Short-term Models

13

Abstract

Most mineral resource estimates are not final. They are interim estimates modified by more information as it becomes available. At the time of actual mining, or just before mining, the nature and requirements of estimation is different. Results that are accurate over a longer time scale are no longer sufficient. This Chapter explains considerations for short and medium term mine plan models.

13.1 Limitations of Long-term Models for Short-term Planning

Resource models are said to be long-term when they are used for long term mine planning, such as Life of Mine (LOM) plans. When a feasibility study is prepared for a new mining project, a mining schedule needs to be prepared to estimate future cash flows from the operation. The LOM plan is based on a reserve model, in turn converted from the resource model. It provides estimates of tonnage and grade for each period involved through to the end of the life of the mine. Often, the LOM plan is scheduled according to variable units of time. For example, it may be that for the first two years of the operation, the schedule is monthly; the following two years it may be based on semi-annual volumes; and from the fifth year until the end of the mine life it may be yearly.

Long-term models are based on widely-spaced drilling, which is gradually filled in as the project advances. The long-term models are usually updated on a yearly basis with information gathered from new drill holes. More accurate forecasts in the short term are often needed as well. It is tempting to use the existing long-term resource model for shorter term predictions. However, because of the dynamics of the operation, the long-term model quickly becomes outdated.

Long-term models are by construction designed to provide global estimates with acceptable accuracy. Global estimates are understood to correspond to volumes equivalent to a year or longer. Therefore, it cannot be expected to perform as well on a block by block basis, or even for a small volume. Sometimes reasonable accuracy is obtained from long-term models for volumes smaller than a year, particularly for disseminated-type deposits, deposits with very simple geology, and grade variables that do not exhibit high spatial variability. In the case of a new operation, the long-term model will generally be based on relatively tight drill hole spacing (infill) covering the initial years of operation, designed to accurately estimate the payback period.

Updating the long-term model is required virtually in all mine operations for several reasons. The most important reason is the need to improve accuracy for Medium- and Short-term mine plans. These plans would correspond, for example, to yearly budgets and quarterly forecasts of mine production and corresponding cash flows.

For month to month mine planning, the model's reliability is increased through infill drilling. The additional drilling will result in improved accuracy of the resource model for the near-term mine operation. Updating the long-term model with the new data and subsequently updating the corresponding mine plans results in less uncertainty about the operation's short-term cash flow.

The definition of "medium" and "short" term models varies widely from one mining company to the next, and also from one geographic area to the next. In many cases, a "short-term" model is in fact a grade control model, the daily ore/waste selection process. In this book, a medium-term model will be any model that is meant to provide estimates on much smaller volumes than the long-term resource model, and is also short-lived. It generally means a volume equivalent to one to six months production, although it depends on the type of mining performed. The models developed for daily ore/waste selection and weekly mine plans are always called here short-term or grade control models.

M. E. Rossi, C. V. Deutsch, *Mineral Resource Estimation,* DOI 10.1007/978-1-4020-5717-5_13,

13.2 Medium- and Short-term Modeling

Updating the long-term resource model using in-fill data implies repeating many of the steps described in previous Chapters. This is regardless of whether the task involves estimation of values, estimation of distributions, or simulations. However, some special considerations are required, particularly if production information is used.

One of the most difficult aspects of updating short-term models is updating the geologic model and estimation domains using production data. In practice, face, bench, or stope mapping from underground workings and a description of blast hole cuttings or production drill holes may be available, but seldom used. This is partly due to data quality, and also to the tight timeframe involved.

The grade model can be updated using both infill drill hole data and production data. The use of blast holes can be controversial for several reasons, including perceived sampling quality, and discrepancies of its grade distribution compared to the exploration drill holes grade distribution. Despite the difference in quality of the individual samples (drill holes vs. blast holes), often the much larger number of blast holes available compensate for the poorer precision of the individual sample. The key to using blast hole samples is that there should be no significant bias.

A different issue is the estimation strategy. The implementation of any estimation method should consider the possibility of blast holes overwhelming the infill drill holes in certain areas; thus, an adequate estimation strategy should carefully consider how blast holes are used.

In all cases, the medium- or short-term block model should be updated only for the relevant portion of the deposit, for example, corresponding to the next three months of production. An example is given here of a medium-term model prepared for the Escondida copper mine in Northern Chile and courtesy of BHP Billiton. It illustrates a practical application of the process.

13.2.1 Example: Quarterly Reserve Model, Escondida Mine

At Minera Escondida in early 2002, medium-term planning was required on 13-week intervals, since this was the forecast period used, and updated on a monthly basis. Therefore, the quarterly planning cycle was in fact a monthly moving-window that represented the planned mined volumes three months at a time. In order to develop a practical methodology and demonstrate the usefulness of updating the long-term resource model, an initial study was developed that consisted of the following:

1. Develop a Sequential Gaussian conditional simulation model and comprising the volume corresponding to the previous year of production, FY01, (July 1, 2000–June 30, 2001) was prepared. The simulation grid was $1 \times 1 \times 15$ m, and was used as a reference to compare the alternative models and methodology developed. The simulation model not only honored the histogram and variogram models of the conditioning data, but reflected actual production figures. The simulated variables were Total Copper (TCu), Sulfuric acid-soluble Copper (SCu), Arsenic (As) and Total Iron (Fe). The conditional simulation model is not described here in detail, as it was only used as a reference.
2. The volume to be mined in the following quarter was defined, and a reserve block model is created within it. The blocks can be the same size as the long-term resource model blocks, or smaller if the additional infill and/or blast hole data justifies it. In the case of the initial study, for each month of the FY01 period, a quarterly model was defined based on actual mined out volumes.
3. The geologic model is updated monthly using information from bench and face mapping, as well as blast hole cuttings. For example, when completing the quarterly model for the month of January, the planned mining volumes corresponding to the months of February through April are considered, and the geologic information available up to December 31 is used.
4. The grade models (TCu, SCu, As, and Fe) were updated using infill drill holes and blast holes through the previous month. The same methodology as used for the long-term resource model is applied, except that smaller blocks sizes were used as warranted by the additional drill holes available. The long-term block model is $25 \times 25 \times 15$ m, while the quarterly model is based on $12.5 \times 12.5 \times 15$ m blocks; therefore, within each block of the long-term model there are 4 blocks of the quarterly model. It is always convenient to define the quarterly block model in a manner consistent with the geometry of the long-term model, such that comparisons can be easily made.
5. The quarterly models are compared with the long-term resource model and with the reference simulation model to quantify the improvements obtained. In the case of the routine, operational procedure, the comparison is done against the monthly reconciliation figures for the prior months, such that a closer control of the long- and the medium-term models is maintained.

The long-term resource model historically underestimated mine production, particularly in-situ TCu grade. The resource estimation methodology was partly to blame, but even after improving the estimation methodology, the resource model still had a small TCu deficiency. This deficiency was traced to a lower-than-expected TCu grade in the exploration drill holes, mostly those drilled using conventional rotary techniques, but also present in reverse circulation holes and in, to a lesser degree, existing diamond drill holes.

The under-representation of TCu grades in the drill holes was explained by the loss of high grade chalcocite (copper sulfide), sometimes present in non-crystalline form, and easy to wash away during the drilling process. Shorter infill drill holes were less likely to loose such material, and so were the blast holes, because of their larger diameter, large numbers, and also awareness of the problem. To improve the short-term grade and tonnage estimates, it was important to incorporate the most recent production information and local geologic mapping.

Another important requisite is that the quarterly model be obtained in a short time, hopefully in two or three days of work, and without requiring significant additional resources other than those already available. An additional requirement is the company's goal: to obtain a model with ±5% accuracy on a monthly basis for both copper grades and tonnages above economic cutoff.

The database used for the study and quarterly model updates is the same 15 m composites database used to estimate the long-term resource model. This included the more recent infill holes, and also the addition of the current blast hole database. The blast holes represent the grade of a full 15 m bench.

13.2.2 Updating the Geologic Model

Since the production geology (bench, face, and blast hole cuttings mapping) was done by a different group of geologists than those that map the exploration and infill drill holes, a prior step of consolidating and homogenizing nomenclatures and coding was required.

The lithology, alteration, and mineralization type models were updated from the existing geologic model (used to estimate the long-term resource model) only within the volume corresponding to the next three months of production. An additional area surrounding this volume was also re-modeled to allow the "tie-in" of the long-term geologic model with the more detailed Short-term model. The updating of the geologic model was done by modifying the existing interpretation from the long-term resource model on plan view. The polygons were adjusted bench by bench, from which three-dimensional solids were created. It is not necessary to apply the same level of detail as for the long-term model (see Chap. 3), since the update is an adjustment of a prior interpretation. If unexpected geologic features are encountered, then it would be necessary to review the original geologic interpretation.

Figure 13.1 shows an example of the resulting Total copper (TCu) estimation domains for Bench 2845. The larger blocks are the long-term resource model blocks, the smaller ones correspond to the same definition of estimation domains, but after updating for the quarterly model. The area shown is the complete volume planned to be mined in this bench in the period considered. Note that there is generally good agreement between the two models of estimation domains, although there are differences near contacts.

