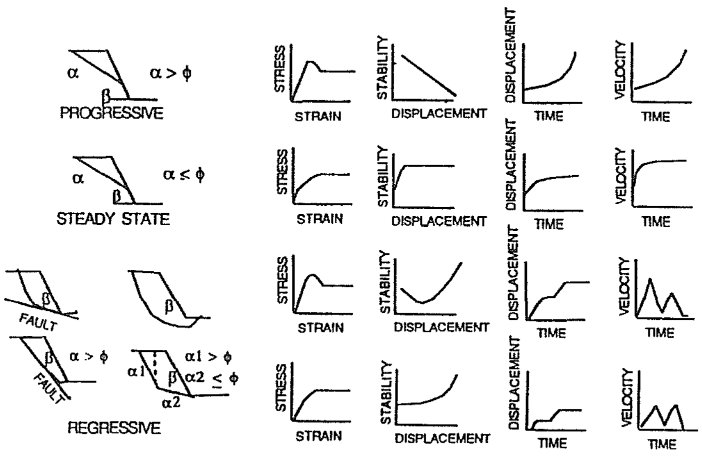


Figure 1. Failure conditions in relation to the stability model.

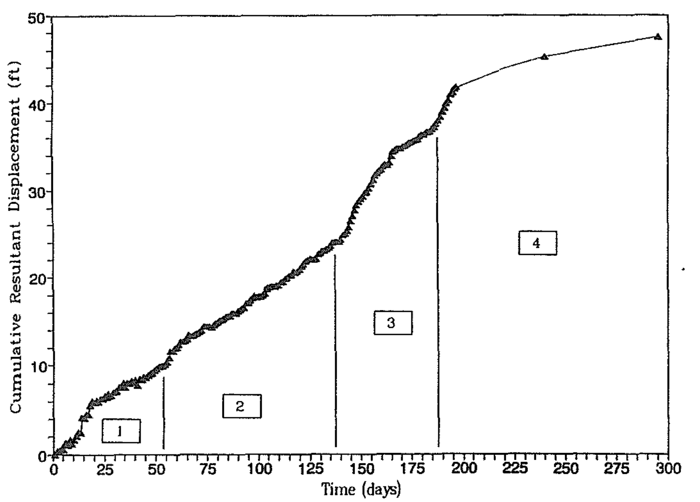


Figure 2. Cumulative resultant displacement of a surface station showing four regressive displacement cycles.

