

STOPE DESIGN CONSIDERATIONS USING ROCK MASS CLASSIFICATION TOOLS AT THE XSTRATA ZINC GEORGE FISHER MINE

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ABSTRACT

The development and use of rock mass classification tools have been a key component for improving stope design over the past 4 years at the Xstrata Zinc George Fisher Mine in northern Queensland, Australia. In 2003, stope extraction data from 3 years of open-stope mining provided an excellent situation to review the assumptions in the feasibility study. Extracted stope profile information, drillhole geotechnical data, underground observations, and oral and written communication were used to develop a thorough stope reconciliation performance database. Without collecting the back analysis data and presenting the data in a usable format, engineers are left to debate opinion instead of engineered judgment. This can lead to biased and uninformed design parameter choices with the potential to repeat poor design. This paper demonstrates some effective, practical examples of empirical data collection where rock mass classifications tools were developed and used to create improved confidence in predicting stope stability and failure profiles. The work contributed to design changes that resulted in a reduction in stope hangingwall dilution and an increase in head grade while continuing to ramp up production from 2003–2005.

INTRODUCTION

This paper describes the development and use of rock mass classification tools in stope design over the past 4 years at the George Fisher (GF) North Mine. The deposit is located 22 km north along strike from the Mount Isa Lead Mine. A joint stope dilution study between the Xstrata Zinc George Fisher Mine (Figure 1) and the University of Saskatchewan, Canada, began in October 2003. The objective was to create a thorough understanding of hangingwall (HW) overbreak using a sound geological engineering approach that focused on data

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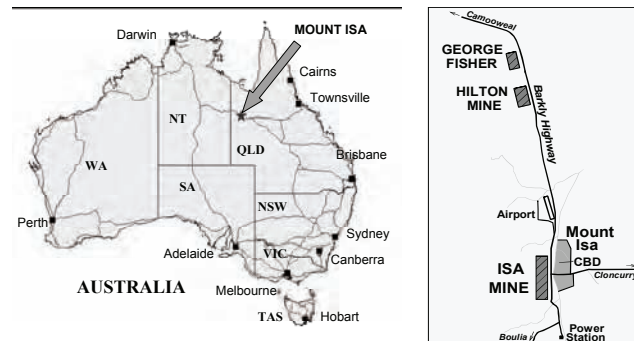


Figure 1.—Location of George Fisher Mine.

collection of historic stope performance, rock mass classification, and underground observations of D ore body. The evolution of the work has created a better understanding of stope HW stability and created a change in mining methodology. The work contributed to a reduction in annual stope dilution from 14.4% (2003) to 6.3% (2005), an increase in zinc grade from 7.4% to 8.7%, and an increase in production from 2.1 to 2.6 Mt [Capes et al. 2006].

BACKGROUND

Geology

The George Fisher Mine contains two similar deposits located approximately 2 km apart with an estimated total resource of 127 Mt at 100 g/t silver, 5.3% lead, and 9.2% zinc (June 2005). The GF North deposit makes up approximately two-thirds of the current production, with the remaining ore coming from GF South (previously known as Hilton). The GF North deposit, the focus of this study, is composed of a series of stratiform ore bodies striking near north-south and dipping west between 30° to 70°. The sphalerite-pyrite-galena-pyrrhotite type mineralized areas are separated by various thicknesses of bedded shales and siltstones [Forrestal 1990].

D ore body is currently the key area of extraction, grading at approximately 5% lead, 9% zinc, and 100 g/t silver with the mineralization being unequally distributed through the ore body, including a high-grade massive sulfide layer sometimes defining the ore/HW contact. Typically, the D ore body HW rock is composed of a siltstone marker rib followed by a package of pyritic shales

(5%–20% fine-grained pyrite) and a series of very fissile black shales (Figure 2). This is topped off with a variably thick mineralized lead-zinc lens. A section of siltstone separates this lens from the massive pyrite marker and massive sulfides, which define the start of C ore body. The thickness and rock mass quality of each package between D ore body and C ore body vary quite significantly along strike and dip as a result of past brittle and ductile deformation from faulting and folding events. A large number of geological structures affect the George Fisher deposit with varying degrees of offset, ore body rotation, drag folding, and metal redistribution [Grenfell and

Haydon 2006]. The two main types of faults are cross-cutting north-east trending faults and bedding parallel faults having their own individual characteristics and zone of influence.

Stope Layout and Design

The mining method at GF consists mainly of transverse open-stope mining where the ore bodies are greater than 10 m in thickness. Transverse stopes are mined from a footwall drive access either on a 30- or 60-m sublevel spacing giving consideration to the local rock mass quality and stope shape relationship. Primary stopes are mined until uneconomic conditions exist and then are filled with either a cement aggregate fill or a paste fill material. A secondary stope is typically mined once the adjacent primaries have been mined to one 30-m sublevel higher than the secondary stope and the fill has been allowed to sit for 28 days and gain sufficient strength to act as a sidewall during extraction (Figure 3). The original “15–20” design (17.5-m crosscut centers), primary and secondary stope strike lengths, respectively, was changed in 2003 to a “10–20” design (15-m crosscut centers). This eventually became a floating design based on local rock quality where primary stopes could be up to 15 m on strike considering the cost of cemented backfill, local ore body width, HW stability, and effects on the mining cycle. In addition, the drill design of stopes within the George Fisher Mine underwent a change in 2003 when D ore body HW drives were eliminated and transverse stope crosscuts were used as the only access for D ore body extraction. One of the main motives behind this design change was

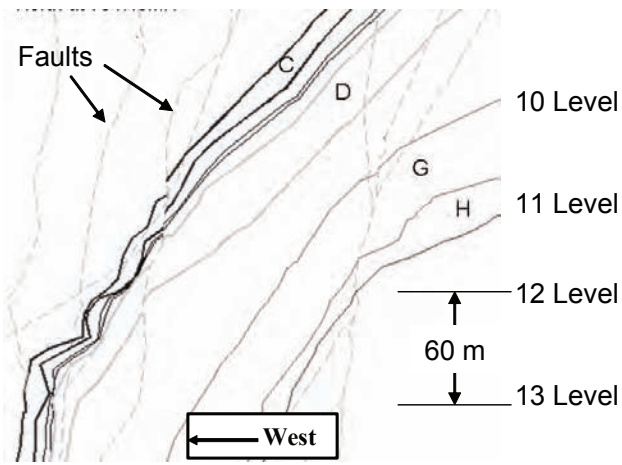


Figure 2.—Typical cross-section of George Fisher ore bodies.