The TCu, SCu, Fe and As grades were estimated using the same methodology as used in the long-term resource model, i.e., ordinary kriging, and using the same kriging plans. The data used was all data available, including blast holes. The estimation was done on three different estimation passes, which helped control the influence of each data type. Blast holes were used only in the first pass, using the smallest search neighborhood and more data restrictions before a block could be estimated. This restricted the influence of the more abundant blast hole data.

Figure 13.2 shows for the same area in bench 2845 the estimated TCu grades for both the long-term and quarterly block models. Grades are color-coded according to the legend shown. Note that there are some differences which are significant for short-term planning, and mostly near contacts. The differences are both gains and losses. The quarterly model better delineates areas of high and low grades. For example, observe at the northern tip of the area shown (North of coordinate 108,000N) where the quarterly model predicts a NW-trending higher-grade narrow structure higher than 3% TCu, and shown in orange. This high grade corridor was not predicted by the long-term resource model.

Overall, results from the medium-term model are as expected. The use of infill drilling and blast hole increases the grade and metal content of the reserve model, and also increases its variability. The local definition of geology and grade increases also the confidence level in the estimated values. The Quarterly model is less smoothed than the long-term model.

Figure 13.3 shows the comparison of the grade-tonnage curves of the long-term (LT) and quarterly (QT) models. Note how both models have very similar tonnages above cutoff, but the QT model presents slightly higher grades for most cutoffs. The cutoffs of interest are 0.7% TCu (direct mill feed) and 0.3% TCu (marginal stockpile).

Figure 13.4 shows the grades for the two models by bench averages, for the Quarterly period beginning February 2002. Note that most benches have very similar estimated grades, although there are some where the overall average is somewhat different. This is particularly the case for Bench 2845, the grades shown also in Fig. 13.2.

Figure 13.5 shows the comparison of the relative differences of monthly TCu grade averages of the LT and QT models for the three-month period beginning in February 2002. They are compared to the conditional simulation reference model, which was calibrated to production data. Negative errors imply underestimation of TCu grades for the month.

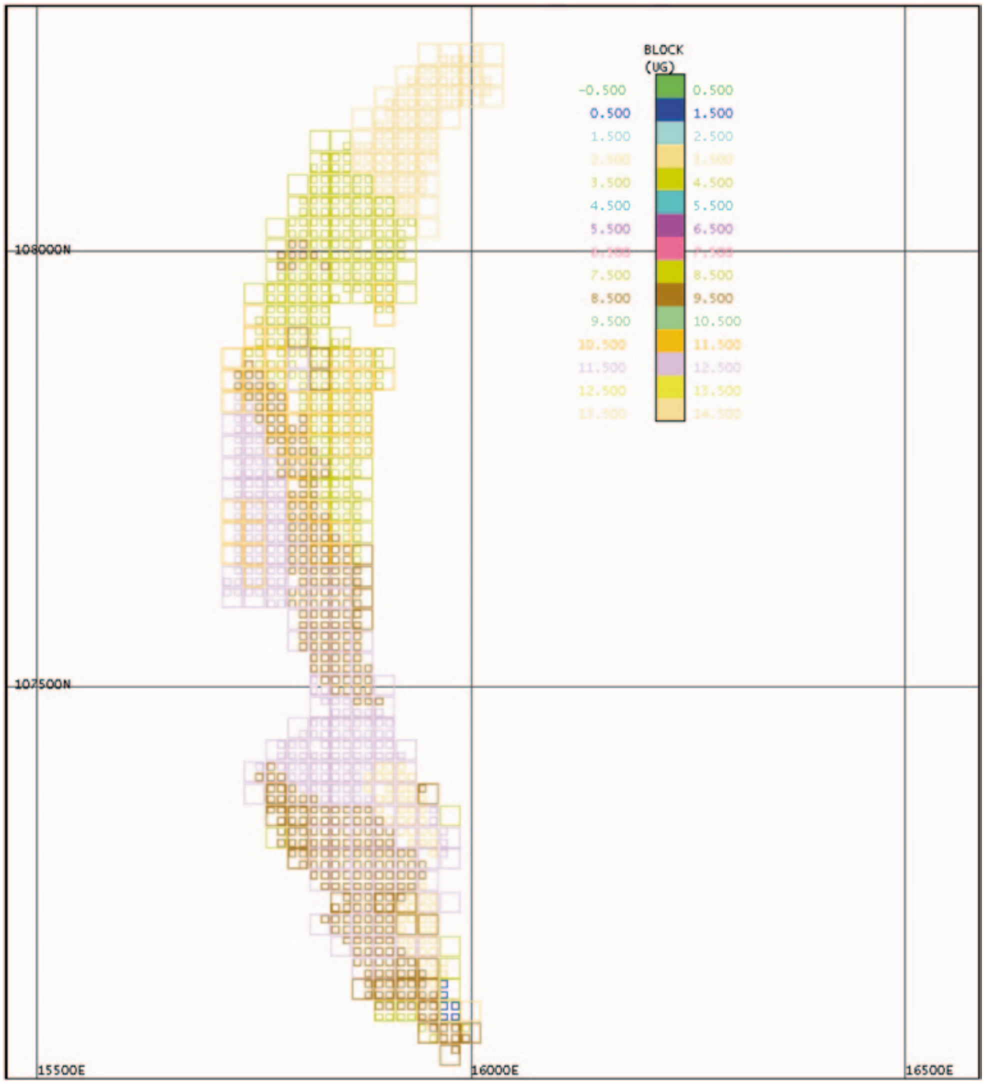


Fig. 13.1 TCu Estimation Domains for the long-term and quarterly models, Bench 2845, Escondida Mine

Note how the QT model monthly averages approximate much better the corresponding grades predicted by the reference model for most months. Although the reference model is only another model (based on a single conditional simulation), by construction represents well the production grades from previous periods. The QT model, based partly on blast holes, is also expected to be a better predictor of production grades.

13.3 Selection of Ore and Waste

The process of ore/waste selection at a mine, or grade control, whether underground or open pit, is the most geological important decision at the mine. The final, irreversible decision as to what is ore and what is waste is made. In open pit mines, the decision is generally made on a daily basis, and commonly based on sampled blast hole information. In the case of underground mines, the process may be based on infill drilling and completed at the time of defining the stopes to be mined (short-term mine planning) as usually the complete stope is classified as either ore or waste. Any mistakes that may occur at this decision point are not only irreversible, but also cannot be compensated by other types of errors, as is sometimes the case with resource estimation.

Grade control is key to the mine's profitability because the resource is finite, and the time of selection is the last opportunity that the mining company has to realize its expected revenue. It is also used to maximize resource recovery, or more frequently in the Western world to optimize recovered dollar value. Also, the processing plant usually works better when a constant grade is fed to it. Sometimes stockpiling is necessary to avoid fluctuating grades. There are four areas of interest in grade control: classification, cutoff grade, loss functions for grade control, and the consideration of non-free selection.

Classification is the process of deciding where to send the mined out material. A block is selected as ore if the revenue from processing it as ore exceeds the cost of mining it as waste. As discussed in Chap. 7, the calculation of cut-

