

MODELOS CORTO Y MEDIANO PLAZO

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Resumen

Los modelos de recursos y reservas de factibilidad o de Largo Plazo no son finales. Se deben considerar estimaciones intermedias y parciales en el tiempo, que son rápidamente reemplazadas por nueva información de perforaciones o de producción. El objetivo y las tareas de estimación de recursos y reservas en momentos inmediatamente previos al minado o durante el minado cambian radicalmente con respecto a los objetivos y metodologías aplicadas en la estimación de largo plazo (estudios de factibilidad, por ejemplo).

Definimos como modelos de recursos y reservas de “Largo Plazo” aquellos modelos utilizados en Planificación Minera de Largo Plazo, que incluye típicamente planificación para períodos mayores a un año, y hasta la Vida de la Mina (LOM, según las siglas utilizadas comúnmente en inglés). Estos Modelos de Largo Plazo se utilizan para la predicción de flujos de caja, como es el caso de Estudios de Factibilidad, e incluyen una secuencia de minado y estimaciones de leyes y toneladas a ser recuperadas por volúmenes significativos pero variables. Por ejemplo, es común que la secuencia de minado se desarrolle por períodos equivalentes a un mes para los dos primeros años de la operación; posteriormente se consideren volúmenes de 6 meses para los dos años siguientes; y finalmente períodos anuales hasta el final de la vida de la mina.

Esto implica que no se tiene una estimación de toneladas y leyes recuperables, y por ende de flujos de cajas correspondientes, con mayor detalle o resolución que a esos volúmenes. De manera correspondiente, los modelos de recursos en los cuales se basan estos modelos de reservas y flujos de cajas se construyen a partir de información (perforaciones) que están ubicados a cierta distancia entre sí. La información y los métodos de obtención de estos modelos de recursos están relacionados lógicamente con su uso y objetivos, que es fundamentalmente estimaciones a volúmenes grandes (globales).

Aunque los modelos de Largo Plazo se construyen utilizando bloques de tamaño relativamente pequeño, dado que se diseñan para obtener buenas estimaciones en grandes volúmenes, no se puede esperar que tengan la misma exactitud cuando se analiza la estimación bloque a bloque. A menudo la exactitud global se consigue por compensación de errores a pequeña escala (bloque a bloque), siempre y cuando no existan sesgos significativos.

Es común que cuando comience la explotación del yacimiento se requiera información de mayor detalle, gradualmente relleno los sectores o espacios con poca información para mejorar las permitir estimaciones de recursos y reservas “locales”. Esto es el caso cuando se hacen planificaciones mensuales, donde el presupuesto anual o semestral está basado en el detalle mensual. La adición de información de relleno y la correspondiente actualización de los modelos de recursos y reservas resultan así en una predicción más confiable para la producción en el corto plazo.

La definición de Modelos de “Mediano” y “Corto” Plazo varía mucho según las operaciones y compañías mineras, y también por región geográfica. Para este trabajo, definimos Modelos de Mediano Plazo a aquellos modelos que se utilizan en la Planificación Mina mensual, trimestral, o bi-anual. Un Modelo de Corto Plazo se construye para servir de base a la Planificación semanal, y más comúnmente diaria; esto es, incluye los denominados Modelos de Control de Leyes, construidos a diario para definir la selección mineral/estéril.

MODELOS DE MEDIANO PLAZO

La actualización de un Modelo de Largo Plazo (MLP) utilizando sondeos de relleno o información de mayor detalle es básicamente el mismo proceso requerido para cualquier trabajo de modelamiento de recursos y reservas. Hay, sin embargo, algunas consideraciones importantes que distinguen a estos Modelos de Mediano Plazo (MMP) y que simplifican su obtención.

La actualización del modelo geológico cuando existen múltiples variables de interés que definen los dominios de estimación suele ser la tarea más onerosa. Típicamente se escogen zonas limitadas que definen el mes o trimestre a ser modelado (por ejemplo, volúmenes trimestrales "móviles" estimados mensualmente). El definir el modelo de Mediano Plazo solo para los volúmenes relevantes permite limitar el alcance del trabajo. Para esas zonas relevantes, se considera la nueva información geológica proveniente de perforaciones de relleno, descripciones de los cuttings de los pozos de tronadura, y mapeos de banco o de frentes de producción.

Como ejemplo de aplicación, las siguientes Figuras ilustran un Modelo Trimestral para una mina de Cu-Au.