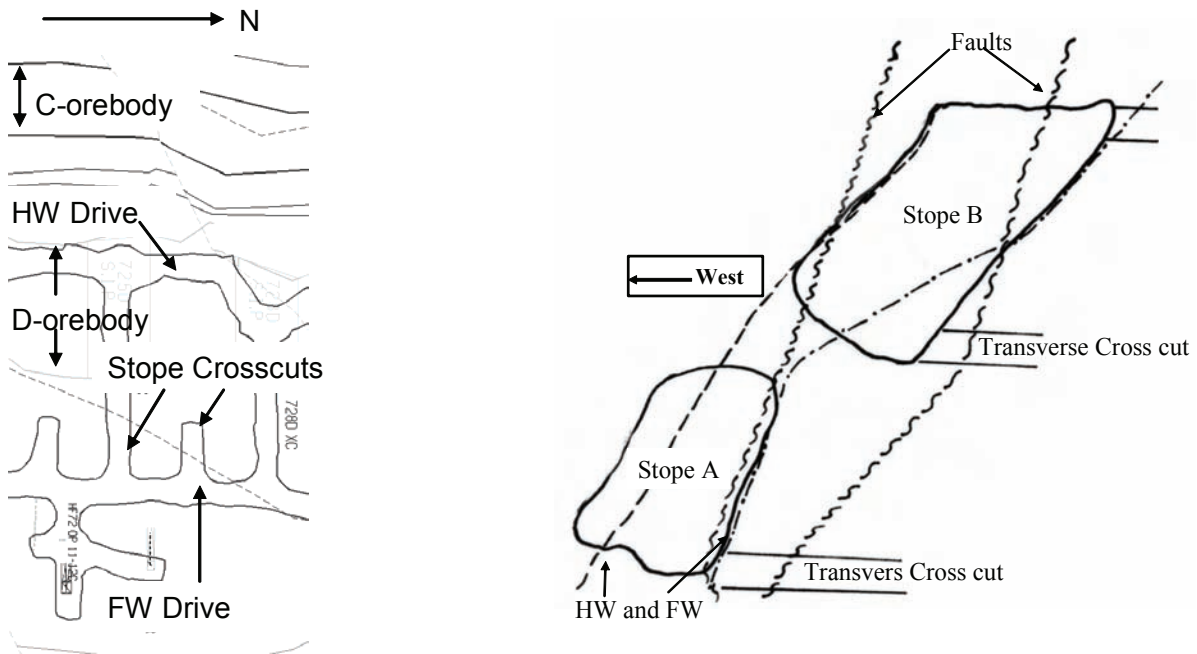


Figure 3.—George Fisher mining layout (plan and cross-section views).

the existence of problems with the stope hangingwalls and adjacent hangingwall drive failures as a response to stoping. The elimination of HW drives created the need for additional diamond drilling of the ore body to delineate the stope extraction wireframes. This addition of drilling and subsequent geotechnical logging has provided substantial relevant information for stope design and reconciliation purposes.

Numerical modeling provided an adequate set of rules for mine-wide sequencing [Beck 2003], which, when adhered to, lead to a sustainable mining method. The suggested guidelines included minimizing overbreak in primaries to avoid connection of voids when mining secondaries, tight-filling stopes, eliminating triple-lift pendant pillars, decreasing stope cycle time, and creating a better sequence to minimize stress effects on extraction. However, even with the mine-wide changes and mine-wide sequencing rules, a set sublevel spacing and crosscut width in poor-quality HW rock mass continued to result in unpredicted major individual stope HW failures and subsequent production inefficiencies. Thus, a thorough understanding of individual stope performance was required in order to achieve further positive design change.

DEVELOPMENT AND USE OF ROCK MASS CLASSIFICATION TOOLS AND EMPIRICAL METHODS FOR DESIGN

Overview

During late 2003, stope extraction data from 3 years of mining provided an excellent situation to review the assumptions in the feasibility study. Stope profile information based on data acquired using the cavity monitoring system [Miller et al. 1992], drillhole (BQ size) geotechnical data, underground observations, and oral and written communication was used to develop a thorough stope reconciliation performance database. On-site research, coupled with subsequent design trial and implementation, resulted in the development of a model that showed a relationship between rock mass quality, span, and resultant extraction profile. The goal of this model was to use it as a template and continually update the knowledge with underground observations and reconciliations of case histories. The model represented significant time, discussion, and research, but with more data, the model would evolve and lead to continued improvements in the understanding of HW behavior and design of stopes.