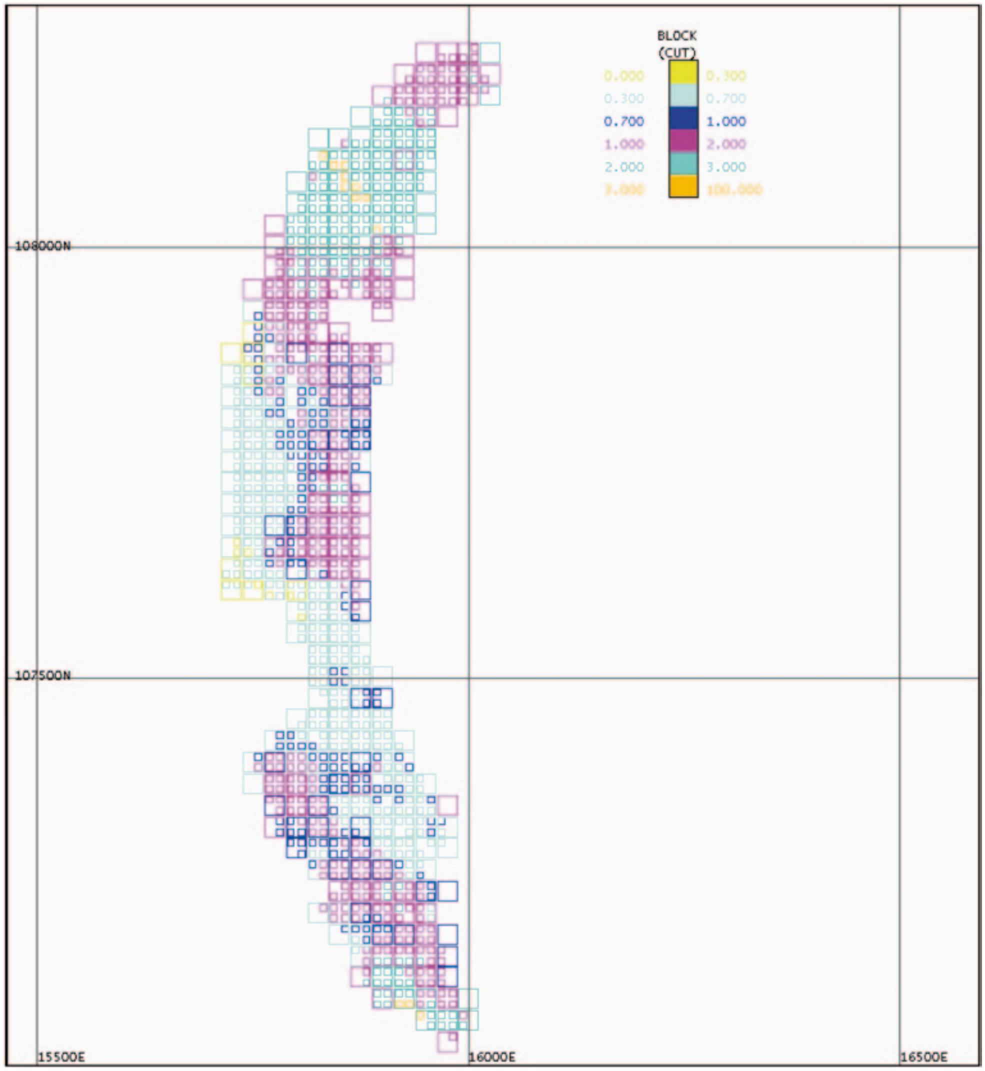


Fig. 13.2 TCu grades for the long-term and quarterly models, Bench 2845, Escondida Mine

off grades may be complex and site specific. Many different costs and variables may come into play. One possible definition of a processing (also called marginal or in-pit) cutoff grade is:

$$z_c = \frac{c_t + (c_o - c_w)}{pr} \quad \left[z_c = \frac{c_t}{pr} \right]$$

where c_t is the unit treatment (milling) cost; c_o is the unit ore mining cost; c_w is the unit waste mining cost; r is the metal recovery factor; p is the unit metal price; and z_c is the grade that makes revenue nil. In this marginal cutoff equation, costs such as General and Administration (G&A) and mining costs are not considered, only the additional costs that may exist when mining ore as opposed to waste. This cutoff grade is applicable when the operation has already committed to moving the material. The only remaining decision is whether it is sent to the waste dumps, stockpiled, or processed.

Grade control attempts to minimize miss-classification. The basic issue is shown in Figure 13.6, where a scatterplot of unknown true values for each block are plotted against the corresponding estimated grades. The most important task in grade control is to avoid as much as possible sending material to the wrong destination.

Chapter 7.4 discussed the issue from the point of views of the Information Effect, including perfect and imperfect selection. In traditional geostatistical literature the term imperfect selection is used to signify that the decision is based on estimates of grade, and without the knowledge of the true values. Perfect selection is thus impossible, because we can never know the true in-situ grades.

Another consideration is that free selection is impossible. Ore and waste blocks cannot be selected independently of each other during mining. This causes dilution and ore loss. There are also other practical (operational) factors affecting the decision, including how exactly the ore/waste markers have been laid out in the extraction area; a certain amount of unavoidable dilution (unplanned operational dilution); and mistakes made at the time of extraction, including some as simple as sending the loaded truck to the wrong destination.

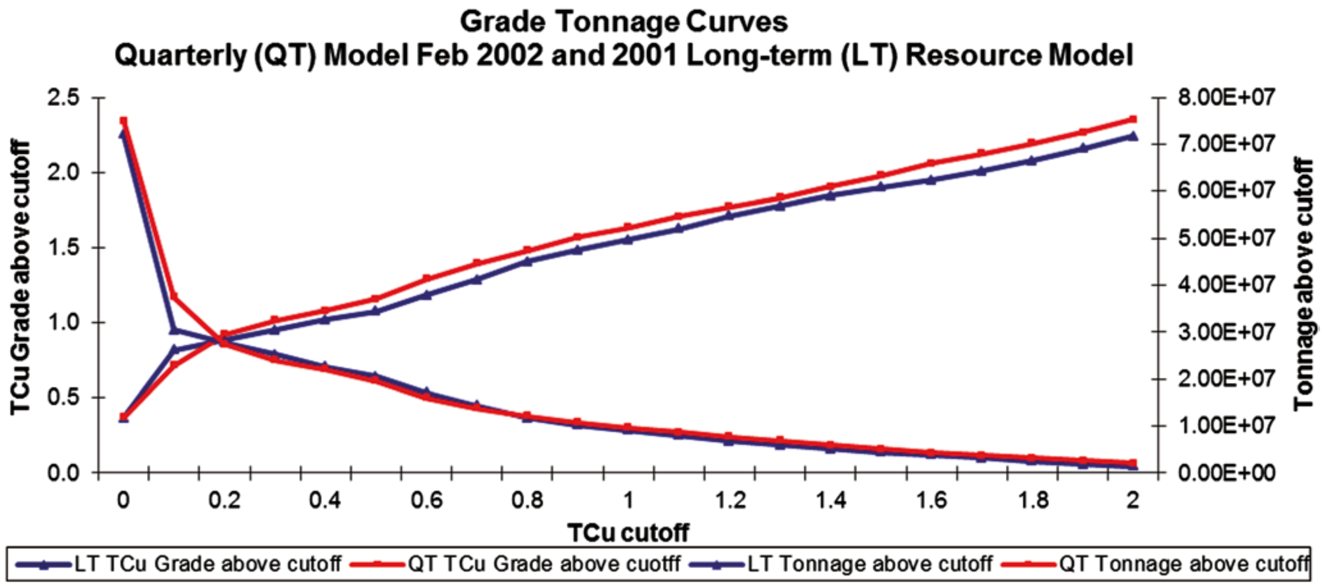
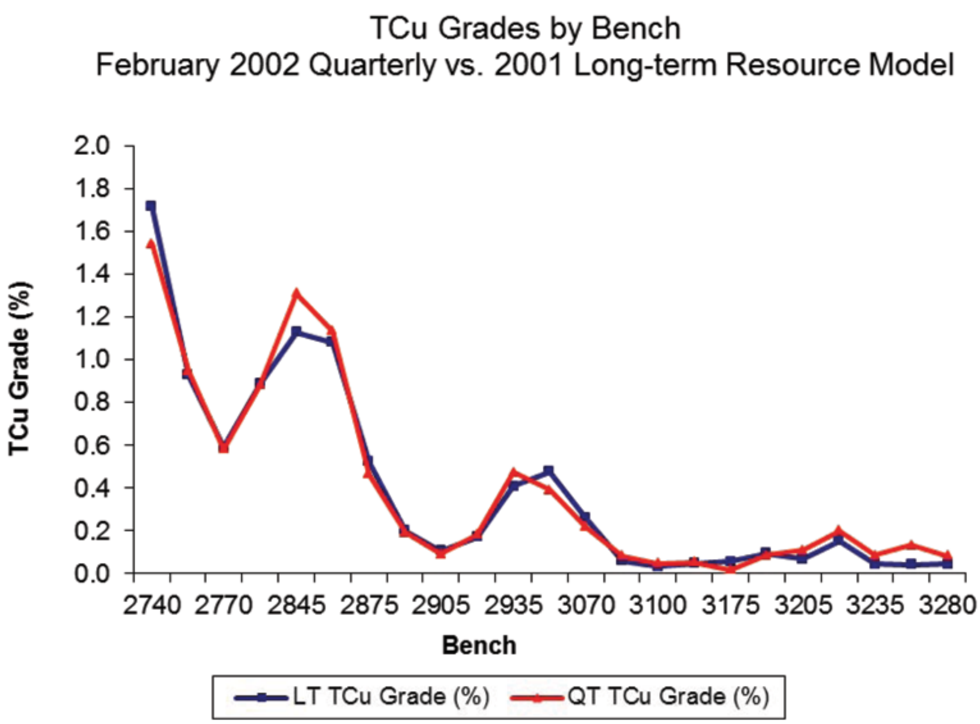


Fig. 13.3 Grade-Tonnage curves, 2001 Long-term (LT) and February 2002 Quarterly (QT) resource models. Note how the QT model has higher grade and less tonnage than the LT model for most cutoffs

Fig. 13.4 Total copper grades by Bench, 2001 Long-term (LT) and February 2002 Quarterly (QT) resource models



In general, sampling errors, estimation errors, limited information, and operational constraints result always in ore loss and waste dilution, which in turn leads to economic losses. These losses can be serious enough to make the operation unprofitable.

One example was the Hartley platinum mine in Zimbabwe, which produced its first concentrate in 1997 and closed in 1999 after what were deemed to be insoluble geologic problems and low mine productivity (Matthey 2001). Hartley is located within the Great Dyke, a geological feature running roughly north-south through the heart of Zimbabwe for about 550 km. The platinum group minerals occur in a layer known as the Main Sulphide Zone, which is typically about 3 m thick. However, the economic mining width may be as little as 1 m, depending on grade, metal prices and the chosen mining method. The reef is difficult to mine because it is not visible to the naked eye. This can lead to significant unplanned dilution and ore loss, which reduces head grades.