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 month at a time. In order to develop a practical methodology and demonstrate the usefulness of updating the long-term resource model, a simple study was developed that had three basic components:

1. A conditional simulation model using the Sequential Gaussian and comprising the volume corresponding to the previous year of production, Fiscal Year 2001 (FY01, July 1, 2000-June 30, 2001). The simulation grid was 1 x 1 x 15m, and was used to compare the alternative models and methodology developed. The simulation model not only honored the histogram and variogram models of the conditioning data, but also was adjusted to reflect actual production figures from the mine to mill reconciliation procedures in place, to properly reflect actual production. The simulated variables were Total Copper (TCu), Sulfuric acid-soluble copper (SCu), arsenic (As) and total iron (Fe).
2. The volume to be mined in the following quarter is defined, and a reserve block model is created within. The blocks can be the same size, or, preferably, smaller if sufficient additional infill and/or blast hole data is available. In the case of the first-time study, for each month of the FY01 period, a quarterly model was defined based on actual mining volumes.
3. The geologic model is updated for each month of the year using the up-to-date information from mapping of benches, faces and blast hole cuttings. This is done so the proposed procedure is mimicked and as realistic as possible. For example, when completing the quarterly model for the month of January 2001, the planned mining volumes corresponding to the months of February through April are considered, and the geologic information up to December 31, 2000 is used.
4. The grade models (TCu, SCu, As, and Fe) are updated using all available infill drill hole and blast hole information through the previous month, December 31, 2000 in the example above. The same methodology as used for the long-term resource model is used, except that smaller block sizes are used, if warranted by the additional drill holes available. In this case, the long-term block model is 25 x 25 x 15m, while the quarterly model is based on 12.5 x 12.5 x 15m blocks; therefore, within each block of the long-term model there are 4 blocks of the quarterly model. It is always a good idea to define the quarterly block model in a manner consistent with the geometry of the long-term model, such that block by block comparisons can be made, and that a whole number of smaller blocks fit into a large one.
5. The quarterly models are compared to the long-term resource model against the reference simulation model, such that a quantification of the improvements is 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 the 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 smaller 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 the reverse circulation and, in to a lesser degree, in existing diamond drill holes.

The under-representation of TCu grades in the drill holes was explained by the loss of high grade chalcocite (copper sulphide), 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, simply because of their large numbers and also awareness of the problem. Therefore, to improve the short-term grade and tonnage above cutoff estimates, the following it was necessary to incorporate the most recent production information, including detailed geologic mapping, and infill and blast hole information.

Another important requisite is that the quarterly model be obtained in a short time, hopefully with 2 or 3 days of work, and without requiring significant additional resources other than those already available. An additional requirement is that the company's goal was to obtain a model with $\pm 5\%$ accuracy for both copper grades and tonnages above cutoff.

The conditional simulation model will not be described in detail, as it was only used as a reference to check against the resource models. Although conditional simulation models can be used on a routine basis to provide for more efficient short-term mine planning, in this example the simulation was used solely as a reference model, a „ground truth“ against which other modeling alternatives were compared.

The database used for the study and quarterly model updates is the same 15m down-the-hole 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. Since the blast holes represent the grade of a full 15m bench, then, in terms of support, except for small differences in the diameter of the drill and blast holes, it can be safely assumed that no consideration for support should be made.

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 homogeneizing nomenclatures and coding was completed.

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 bench (plan view) polygons. The polygons were adjusted bench by bench, from which three-dimensional solids were created. It is not necessary to develop the same level of detail as normally required for the long-term model (see Chapter 3), since the update is generally only an adjustment of a good prior interpretation given new, more detailed information. If unexpected geologic features are encountered, then it would be necessary to review the original geologic interpretation, the general data density, and overall question the ability of the long-term model to reasonably represent geologic controls.

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 around the edges.

The TCu, ScCu, Fe and As grades werre estimated using the same methodology as used in the long-term resource model, i.e., ordinary kriging 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 helped restrict the influence of the significantly more abundant blast data, restricting somewhat its effect.

Figure 13.2 shows, for the same area in bench 2845 as before, 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 as expected, significant for short-term planning, and mostly near the edges. The differences are both positive and negative, and better delineate areas of high and low grades. For example, observe at the northern tip of the area shown (North of coordinate 108,000N) that the quarterly model predicts a NW-trending higher-grade narrow vein or structure, which is higher than 3% TCu, shown in light brown, and not predicted by the long-term resource model.