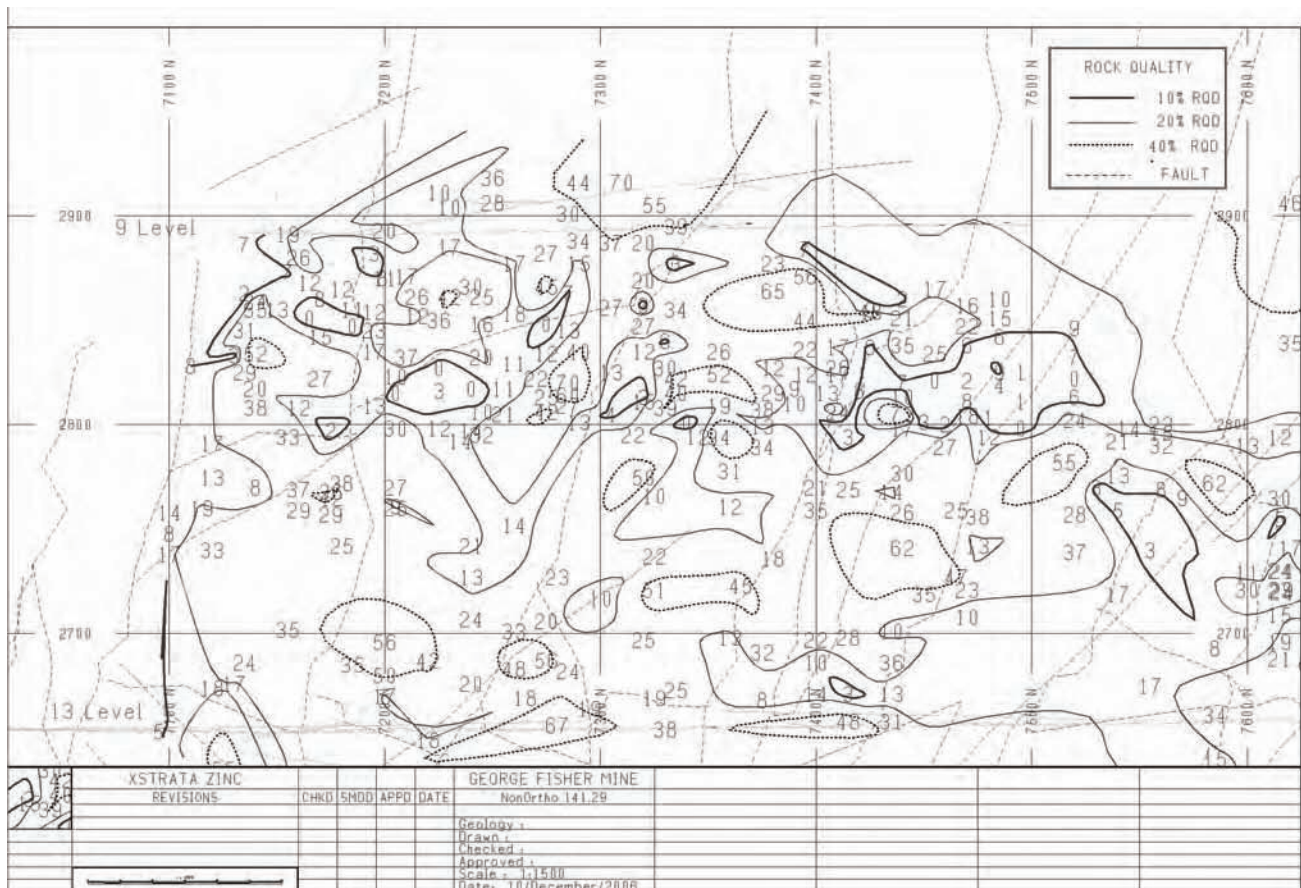


Figure 4.—RQD long section contoured.

Phase 1: RQD Long Section

The first step of the study was to create a long section of the available geotechnical data. The comprehensiveness and density of geotechnical data depend on the year in which the drillhole was logged. Most holes after 1999 have RQD data available (BQ: 36.5–40.7 mm) as recommended by Hadjigeorgiou [1999]. Rock Quality Designation (RQD) [Deere 1964] was originally developed for drill core ≥ 54 mm, although the “BQ RQD” is used to compare site data. The RQD data were averaged for the first 5 and 10 m of rock into the hangingwall and placed on a long section showing significant variation along strike and dip. The 5-m average RQD long section (Figure 4) has evolved from a scrappy, hand-contoured, coffee-stained desk map to a sharable plan able to be accessed by the technical services team on the mine design software. The original section was hand-contoured while the up-to-date section was contoured by triangulating the RQD values on the mine software as topographic surfaces between points and joining equal elevation points as contours. There is a substantial increase in RQD data density above the 2,800 elevation due to the removal of the HW drives and requirement for diamond drilling and subsequent geotechnical logging.

Phase 2: Empirical Methods

The next step was to gather additional averaged data for use in empirical design methods, such as the modified stability graph method [Potvin 1988] and dilution graph method [Clark and Pakalnis 1997], which require a single estimate of rock mass properties for the surface being analyzed. This involved acquiring geotechnical parameters such as joint condition, joint alteration, and joint roughness used in the Q-system [Barton et al. 1974]. Based on many underground observations and stope reconciliations, the rock in the HW area was assigned three broad categories (Figure 5). Each stope was assigned values within a category and plotted with actual ELOS values using the dilution graph format. ELOS (“equivalent linear overbreak/slough”) is defined as the volume of the HW overbreak divided by the stope HW surface area [Clark and Pakalnis 1997]. ELOS is a useful tool for mine planning because it provides the ability to quantify a diluted stope shape to enhance mine scheduling purposes. The dilution graph was modified with new ELOS lines calibrated to data collected for stoping in poor-quality rock masses where minimal data have been presented except for Capes et al. [2005] and Brady et al. [2005] (Figure 6). The “calibrated” curves on the modified dilution graph (Figure 6, right) have been used to predict HW overbreak in both GF North and South operations for the last few years. The calibrated curves, which need to be statistically verified as a final part of the study, are an effective tool for local stope design as they contain a large amount of data in

the required hydraulic radius (H.R.), modified stability number (N') ranges.

However, further work was required to examine why some cases did not agree with the design approach based on the average rock mass conditions. When Category 3 rocks (RQD < 10%–20%, $N' < 1$) were analyzed, the failure prediction did not seem to correlate with the ELOS predictions based on a 5-m average of HW conditions. Two such examples included having a 6-m ELOS in a primary stope where <0.5 m was expected ($N' = 0.5$, H.R. = 3) and an 13-m ELOS where approximately 4 m was expected ($N' = 0.8$, H.R. = 7). For category 1 (RQD > 40%, $N' > 6$) and category 2 (RQD 20%–40%, $N' = 1$ –4) rocks, the majority of cases of ELOS prediction were within acceptable error for stope prediction requirements, but an improved model was desired to examine the cases that did not fit.