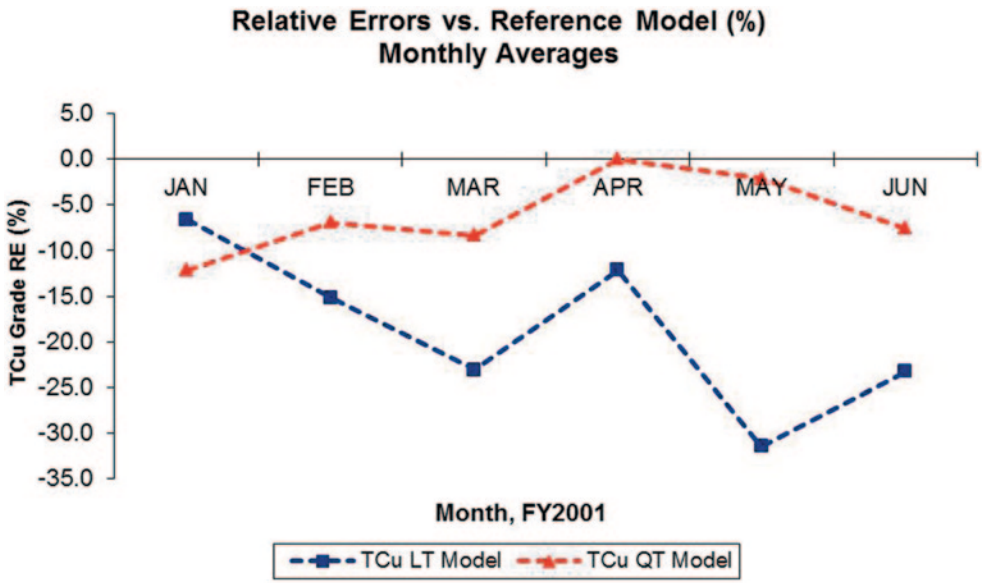


Fig. 13.5 Relative errors, Long-term (LT) and Quarterly (QT) models vs. Reference model calibrated with production data

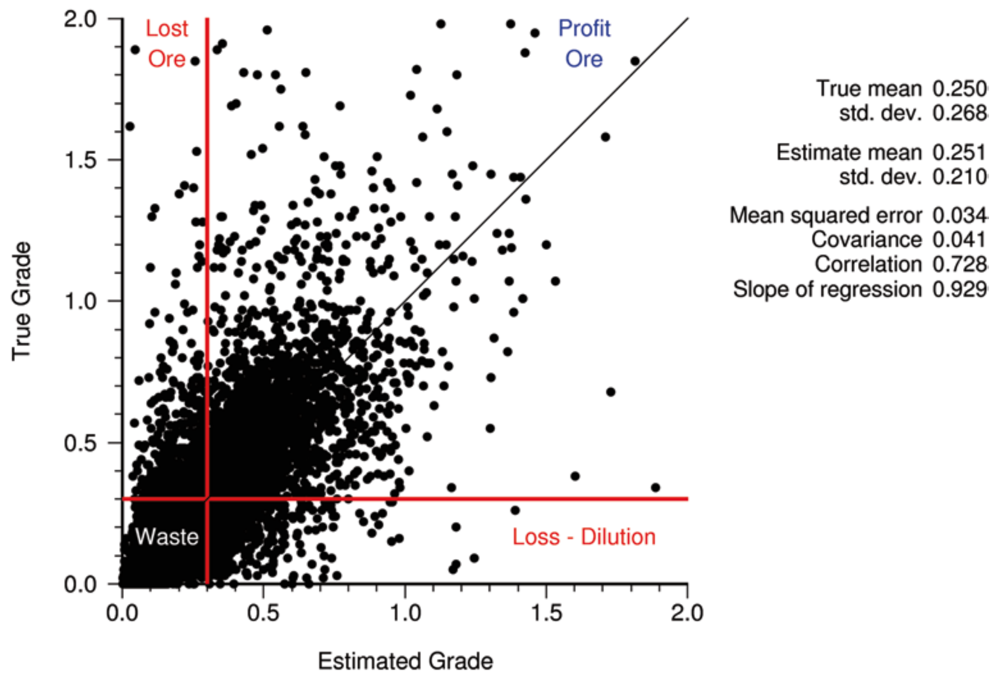


Fig. 13.6 Miss-classification in grade control

Grade control methods should attempt to minimize all possible sources of error, and not just the error prediction of the in-situ grade. Grade control should always be viewed as a complex process in which at least three basic aspects must be considered: data collection and quality; grade control model to determine ore and waste boundaries; and operational procedures and constraints, including mining methods, mining practices, and operational culture.

Firstly, data collection and data quality are always important, but it becomes even more critical when operational constraints limit the time and availability of sampling crews. Thus, the quality of the samples used to make the decision is impacted. Secondly, the samples are modeled to provide a prediction of grades, block dollar values, and other important attributes. The actual selection of ore and waste is based in those estimates. And third, all related operational procedures should be considered and controlled. The grade control method should consider the type and limitations of available sample data, the geotechnical and blasting conditions, and also the operational constraints that may render certain grade control practices non feasible.

Data collection and quality is highly dependent on the mining method, and to some extent the geometry of the orebody being mined. In open pit mines, blast holes are the most common source of data for grade control. Ocassionally, reverse circulation (RC) grade control drilling is done.

The additional cost of the dedicated RC drilling should be paid for by the increased economic benefit of the improved grade control, since almost always blast holes still need to be drilled for blasting. Grade control using RC drilling is a fairly common practice in gold mines in Western Australia and parts of Africa. It is generally applicable if the ore is of high intrinsic value (such as high grade Au) and if the higher-grade distribution is sub-vertical. Unfortunately, not all operations perform a detailed cost-benefit analysis of the use of RC drilling for grade control. The costs of using RC drilling may be higher than the economic benefits derived from the improved grade control.

In the case of underground mines, mining methods are much less flexible and therefore there is generally little or no opportunity for ore and waste selection at the time of extraction. When a stope is defined as being ore, typically the complete stope is considered ore (with the planned and unplanned dilution as encountered). This implies that the grade control data is actually the data used to design the stopes during short-term planning. In such case, infill drilling is used to decide what is ore and waste. The challenge for underground mines is thus greater, because generally infill (or production) data spacing is less than the equivalent blast hole grids in open pit mines.

The modeling of grade control or infill data can be accomplished using conventional or geostatistical methods. Among the latter, conditional simulations is usually the better option, since ore/waste selection is dependent more on the variability of the grade distribution than on its average grade. Kriging-based methods can very easily fail (as can the more conventional methods) because of its characteristic smoothing effect which can lead to miss-classification. Additionally, using minimum-variance estimation methods imply penalizing the over- and underestimation errors equally, i.e., a symmetric Loss Function (Journel 1988; Srivastava 1987). This is generally inappropriate for mining scenarios, since sending waste to the plant generally has a different cost compared to sending ore to the waste dump.

Grade control models are dependent on mining practices and methods. It is possible that more detailed and sophisticated grade control methods can provide a better ore/waste selection, but the mining method has to able to capitalize on that opportunity. It may be an overkill to develop and implement a sophisticated grade control method if the mining method and operational practices are not good enough to take advantage of the additional level of detail.

13.3.1 Conventional Grade Control Methods

Conventional methods used for grade control include blast hole averaging, inverse distance methods, and nearest-neighbor-based methods. For the mathematical description of the methods the reader is referred to Chap. 8. Here the more common industry practices are discussed.

Unfortunately, even after major technological advances in many aspects of grade control including geostatistical modeling, most operations still do not fully appreciate the importance of grade control, and devote insufficient resources and thought to this task. The flexibility that open pit mines generally enjoy is not always fully utilized. Many operations work with very simple methods that are not optimal. This is also true for underground mines. Indeed, it is more difficult to perform effective grade control in underground mines because of operational constraints, but still, too few operations have profited from modeling advances over the last 20 or 30 years.

In open pit mines, probably the most commonly used method to predict in-situ grades is a simple arithmetic average of the available blast holes. A block model is defined, generally with the block size similar to the blast hole spacing, and the predicted block grade is the arithmetic average of the blast holes that fall within the block. Multiple variants exist, as for example the "four-corner" average method, popular in some gold mines in Northern Nevada (Douglas et al.1994), whereby the average of the four blast holes at the corners is the block grade estimate.

Other commonly used methods include the nearest-neighbor method and inverse-distance methods, implemented in a number of variants. In all cases, the main characteristics of the methods are that (a) a simple estimator is used to assign grades to blocks, and (b) the blocks are relatively large with respect to the average distance between sample points. The second characteristic is unjustifiably common, and a major source of inaccuracies, since the data density is generally sufficient to justify much smaller blocks. Smaller blocks would lead to better definitions of ore and waste boundaries.