Overall, results from the quarterly are as expected. The use of infill drilling and blast hole increases the grade and metal content of the reserve model, and at the same increases its variability. The local definition of geology and grade increases also the confidence level in the estimated values.

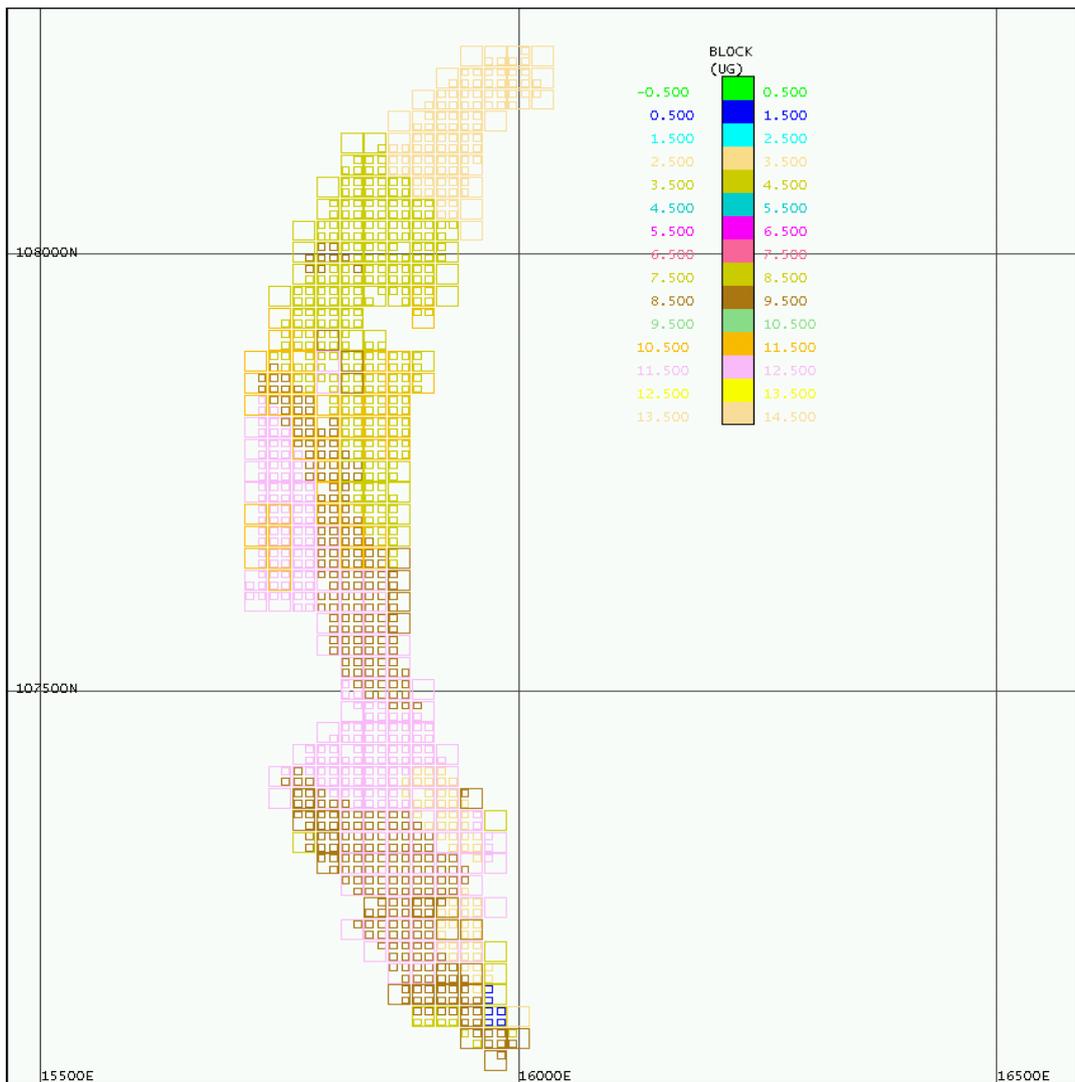


Figure 1: TCu Estimation Domains (UG) for the long-term and quarterly models, Bench 2845, Escondida Mine.