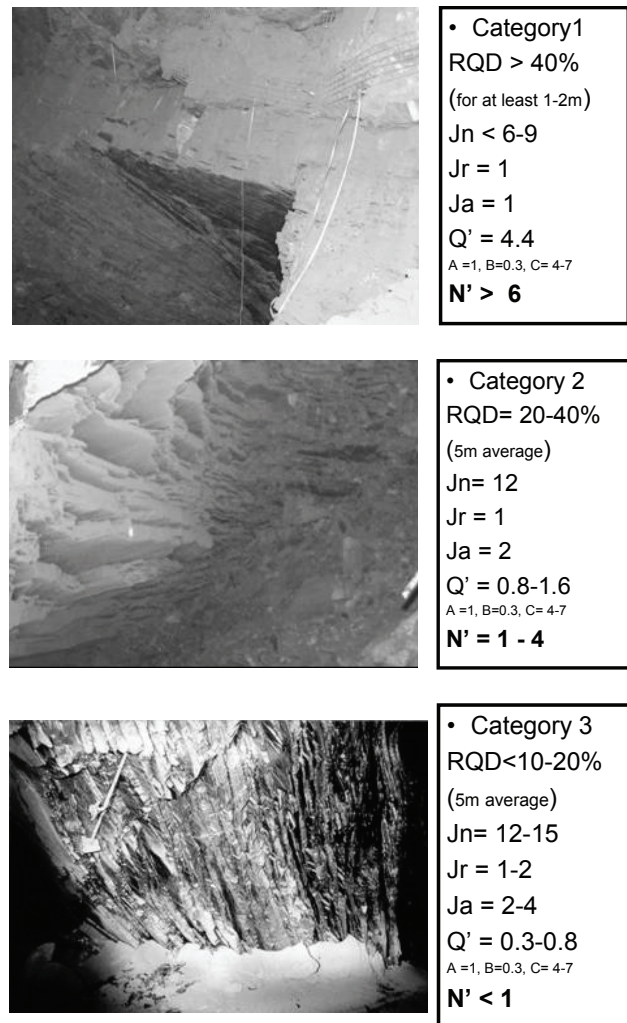


Figure 5.—Examples of HW rock categories at George Fisher Mine.

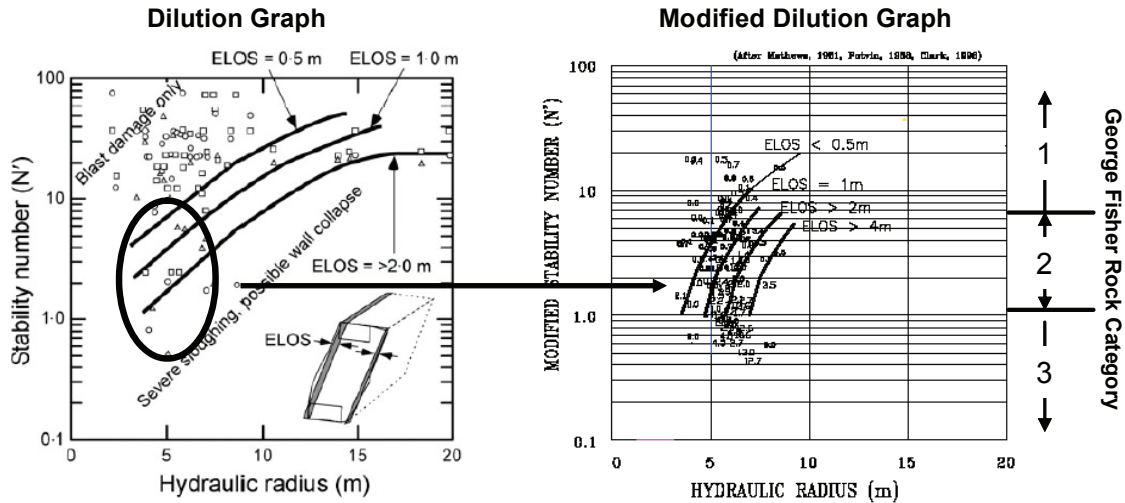


Figure 6.—Dilution graph (from Brady et al. [2005]) and modified dilution graph with additional data from poor-quality rock masses (after Capes et al. [2005]).

Phase 3: Rock Category and Failure Profile Relationships

The next step was to investigate the additional factors that have been documented to relate to HW overbreak from a comprehensive literature review and studies conducted at the George Fisher Mine. These additional factors, including faulting, stress, blasting, undercutting, and time, were identified to examine how or if they played a role in erroneous ELOS prediction. During this next phase of the study, interesting relationships were observed between failure shape, rock quality, and span when the RQD drillhole data were overlain as logged intervals on HW failure profiles in the mine design software. Thus, the methodology was changed to first investigate these novel relationships before examining the additional factors.

RQD cross-section plots were created for individual stopes and provided valuable data for understanding the variability of rock quality near the HW/ore boundary (Figure 7). The RQD plots coupled with underground observations created the idea of different failure profiles (Figure 7) for the different rock mass categories shown in Figure 5. Areas where the RQD < 10%–20% typically failed into the next rock mass category without arching. Areas with RQD 20%–40% would arch to become stable within span constraints or would change profile when a different rock category was intersected. Stability existed in other stopes where the stope failed to a composite beam or plate of rock in which the RQD > 40% for at least 1 m. These relatively thin zones of higher RQD rock define a more stable domain, and the position of this domain was found to define the extent of failure, within span constraints. These stable domains were not always a consistent lithology, but could often be defined as pyritic shales, siltstone beds, or a narrow mineralized area. Data were collected to create a design tool demonstrating the

stability relationship between the composite beam thickness (meters of BQ core >40% RQD) and span (Figure 8). Figures 9–12 demonstrate four case histories of the model incorporating the three rock categories. Recognizing the changing rock mass condition with

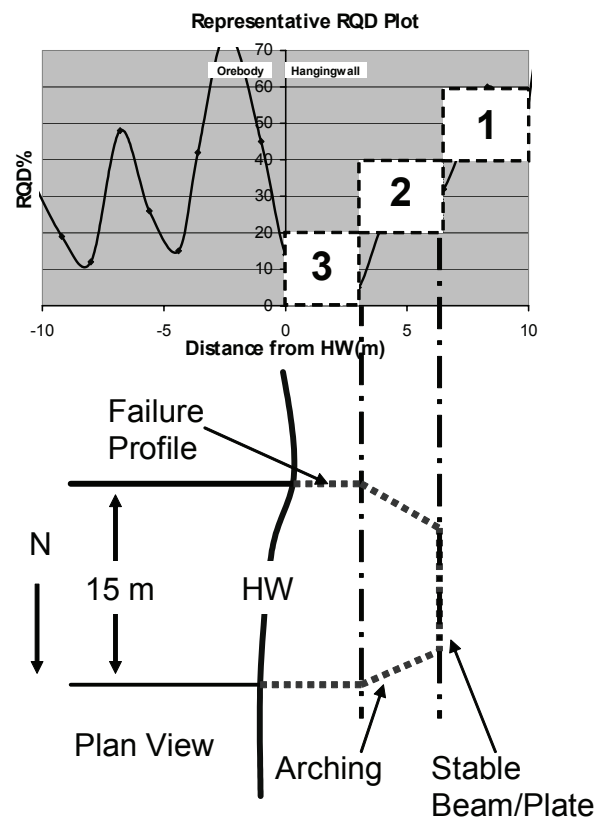


Figure 7.—RQD plot and representative failure shape profiles based on rock category and span.