13.3.2 Kriging-based Methods

Kriging-based grade control became popular in open pit mines during the 1980s. Different types of kriging algorithms were used, but most commonly ordinary and indicator kriging were applied, for example in gold mines in Northern Nevada.

In the case of ordinary kriging, the application of the method is similar to those described as conventional methods above. Ordinary kriging is used to provide an estimate of grades, based on which the selection panels are drawn. The advantages of kriging over other estimation methods were discussed in Chap. 8 and include the minimization of the estimation variance. In practice, kriging has been only marginally more successful at grade control compared to conventional methods because of the inherent smoothing and the use of inadequate kriging plans. Also, the minimization of the estimation variance is not optimal for grade control (Srivastava 1987).

Multiple variants of the indicator kriging approach have been used. A common application considers a single indicator estimated at the ore/waste boundary of interest, thus providing the probability of any block or point within the blast being ore or waste. Generally point kriging is performed, usually at a larger-than necessary grid spacing. Occasionally, block kriging may be done, ignoring the fact that the average of estimated probabilities within a block is not the same as the point probability derived from the ore/waste indicator (Chap. 9). Nonetheless, the practice is to analyze equal-probability contour lines for several values and decide based on visual observations which one adjusts better to prior production. Commonly, in gold operations that use this method, probabilities of being ore of about 30–40 % are used to define ore/waste boundaries.

A method that has proven successful in several operations is the "Breakeven Indicator Method" (BEI), as described in Douglas et al. (1994). It was implemented first at Independence Mining Company's Jerritt Canyon, north of Elko, Nevada, in the early 1990s.

The BEI grade control method uses a combination of both indicator and grade kriging. An ore/waste indicator variable is used to predict the probability of ore occurrence at a given location $P_o(x)$, which is obtained by kriging the ore/waste indicator variable. The ore-grade blast holes are then used to krige an ore grade $Z_o(x)$ for the location x. Similarly, the waste-grade blast holes are used to krige a waste grade, $Z_w(x)$, for the same location. Then, the expected revenue is estimated from the kriged probability $P_o(x)$ and ore and waste grades:

$$E(R) = P_o \cdot R(Z_o) + (1 - P_o) \cdot R(Z_w) \tag{13.1}$$

The revenue function is traditionally calculated as

$$R = (gold\ price) * (metallurgical\ recovery) \\ * (grade) - (costs)$$

where "costs" generally imply metallurgical processing costs only. The method offers the flexibility of adding additional costs if desired, to work on what would amount to a higher ore/waste cutoff grade.

If the expected revenue from Eq. 13.1 is negative, the material at the location is waste. If the expected revenue is positive, the material at the location is ore. If the grade of ore is high, the corresponding revenue will be high, allowing for a block with a low probability of being ore to be sent to the mill. In this case, the ore pays for large amounts of waste, which ensures all high grade ore is recovered. Alternatively, if the ore grade is low, the revenue will tend to zero and the estimated probability of ore will have to be close to 1: the lower grade ore will not pay for much overbreak. Thus, the method requires that the low grade be most surely higher than the economic cutoff. This can be seen by calculating the probability that corresponds to the economic breakeven cutoff, $E(R) = 0$:

$$P_o(BE) = \frac{-R(Z_w)}{R(Z_o) - R(Z_w)} \tag{13.2}$$

The method should be applied on small blocks, one third to one half of the blast hole spacing, allowing the grade control engineer to define dig lines based on revenues. The BEI method is designed to improve grade control performance most along contacts of ore/waste zones. If the panels to be mined are very large (wide), the ratio of contact surface area per ton of ore is small. The opposite is true for panels that are narrow for which this method would provide the most improvements.

If compared to the single indicator kriging method outlined before, the BEI is equivalent to working on a variable probability of being ore, which is dependent on the revenue function defined.

13.3.3 Example: Grade Control Study

A comparison of several grade control methods was performed for the copper-molybdenum Ujina open pit mine in Northern Chile. It is summarized here, courtesy of Compañía Minera Doña Inés de Collahuasi (CMDIC). The company mines a Cu-Mo porphyry deposit with a significant Cu enrichment blanket, which was the main target of mining at the time. As a massive, disseminated-type deposit, it could have been assumed that grade control is a simple process; however, there are factors that made grade control at Ujina a complex process.

The differences observed among the methods tested will be larger if the grade distributions being modeled are more variable. Also, if there are many different possible destinations for ore and waste, the grade control process is more complicated: the grade ranges that are used to separate the material become narrower. Table 13.1 shows the possible destinations for ore coming out of the Ujina pit at the end of 1999.

A quick inspection of Table 13.1 suggests that a large degree of accuracy and precision is required of the grade control method, since the mining method and metallurgical processing requirements are very specific.

The methods tested included the inverse distance cubed (ID^3) as used at the time by the mine; ordinary kriging (OK); the breakeven indicator method described above (BEI); and the maximum revenue method, based on conditional simulations and loss functions as described further below. Only a short summary of a long and detailed study is presented here

Table 13.1 Material type classifications as of December 1999, Ujina

Material Type	*Dispatch Code*	*Destination*	*Description*
High-grade sulfide	SAL	Stock 1	TCu >= 2.0%
Medium-grade sulfide	SME	Stock 2	1.0% =< TCu < 2.0%
Low-grade sulfide	SBA	Stock 5	0.8% =< TCu < 1.0%
Marginal-grade sulfide	SMR	Stock 4	0.4% =< TCu < 0.8%
Sub-marginal grade sulfide	SSM	Stock 6	0.2% =< TCu < 0.4%
High As sulfide	SAS	Stock 3	As > 100 ppm y TCu >= 1.0%
High-grade Oxides	OXA	Stock 10	TCu >= 1.0%
Medium-grade Oxides	OXM	Stock 12	0.6% =< TCu < 1.0%
Low-grade Oxides	OXB	Stock 11	0.3% =< TCu < 0.6%
Low-Oxi	OXL	Stock 30	TCu >= 0.2%, with clays and Fe oxides
Mixed	MIX	Stock 13	Mixed, TCu > 0.7%
Waste Rock Types	IGS, IGC, RIO, SUE, PLR, OTR	Waste dumps	Waste, TCu < 0.2%

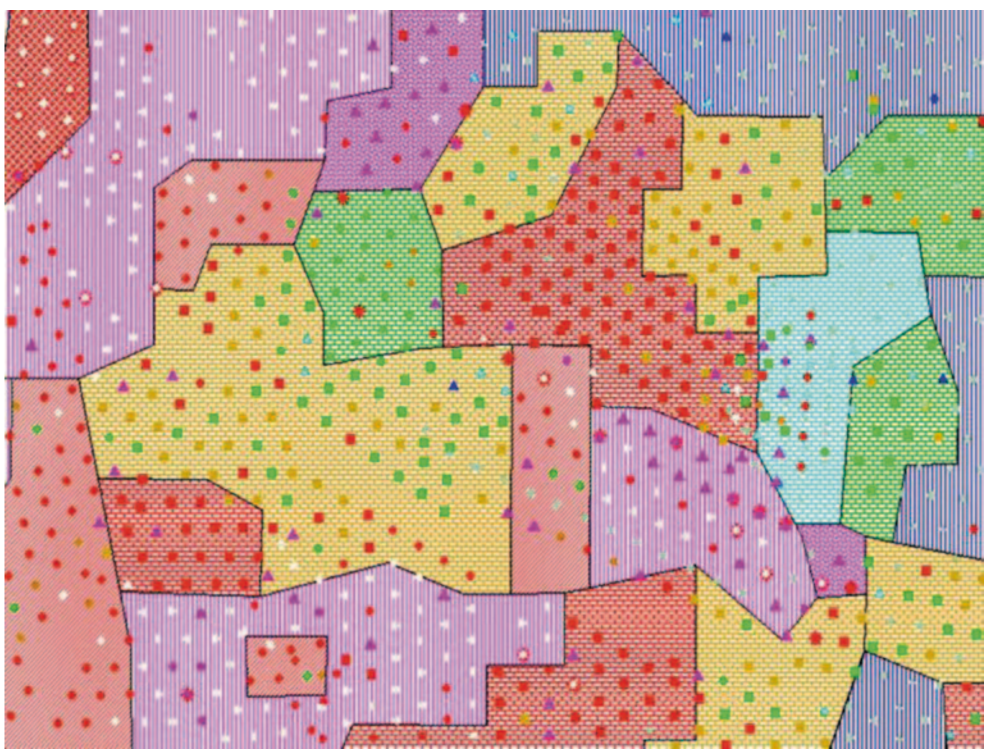


Fig. 13.7 Blast holes, color- and shape coded by destination, and grade control panels based on ID^3 interpolation. Blast hole spacing is approximately 8 × 8 m, and the area is 250 m per side. Blast holes and panel hatching represents Stocks 1 through 6 in Table 13.1

to illustrate the performance of different grade control methods, even in deposits with relatively low variability.