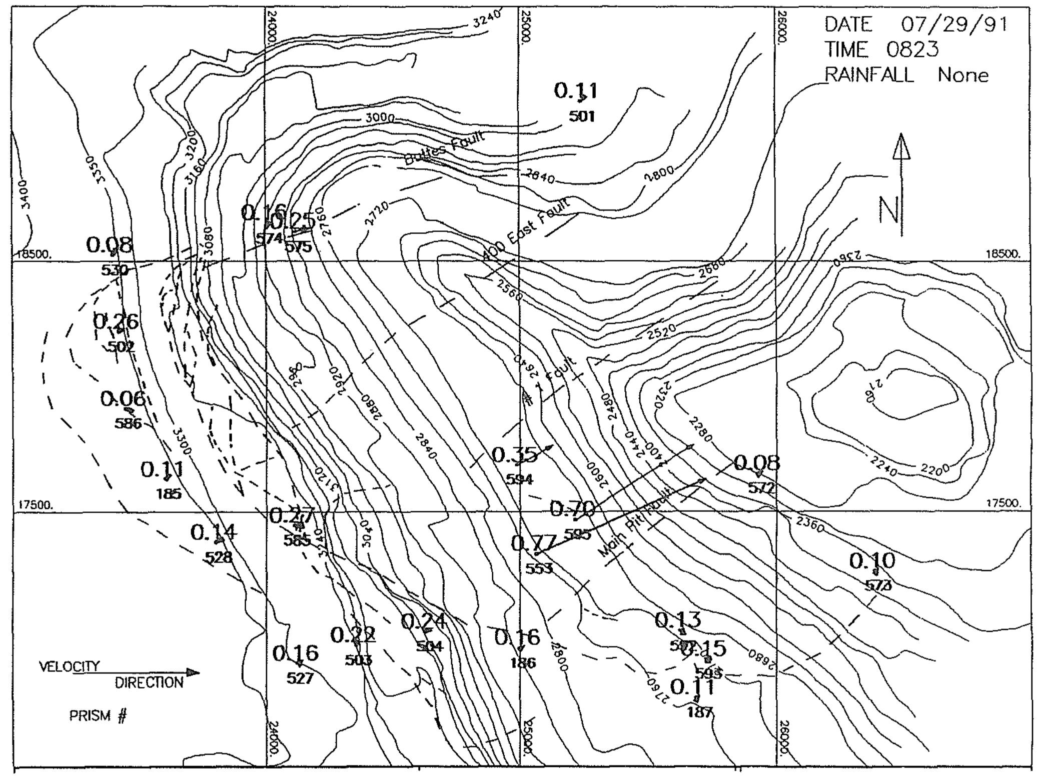


Figure 3. Variable slope velocity of a large-scale slope failure.
Note high velocity directly above the active mine bench at the 2680 level.

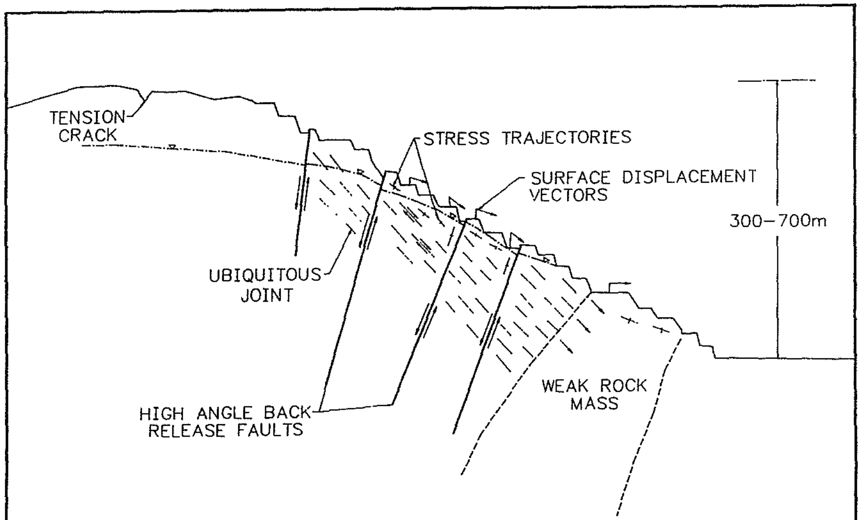


Figure 4. Failure mechanism and geologic conditions for large-scale regressive slope displacement.

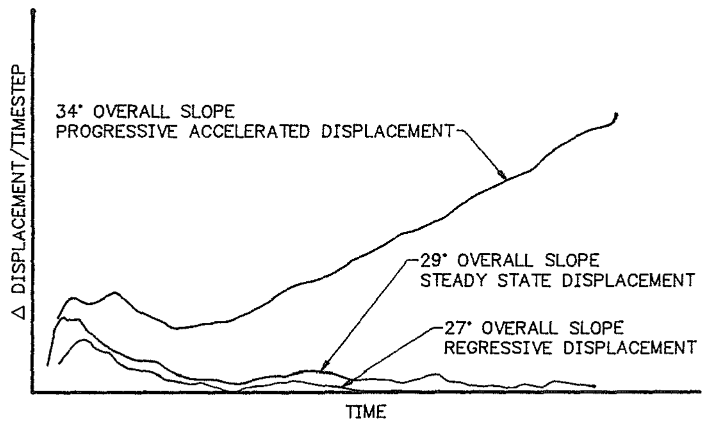


Figure 5. Velocity histories for three slope geometries.