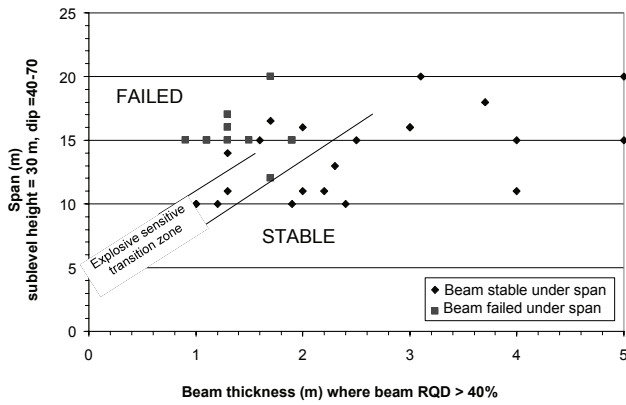
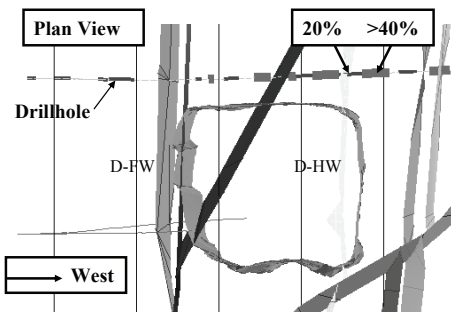
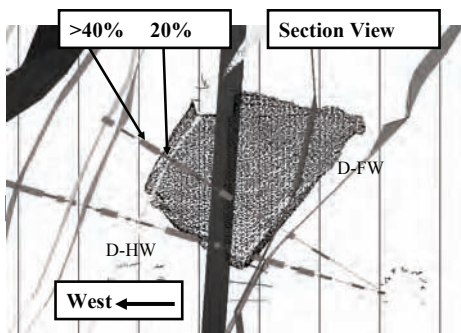


Figure 8.—Composite beam thickness versus span stability graph.



RQD Plot 716D 12C-11L (7163N Central)

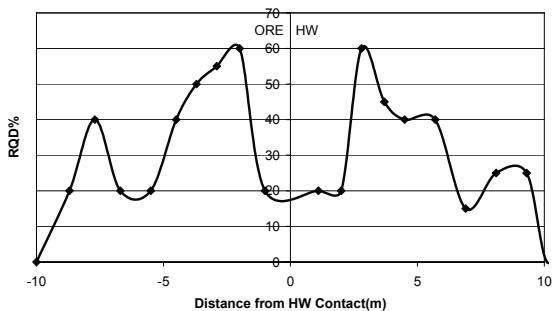
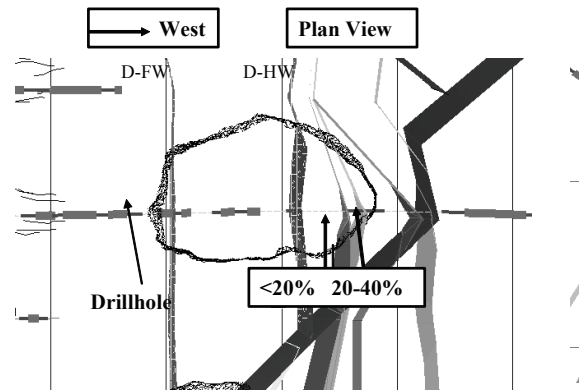
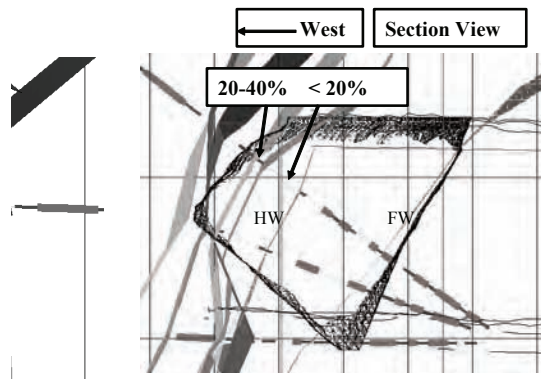


Figure 9.—716D 12C-11L. Secondary stope where a major failure was expected using predicted stress conditions. Local rock mass conditions limited depth of failure to 2-3 m as stope failed through category 3 rock to the category 1 composite beam (February 2004).



RQD Plot - 742D 10L-10C (7440N Upper)

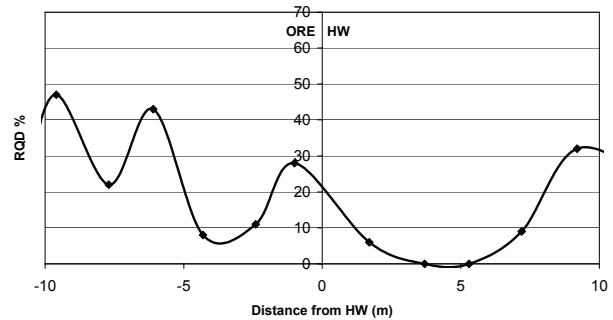
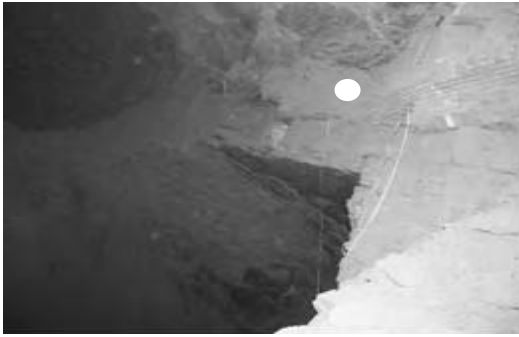