Figure 13.7 shows a small area of Bench 4270 with the Total Copper (TCu) blast hole grades and selection panels as defined by ID^3, which was the method used by the operation. Figure 13.8 shows the same area with panels as defined by the BEI method. And finally, Fig. 13.9 shows the comparison of the panels defined based on these two methods. In this area only sulfide material was present, corresponding to destinations (Stocks) 1 through 6 in Table 13.1. These figures demonstrate that, locally, the differences among the different grade control methods can be significant.

The comparison among the four methods tested was made against a reference model corresponding approximately to

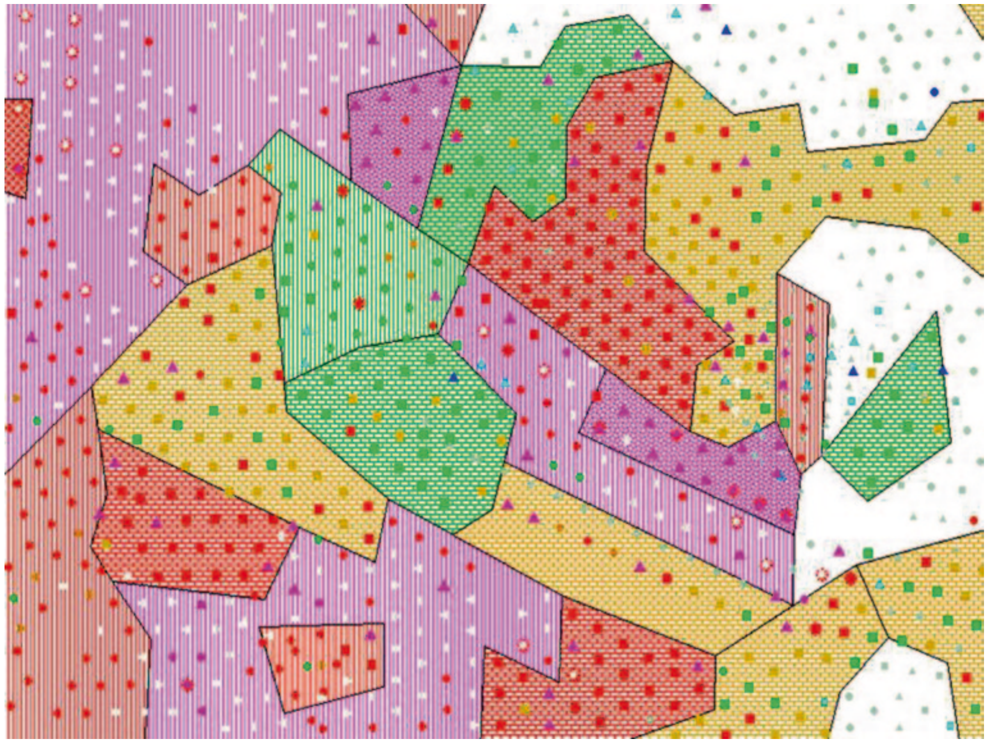


Fig. 13.8 Blast holes, color- and shape coded by destination, and grade control panels based on the BEI method. Same area as Fig. 13.7

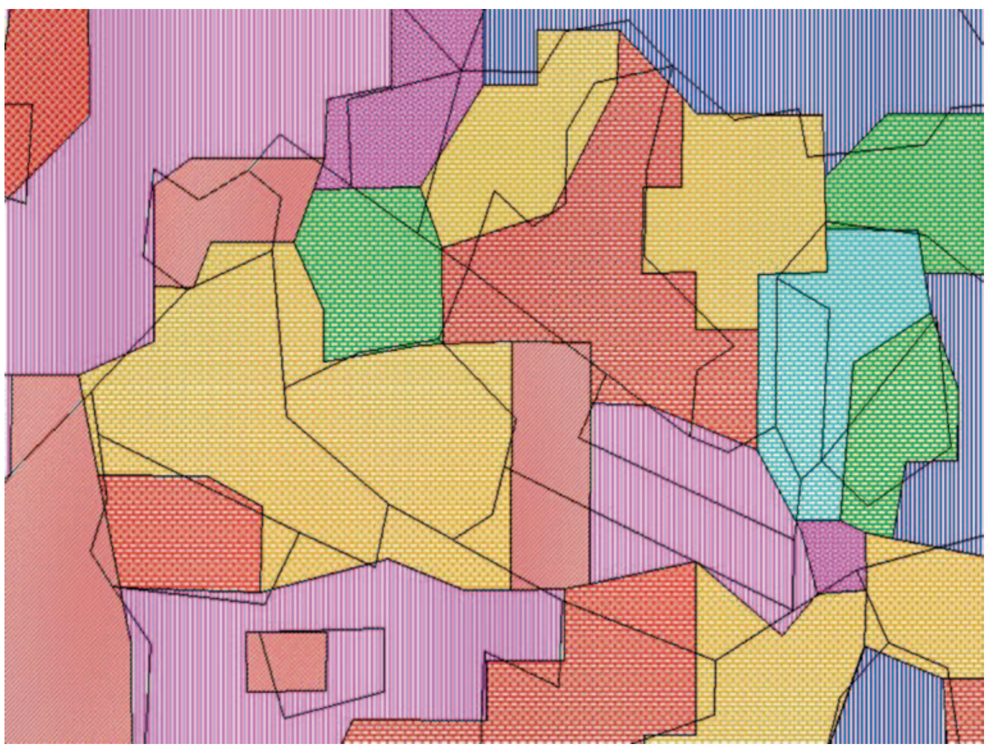


Fig. 13.9 Comparison of grade control panels according to the ID^3 used by the mine and the BEI methods, same area as Figs. 13.7 and 13.8. Note the sometimes very different selection panels

two years production from the open pit. The reference model is a single realization of a Sequential Gaussian simulation for all variables involved, and adjusted to production data. The same areas were re-modeled based on the available blast hole database, and selection panels for each destination re-drawn according to the results of each method tested. The study involved development of an appropriate revenue function, consideration of mining practices and constraints, and compared alternative methods to the actual grade control panels developed by the mine using ID^3.

Table 13.2 TCu performance factors of the ID^3 method by destination relative to the SGS reference model

Destination	Tonnage (Dest./Reference)	TCu Grade (Dest./Reference)	Cu Metal Content Cu (Dest./Reference)
SAL	1.10	0.91	0.99
SME	1.16	1.06	1.22
SBA	0.18	1.15	0.21
SMR	0.50	1.36	0.68
SAS	0.55	1.02	0.56
OXA	1.29	0.85	1.10
OXB	1.16	1.08	1.25
OXL	0.44	1.54	0.68
MIX	0.52	0.90	0.47
TOTAL	1.16	0.84	0.98

Table 13.3 TCu performance factors of the BEI method by destination relative to the SGS reference model

Destination	Tonnage (Dest./Reference)	TCu Grade (Dest./Reference)	Cu Metal Content Cu (Dest./Reference)
SAL	1.10	0.92	1.00
SME	1.09	1.00	1.09
SBA	0.45	1.01	0.45
SMR	0.43	1.01	0.44
SAS	0.87	0.95	0.82
OXA	1.13	0.93	1.05
OXB	1.98	0.98	1.94
OXL	1.49	1.41	2.10
MIX	0.71	0.78	0.55
TOTAL	1.11	0.89	0.99

Only the results for the ID^3 and BEI methods are presented here. The simulation-based method produced similar and slightly better results compared to the BEI method, but it is more complicated and slower to implement. The OK method produced marginally worse results.

Tables 13.2 and 13.3 show the relative performance of the ID^3 and BEI methods with respect to the reference model for tonnages, TCu grade, Cu metal content, and revenues. The closer the value to 1.0, the better the method reproduces the to reference model, and, by extension and within the approximations of the reference model calibrations, actual production. A factor greater than 1 implies overestimation with respect the reference model. The destinations corresponding to waste, SSM, and OXM are not shown due to the low tonnages produced within the evaluation period. The overall ore and marginal ore production for the period was about 59.5 million tons, so the statistical mass available for comparison is significant.

Note how for most destinations and variables considered, the BEI method is superior. Recall that a 1 % difference between the two methods represents close to 600,000 metric tons of ore, or about 10,000 metric tons of contained Cu. Considering the depressed Cu prices at the time, a 1 % difference in contained Cu represented about US$ 16 million. At 2013 copper prices, the dollar value of the difference would be between US$ 70 and 80 million. In most cases, even though the differences in percentage points may be small, they represent significant economic improvements given the size of the operation.

The added economic benefit of the BEI method results from virtually no additional expenditure, since all operational practices remain the same. Also, the panel drawing process is facilitated by the use of smaller blocks and less sharp corners (Figs. 13.6 and 13.7). This in turn results in less unplanned operational dilution, because the shovels will extract the material following more faithfully the delineated zones. Although real, this effect is more difficult to quantify.

13.4 Selection of Ore and Waste: Simulation-based Methods

The objective of the simulation-based methodologies is to optimally select ore from waste according to different optimality criteria. Also, it provides more flexibility to handle several destinations for recoverable material, including ore blending with different metallurgical responses. Minimum-variance algorithms such as kriging have traditionally been the optimization criteria in most geostatistical applications, but are not always appropriate (Srivastava 1987).