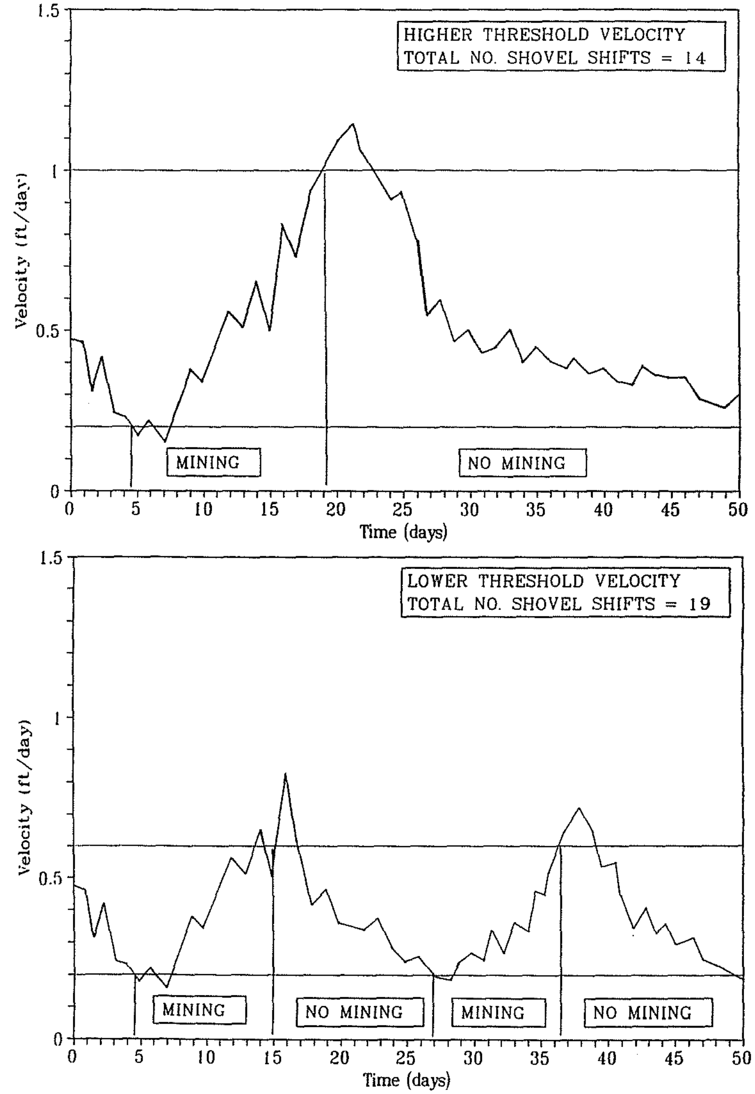
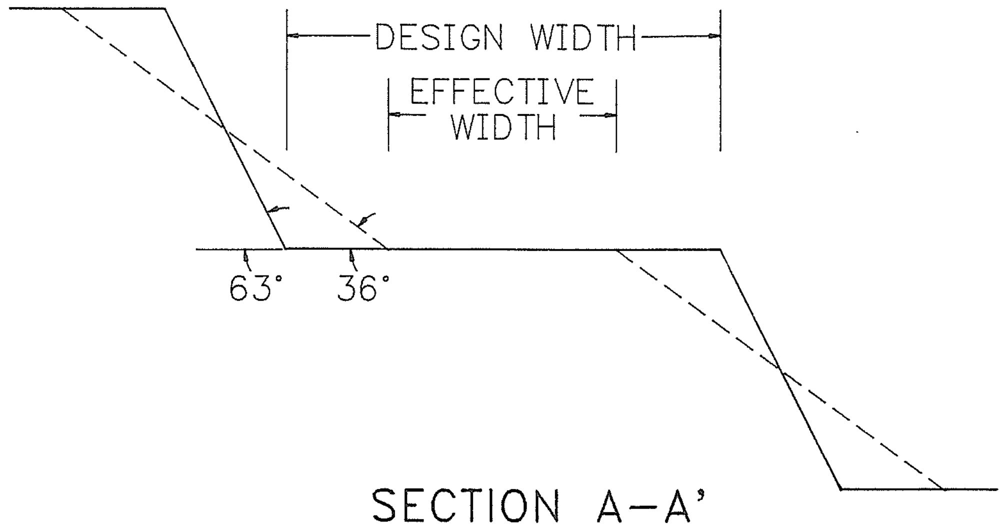


Figure 6. Controlling slope displacement velocity by selective mining in an unstable area.