Figure 10.—742D 10L-10C. Primary stope where 6- to 8-m depth of HW failure was predicted. Failure depth was 8-10 m. Stope failed without arching through first 5 m of category 3 rock, then arched off to a stable profile through category 2 rock (September 2006).

distance from the hangingwall contact enabled an improved and accurate method of estimating HW dilution. RQD was noted as the most significant contributor as a geotechnical input into stope design and is the most available and most easily obtained data. Stope performance prediction now includes a cumulative overbreak prediction where the extraction profile is estimated through the different domains to create an expected stope shape for mine planning. Further work is being conducted to look at correlations between arching failure angle, span, and RQD. This may be difficult to quantify due to the lack of available data to estimate bedding perpendicular joint spacing for individual stopes.

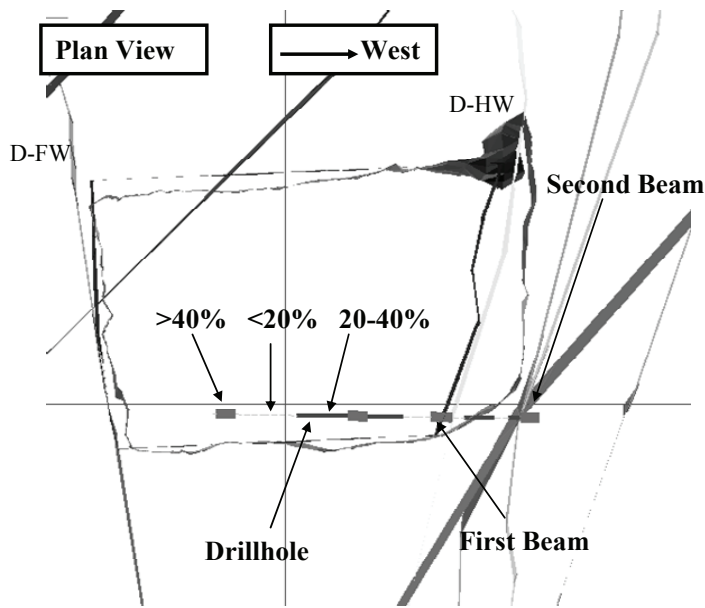


11-m HW span



17-m HW span

NOTE: Dot represents similar position under each span. Span was increased in the upper left of photo.



RQD Plot 742D 12C-11L(7440N upper)

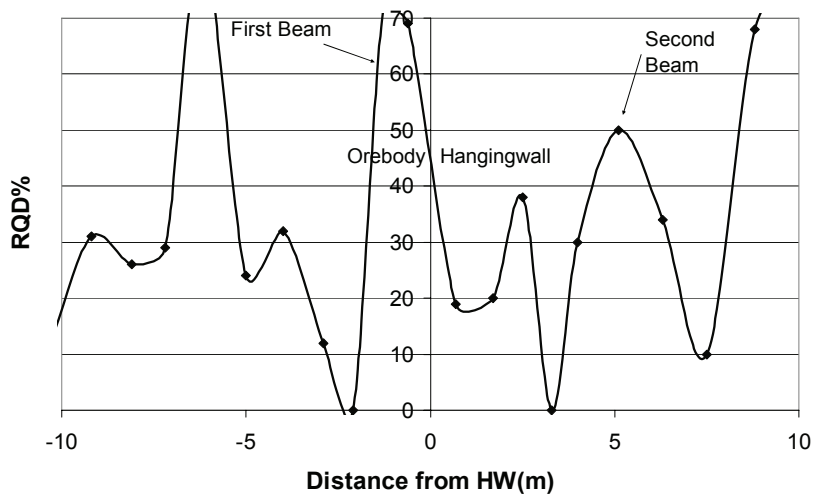


Figure 11.—742D 12C-11L. First composite beam (category 1) was stable under 11-m span, but failed under a 17-m span, arching through the category 2 rock to the second composite beam (May 2004).

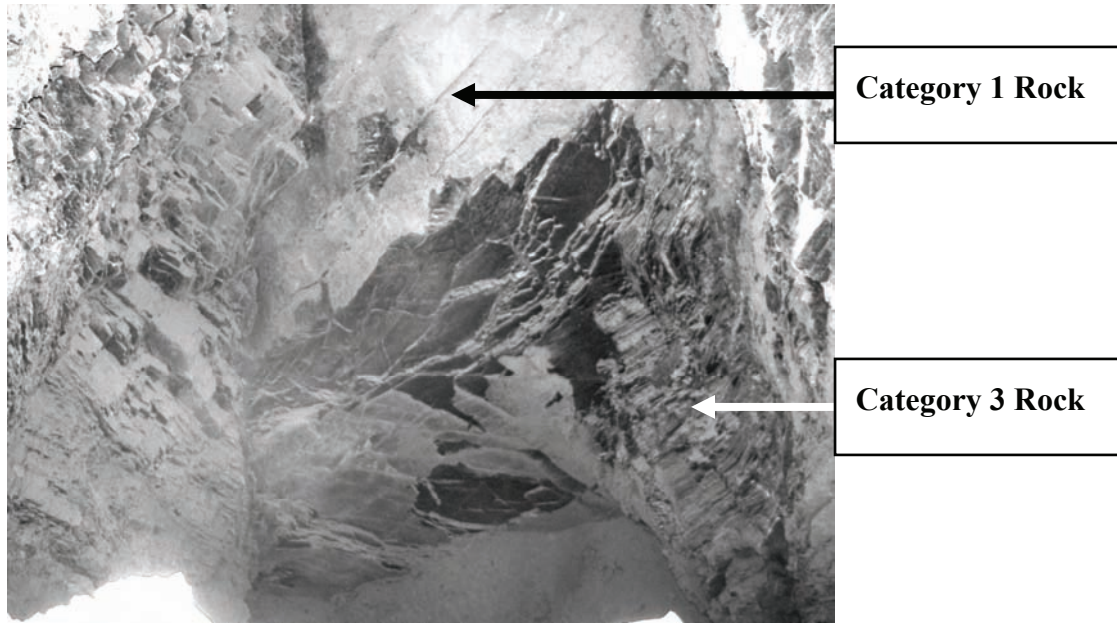


Figure 12.—Looking west through the open stop. Photo shows failure through category 3 to category 1 rocks.