In open pit and underground grade control, optimization should always be based on maximizing the economic value of the recovered material. The material selected for metallurgical processing should provide the maximum possible economic benefit given all operational constraints. Other possible optimization criteria, such as maximizing resource utilization, is not applicable in the case of grade control, since the decision is short-term in nature, and aims at making the most out of the current operation on a daily basis.

Loss Functions can be used to optimize based on pre-determined functions that assign value to estimates, or equivalently, costs to mistakes. They were described in Chap. 12, and further reading can be found in Journel (1988), Isaaks (1990), and Goovaerts (1997). Conditional simulation is used to provide a model of uncertainty that can be used to optimize grade control. One alternative is the Minimum Loss/Maximum Profit method as presented below, which has been implemented with success in several open pit operations. The expected profit calculation is

$$P_{ore} = \sum_{\text{all realizations}} \left[- \underset{\text{ore mining cost}}{c_o} - \underset{\text{milling cost}}{c_t} + \underset{\text{revenue}}{prz^{(l)}(\mathbf{u})} \right]$$

$$P_{waste} = \sum_{\text{all realizations}} \left[- \underset{\text{waste mining cost}}{c_w} - \underset{\text{lost opportunity}}{c_{lo}} \right]$$

establish c_{lo} by calculating revenue if it were milled

$rev = prz^{(l)}(\mathbf{u})$

$$c_{lo} = \begin{cases} 0 & \text{if } rev < 0 \\ rev & \text{if } rev > 0 \end{cases}$$

13.4.1 Maximum Revenue Grade Control Method

The Maximum Revenue grade control method is a two-step procedure, first outlined by Isaaks (1990), and applied with success at some mine operations, for example Aguilar and Rossi (1996). Initially, a set of conditional simulations is obtained from the blast hole data available. These conditional simulations provide an uncertainty model for grades at any specific point within the blast. Second, an economic optimization process is implemented using loss functions to obtain the optimal ore/waste selection. The Loss Function quantifies the economic consequences of each possible decision.

The simulations are used to build models that reproduce the histogram and spatial continuity of the conditioning data. By honoring the histogram, the model correctly represents the proportion of high and low values, the mean, the variance, and other statistical characteristics of the data. By honoring the variogram, it correctly portrays the spatial complexity of the orebody, and the two-point connectivity of low and high grade zones. These are critical variables for the optimization of ore/waste selection because it depends on accurately predicting the variability of high to medium to waste grade transitions.

Typical grade control simulation grids can be 1 m by 1 m by bench height (corresponding to the sampled blast hole column). These are used directly in obtaining the uncertainty model for ore/waste selection panels. Larger grid sizes may be used and sometimes required because of time or general computer hardware limitations, still providing reasonable estimates when enough simulated points are included within the selection panels.

Given that conditional simulation models are sensitive to departures from its stationarity assumption, it is critical that they be controlled by geologic models. The use of geologic boundaries may introduce issues of ergodicity, which should be carefully handled. A constantly updated geologic model, in addition to constant geologic control at the pit is required to ensure that the uncertainty models derived from the conditional simulations are realistic and also representative of local geology.

Other important aspects include the behavior of the high-grade population, which is required to control the simulated high grades, see Parker (1991) and Rossi and Parker (1993). Issues such as limiting the maximum simulated grade should be carefully considered, since it may significantly impact the selection panels. The issue should be resolved through calibration with existing production data.

A small number of realizations, perhaps 20 or 30, are typically used. This reflects practical limitations, since grade control is a process that has to be completed in a short period of time; but it may also be a sufficient number of simulations to adequately describe the model of uncertainty, given the data density available.

Recall that the model of uncertainty provides the probability of that node in the grid of being above (or below) any grade z:

$$F(z;x\,|\,(n)) = \Pr ob\{Z(x) \le z\,|\,(n), \alpha = 1,\ldots,n\} \qquad (13.3)$$

where $F(z;\underline{x}|(n))$ is the cumulative frequency distribution curve for each point $\underline{x}$ of the simulated grid and obtained using the $(n), \forall = 1, \ldots, n$ conditioning blast holes.

In grade control, the selection decision (which material is ore and which is waste) has to be based on grade estimates, $z^*(x)$, while still attempting to minimize miss-classification. Since the true grade value at each location is not known, an error can and will likely occur. The loss function attaches an economical value (impact or loss) to each possible error, as described in Chap. 12.

The minimum expected loss can be found by calculating the conditional expected loss for all possible values for the grade estimates, and retaining the estimate that minimizes the expected loss. In grade control, the expected conditional loss is a step function whose value depends on the operating costs (Isaaks 1990). This implies that the expected conditional loss depends only on the *classification* of the estimate $z^*(x)$, not on the estimated value itself. For example, the loss incurred when a block of leach ore is sent to the mill is a function of the difference in processing costs related to both leach and mill; it will, of course, also depend on the *true block grade,* but not on the *estimated block grade* value itself.

13.4.2 Multivariate Cases

Grade control in the presence of multiple variables introduces additional challenges that can be easily handled. The Ujina open pit example briefly discussed above is in fact a multivariate grade control issue. There are multiple variables that add to the value of each parcel of material (copper and molybdenum), and also multiple variables that detract from its worth, such as Arsenic or the presence of clays. The multiple variables can all be mine products, or a combination of mine products, metallurgical performance variables, and contaminants in general.

In cases where there are spatial relationships between the variables of interest, then either co-estimation or co-simulation (Chaps. 8–10) can be performed. This is most important when simulating for grade control, since modeling relationships among different variables is consequential. In Chap. 14 two multivariate simulation case studies are presented.

13.5 Practical and Operational Aspects of Grade Control

There are many operational aspects that need to be considered for an effective grade control. The most important are (a) the relationships between the grade control activity and mine planning; (b) the practicality of obtaining representative samples; (c) time constraints, always present in any operation; the daily production target is the operation's main driver which does not allow for detailed modeling and planning work; (d) the gathering and use of geologic data; (e) the appropriate staking of the ore/waste zones; (f) the control of the mining process; (g) the destination of each truck or load of material; and (h) the accounting of material movement and overall reconciliations.

Each one of the aspects mentioned deserves detailed discussions and are outside the scope of this book. However, they are highlighted here to remind the reader that adequate grade control involves multiple areas of an operation, and cannot be developed in isolation from other aspects of the mine. Issues related to material accounting, particularly volumes or tonnages extracted and mine-to-mill reconciliations are among the most important. As argued in Chap. 11, they can also be the basis for model performance evaluations.

Operational details, sometimes seemingly trivial, can have a significant impact on the bottom line. Without pretending to be exhaustive, some illustrative examples mostly applicable to open pit mines are:

- Sufficient laboratory capacity to provide the assays' results in the required amount of time, usually 24 h or less for 200 to 300 samples or more;
- Traffic and destination control in the pit, particularly if truck dispatch systems are not available; in areas where manual labor is relative cheap, it is common practice to place an individual at the pit exit to verify that trucks go to the correct destination;
- Truck weighing, as a control to truck factors and volumetric measurements;
- If visual indicators of ore are available (such as green or blue oxide Cu minerals), mine geologists should visit daily the waste dumps, to ensure that the operation is not misplacing the ore loads; also, a 24-h operation should have adequate artificial lighting in the pit, more so when visual aids are used in grade control.
- The amount of broken ore in the pit should be sufficient to feed the mill for a few days; an operation where loading is always pressuring for more blasting goes counter to good grade control practices.
- Confirm the in-situ bulk density of material loaded; the operation should monitor in situ density variations, sometimes taking bulk samples from the pit. Also, consider the estimate of humidity in the rock, which is generally a simple global estimate. These estimated values affect the conversion of volumes into tonnages, with a direct impact in the accounting of metal moved.

Semi-Automatic Dig Lines Definition A computational algorithm can be used to develop semi-automatically dig lines (Neufeld et al. 2005). While it is unlikely that all issues will be solved, always presenting the optimal solution, the process of defining dig lines can be sped up. It is expected, though, that a degree of manual intervention and validation will always be required.

The process of automatically defining dig limits is based on pre-defined operational and selection criteria. Figure 13.10 shows two cases for dig limits. The model used to define the ore/waste selection panels is the same in both cases; the difference is how much one dig limit considers the ability of the mining equipment to mine to the exact limits defined.