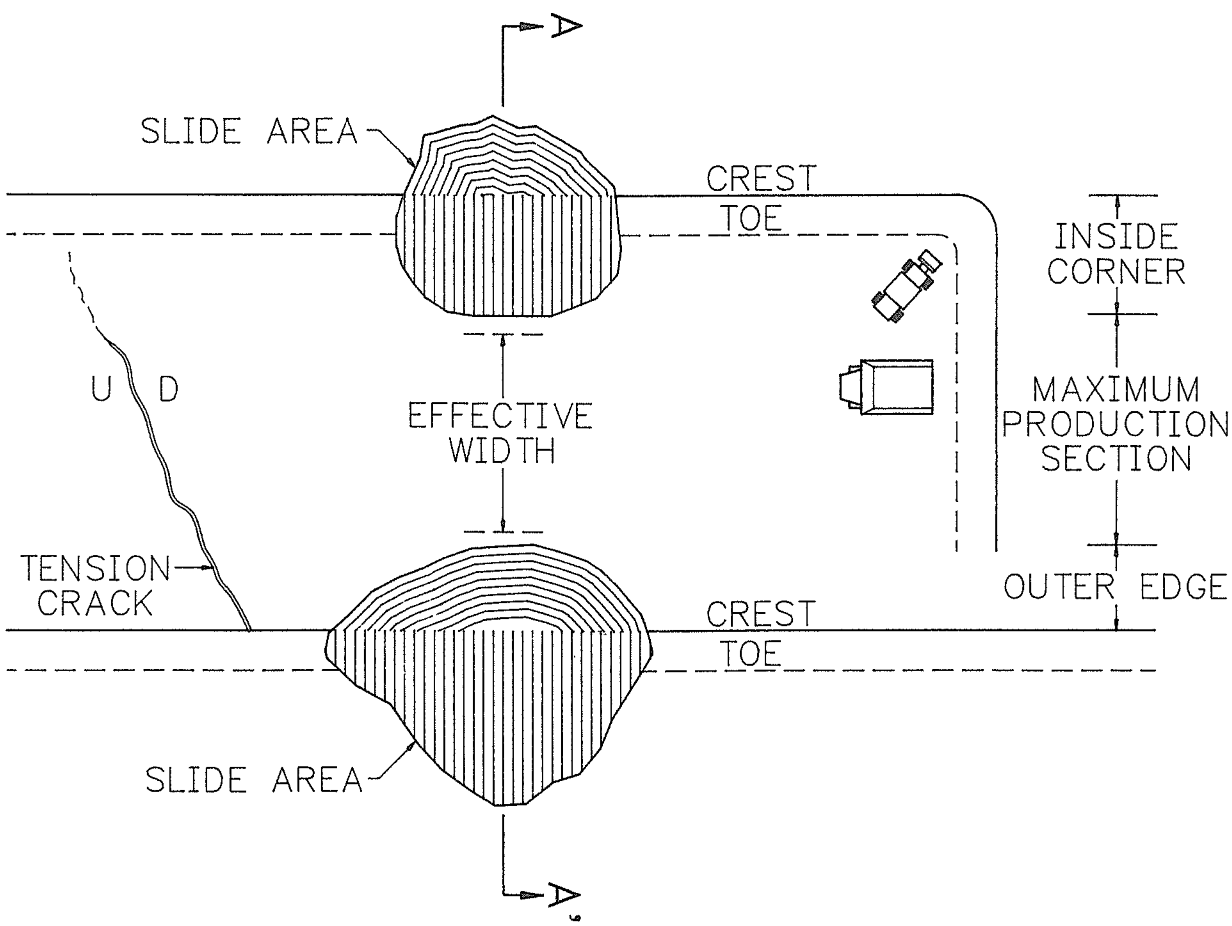


Figure 7. Operational considerations for pushback widths.