Phase 4: Additional Factors

With the improved model for predicting HW performance developed, the effect of faulting, blasting, stress, undercutting, and time were then investigated to examine how much of a role they played in stope HW stability.

The effect of faults at the mine was debated, as some stopes with faults passing through had major overbreak, while others remained stable under similar stoping conditions. Some trends between faulting and lower RQD values were evident, but were not consistent. The RQD long section showed that RQD and stope stability could not consistently be related to the location of faulting, although poor rock quality can be associated with some of the faults (Figure 5). For example, there has been failure next to the S73 fault on the majority of levels where the stopes have been mined to the north of the fault (Figure 13). This correlation cannot be seen with all the north-east trending faults and agrees with the geological descriptions that the major faults show variable dextral offset, displacement and features [Grenfell and Haydon 2006].

Blasting has not been analyzed to the same degree as the other factors. This is due to the consideration of a 25-m-wide ore body, the failures are occurring at a depth much greater than the influence of ANFO higher-density explosive (approximately 0-2 m back break based on local opinion) or ISANOL lower-density explosive (approximately 0- to 0.5-m back break) in the HW ring of an 89-mm-diam production hole, and that the majority of HW rings have been charged with low-density explosive. The study has only touched on a few areas where data have been

available to show the effect of using low- or high-density explosive near a category 1 rock mass. This is shown as the explosive-sensitive zone on the composite beam versus span stability plot (Figure 8).

The performance of primary and secondary stopes was compared to examine the effect of the different stress conditions under which the stopes were mined. There were no consistently observed trends to indicate that stress levels influenced stope HW behavior on a mine-wide scale based on a comparison of depth of HW failure in primary stopes and secondary stopes in the same area of the mine. The comparison of depth of failures averaged 4.2 m for secondary stopes (58 samples) and 4.3 m for primary stopes (75 samples). However, secondary stopes have most often either performed in a stable manner or had >5-m depth failure, with significantly fewer cases in the 0- to 5-m depth of failure category (Figure 14). The sizes of primary and secondary stopes have varied through the life of the mine, providing a good spread of data for comparison. Stope HW performance can be seen as a function of individual stope design choices, where the stope depth of failure is plotted on a long section with rock quality (Figure 13). Secondary stopes extracted with adequate sequencing have performed well under larger HW spans than primary stopes where the HW RQD in the secondary stope has been better than that of the primary. On the other hand, there have been additional HW and back failures in secondary stopes where mining has been conducted out of sequence (triple-lift pendants), where cemented fill has failed in primaries, and some areas where primary stopes in the lift above were not filled for up to 8 months, resulting in additional fall-off and a subsequently worse

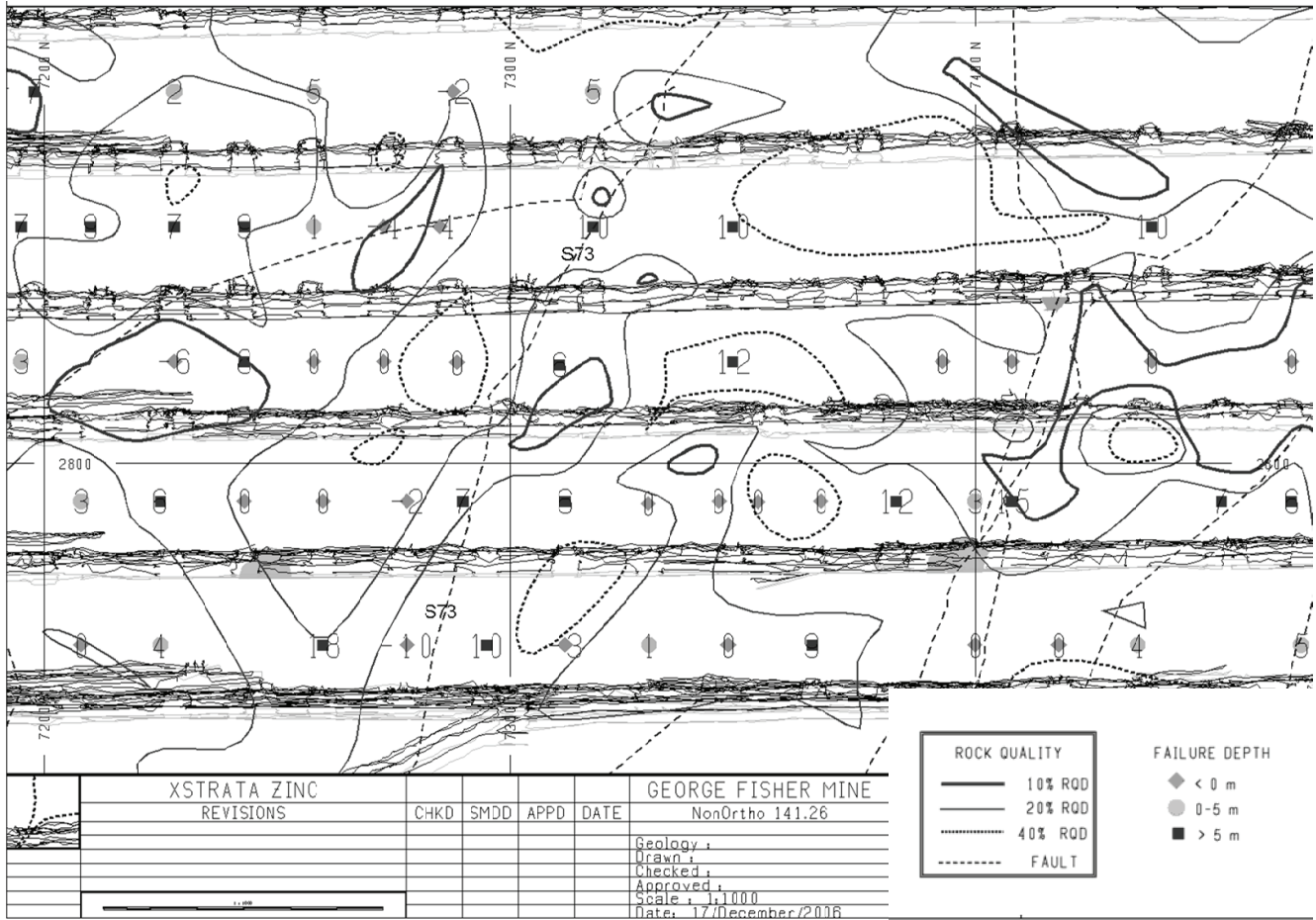


Figure 13.—Long section with faults, stopes, and depth of failure (plotted at stope centerlines).