The optimal dig limits can be posed as an optimization problem. Sequential annealing (see Chap. 10) can be applied by defining the objective function as:

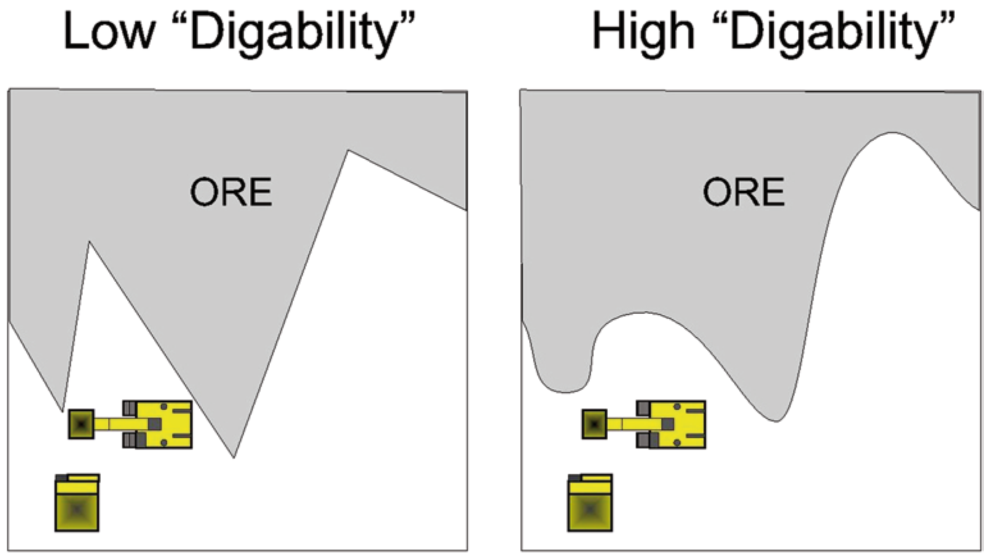


Fig. 13.10 Comparison of two ore/waste dig limits. The left option is more precise, but less realistic, and impossible for the shovel to dig to. Therefore, a large amount of unplanned dilution would be expected. The right option is a smoother dig limit, easier to dig for the shovel, but that it may be sub-optimal, depending on the characteristics of the mining equipment

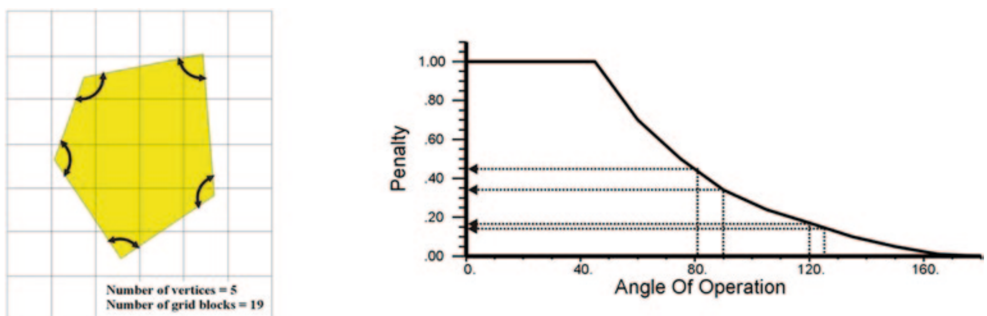


Fig. 13.11 Example of an ore polygon, with 5 vertices and affecting 19 blocks. The penalty assigned is a function of the angle of operation of the shovel

$$O_{global} = O_{profit} - O_{digability}$$

The initial profit is calculated as the sum of all fractional blocks that are considered ore (profitable):

$$O_{profit} = \sum_{ix=1}^{nx} \sum_{iy=1}^{ny} frac_{(ix,iy)} \cdot \overline{P}_{(ix,iy)}$$

where P represents the profit assigned to each block in the model, and "*frac*" represents the volume within each profitable block.

The initial digability is calculated based on the characteristics of the mining equipment, taken for example from an equipment curve, and interpreted as the sum of the penalties for each angle in the ore/waste polygon, see Fig. 13.11:

$$O_{digability} = \sum_{iv=1}^{nv} pen_{iv}$$

Using simulated annealing, the vertices and angles can be moved within a small circle (tolerance) to change the angle that it defines, and thus changing the penalty and overall profitability. A vertex is randomly selected and moved within a small distance (see Fig. 13.12). New profit and penalties are calculated, and the new objective function obtained. The results are sorted into accepted or rejected perturbations based on its impact on the objective function, and the process is iterated until convergence is achieved.

The dig limit selection algorithm can be made semi-automatic if the option of an additional constraint is added manually, allowing for the technician to account for the limitations of mining equipment and the value of the material. The dig limit algorithm works by systematically giving up ore or taking in additional waste to pay for the increased digability, i.e., less sharp angles defining the corners of the ore/waste selection panels.

13.6 Summary of Minimum, Good and Best Practices

At a minimum, all short-term models should be updated to include new data that becomes available. Proper procedures for validation and checking should be in place, and the complete sequence of updating the model should take less than a

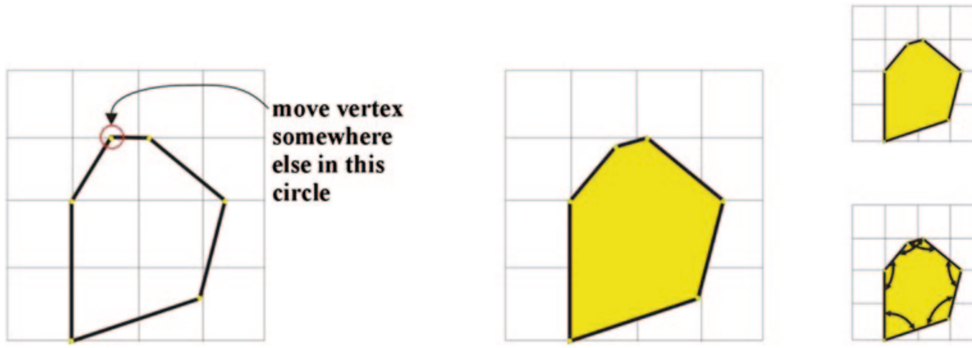


Fig. 13.12 A vertex is randomly selected and moved, a new shape obtained and new profit and penalties obtained

week of work. The ore/waste selection process will normally be based on a conventional method, perhaps some form of Kriging duly restricted with geology. Blast hole sampling should routinely provide acceptable samples for ore/waste selection. Information from relevant prior blasts should be used in defining current dig lines. Geologic mapping should aid in the daily task of defining the dig lines, which is generally a manual operation. Proper material accounting, reconciliation procedures, and constant presence and control by the mine geologist in the field should minimize the probability of making gross mistakes.

Good practice of medium- and short-term modeling requires a well defined and consistent methodology for updating the resource model, satisfying both the needs of short-term mine planning department and the short-term prediction of metallurgical performance. A sufficiently detailed study would have determined all the important implementation parameters and methodological details, including the procedures required to update the geologic model. The short-term models should be produced at regular time intervals, be always reconciled with recent past production, and compared against the original long-term resource model for the same areas. The model updating process should be semi-automatic, although always fully validated. Good practice in ore/waste selection requires the recognition of the limitation of selecting on grade, and therefore the use of an optimal selection method, with consideration of the basic economic parameters. Dig lines are usually hand drawn, and control and accounting procedures are strict. Reconciliation is usually kept on a blast-by-blast basis, and reported monthly.

Best practice in medium- and short-term modeling, in addition to the above, involves using conditional simulation models to provide for the uncertainty model and the risk assessment that short-term mine planners need. Other aspects of the model updating should be similar to what is defined as good practice, but the models are more likely to be simulation models. Similarly, the ore/waste selection should have been fully optimized, including the possibility of automatically drawing dig lines on a daily basis. In all cases, reconciliation procedures should be in place, and should be used to feed back and maintain an optimum implementation of the method as mine conditions change.

In addition, best practice in long- and medium-term modeling involves the development of dynamic models, which are constantly updated, not only in terms of grade estimation, but most importantly in terms of the geologic model. Production data and infill drilling are used with production mapping (drift or bench) to update on a regular basis portions of the long-term model that is therefore constantly up to date. It amounts to merging the medium and long-term model into a single model, updated, for example, on a monthly basis.

13.7 Exercises

The objective of this exercise is to review some concepts related to grade control. Some specific (geo)statistical software may be required. The functionality may be available in different public domain or commercial software. Please acquire the required software before beginning the exercise. The data files are available for download from the author's website—a search engine will reveal the location.

Consider the molybdenum data in `bh-data.dat`. You will be asked to conduct a full geostatistical study from histograms through simulation. The exercise will go quickly because the data are closely spaced and reasonably well behaved.

Question 1: Plot a location map and histogram of the Mo data. Comment on the spacing of the data. Your final estimation/simulation model should be at a spacing of about 1/3 to 1/2 of the blasthole spacing. We will not consider any volume averaging in the simulation. Decluster the data if you consider it necessary.

Question 2: Calculate and fit the variograms of the molybdenum grade and estimate a model with ordinary kriging. Perform cross validation if time permits and ensure that no conditional bias exists in the estimates.

Question 3: Calculate and fit the variograms of the normal scores transforms of molybdenum.

Question 4: Simulate 100 realizations of the grade. Plot the average grade and four realizations to verify that the simulated realizations are reasonable. The average grade model should look very close to the kriged model created previously.

Question 5: Calculate the expected profit assuming a cost/price/recovery structure that will give about 50 % ore in the model area.

Question 6: Establish initial polygon limits for an ore/waste interface. Optimize the dig limits for different digability settings.

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