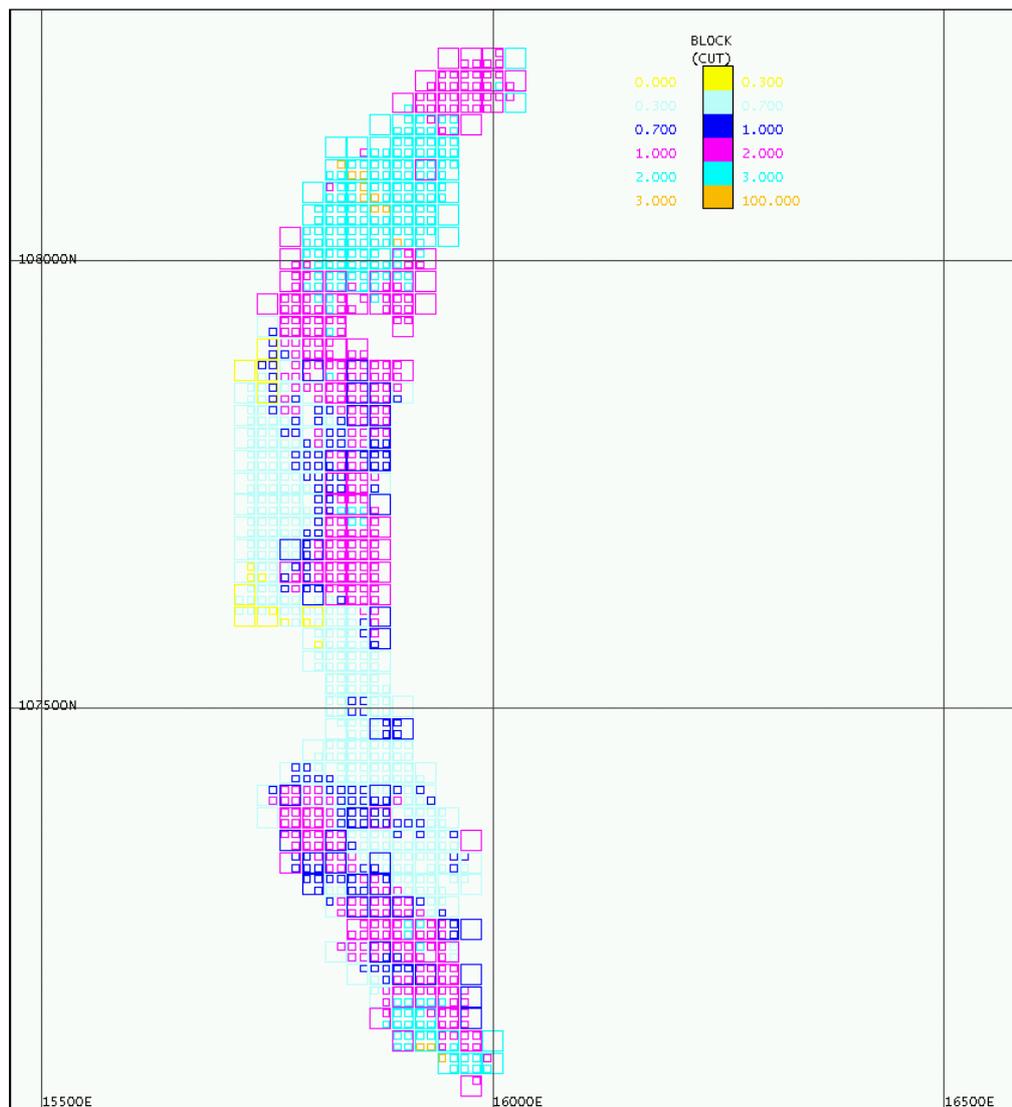


Figura 2: TCu grades for the long-term and quarterly models, Bench 2845, Escondida Mine.

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. For reference, the cutoffs of interest are 0.7% TCu (mill feed) and 0.3% TCu, the marginal stockpile cutoff.

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 the case particularly for Bench 2845, the grades shown also in Figure 13.2.

Figure 13.5 shows the comparison of the relative errors of monthly TCu grade averages between each model (LT and QT, three-month period beginning in February 2002) with the reference model, derived using conditional simulations and calibrated with production data, as explained above. Negative errors imply underestimation of TCu grades for the month. 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 much better the production grades from previous periods, and is therefore expected to be a better predictor of production grades.