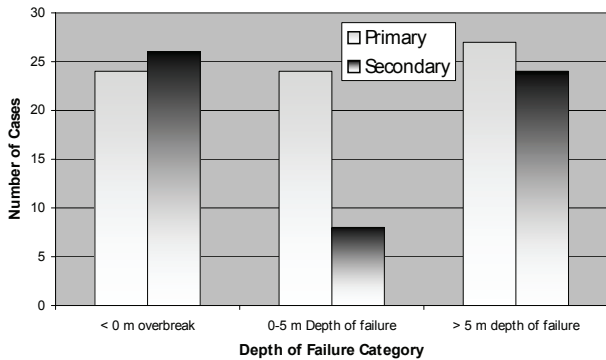


Figure 14.—Comparison of failures in primary and secondary stopes.

stress condition for mining the secondaries. During 2006, significant delays in cement filling and/or lack of tight filling of previous lifts of the primaries were experienced. Three of the four secondary stopes (719D, 723D, 730D 10L–10C) experienced large back failure/caving above the designed stope level. One particular stope had 10 m of HW

failure and caved 20 m above the designed stope level when it was predicted to have 2–3 m of HW failure based on the cumulative overbreak model. The primary stopes had all been mined at a 15-m strike span to access the tonnes sooner and were left open for up to 254 days, during which time sidewall failures occurred off of the secondary stope pillars, resulting in a stope <15 m wide. It is important to capture such information to verify the limits to which the design model can be applied and to reiterate the potential downstream mining-influenced effects where sequencing and backfill conflict with meeting production targets.

Current work is being completed to determine the effects of producing stopes adjacent to voids that have remained open for extended periods of time. The mining cycle, from firing to filling, is being analyzed with respect to stope behavior prediction and performance, with emphasis on time-related failure of the stopes. Analysis to date has shown that relaxation/failure of the secondary pillar seems to be reduced by the timely filling of the primary stopes on the same level as well as the next lift above.

Being able to recognize potential poor stope performance at an early stage in the design process will allow for more efficient mine planning and extraction, with manageable dilution. So far, telltale signs of poor-performing stopes have been mining difficulties when establishing secondary crosscuts, such as arching failure of the crosscut profiles, and difficulty installing ground support and reinforcement. HW and/or back overbreak from the previous lift must also be considered to determine stope HW design choices for the next lift. In conjunction with the model, the time-related failure study, together with observational data, should give the planning team more insight into predicting future stope performance and managing stope stability.

Summary: Results of Trials and Methodology

Management and operational support to implement changes to individual stope designs was very positive, and many ideas were discussed to improve mine performance based on the understanding of HW behavior. Based on the cumulative overbreak model where the rock mass conditions meet the specific criteria, these design choices include the verticalization of HWs, use of cabled and non-cabled ore chocks for HW stability, use of ore skins where mining consequences from HW failure are high, and an effective rock mass management strategy [Capes et al. 2005]. The development and use of the model resulted in significant research and development benefits to the company. Figure 15 shows the contribution of the stoping methodology change as a reduction in annual stope dilution from 14.4% (2003) to 6.3% (2005), an increase in zinc grade from 7.4% to 8.7%, and an increase in production from 2.1 to 2.6 Mt [Capes et al. 2006].

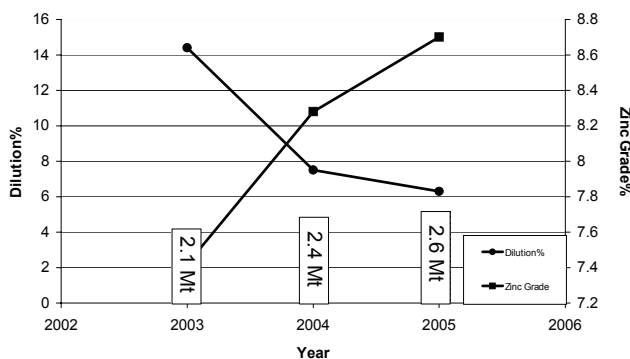


Figure 15.—George Fisher North stope dilution percent, total mine tonnes, and zinc grade versus time.

CONCLUSIONS

Dedicated on-site research using a sound geological engineering approach of data collection, communication, underground observations, analysis, and implementation of ideas resulted in a thorough understanding of HW

overbreak. This was achieved by creating a model for design based on historical performance, understanding why it worked and why it did not work, appreciating it for the insights it provided, and continually reanalyzing as more information became available. The study has reemphasized the belief in understanding relevant factors for individual stope design. The averaging approach for estimating rock mass properties for design methods does not always provide the best answer based on available information. Many considerations must be given to design individual stopes. Generalizing all stope designs into one category can lead to poor design. Individual fault characteristics play a role in stope stability. There is no discernible difference in average HW depth of failure between primary and secondary stopes, although mining out of sequence, triple-lift pendants, and poor cement fill quality or lack of tight fill and/or delayed fill in primary stopes may lead to greater HW failure and create subsequent stope back failure. The model created the ability to develop innovative empirical stope design tools to control and reduce dilution in conjunction with following a set of mine-wide extraction rules developed from numerical modeling. The development and use of the model resulted in significant research and development benefits to the company. The confidence in prediction created individual stope design changes that contributed to a reduction in dilution and an increase in head grade while continuing to ramp up production from 2003–2005.

ACKNOWLEDGMENTS

The authors would like to thank Xstrata Zinc and the University of Saskatchewan for their support, as well as the various technical services and operations personnel and consultants who have shared their thoughts during their time at George Fisher Mine Operations.

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