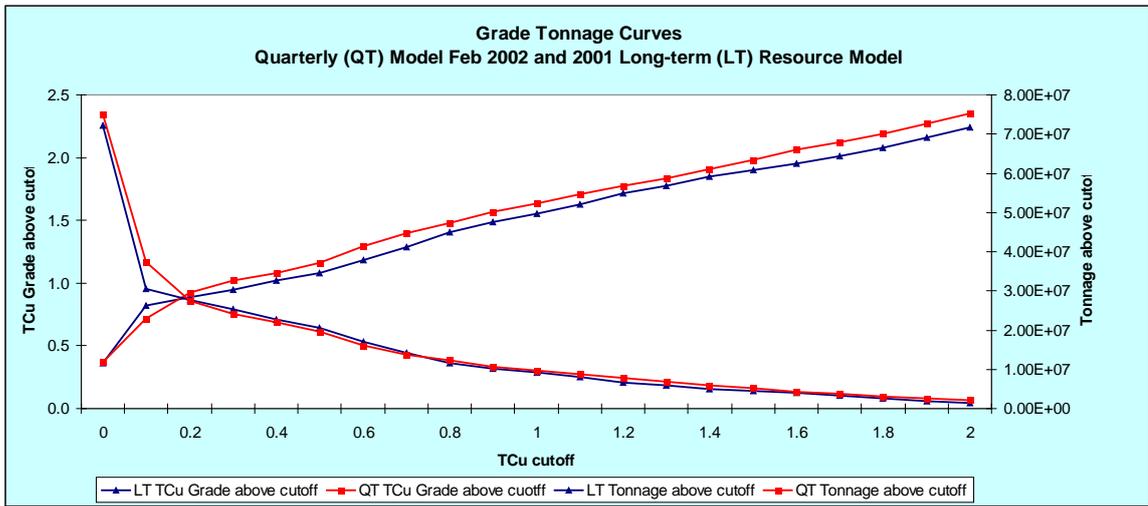


Figure 13.3: Grade-Tonnage curves, 2001 Long-term (LT) and February 2002 Quarterly (QT) resource models. Note how the QT model has higher grade and less tonnage than the LT model for most cutoffs.

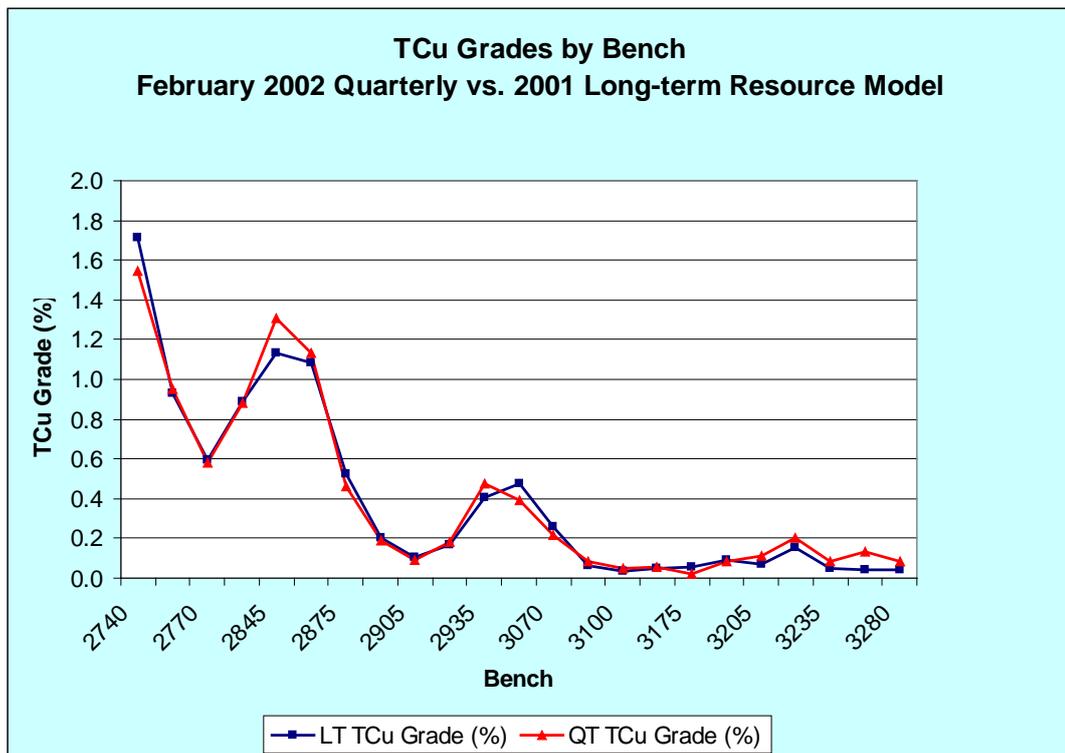


Figure 13.4: Total Copper Grades by Bench, 2001 Long-term (LT) and February 2002 Quarterly (QT) resource models.

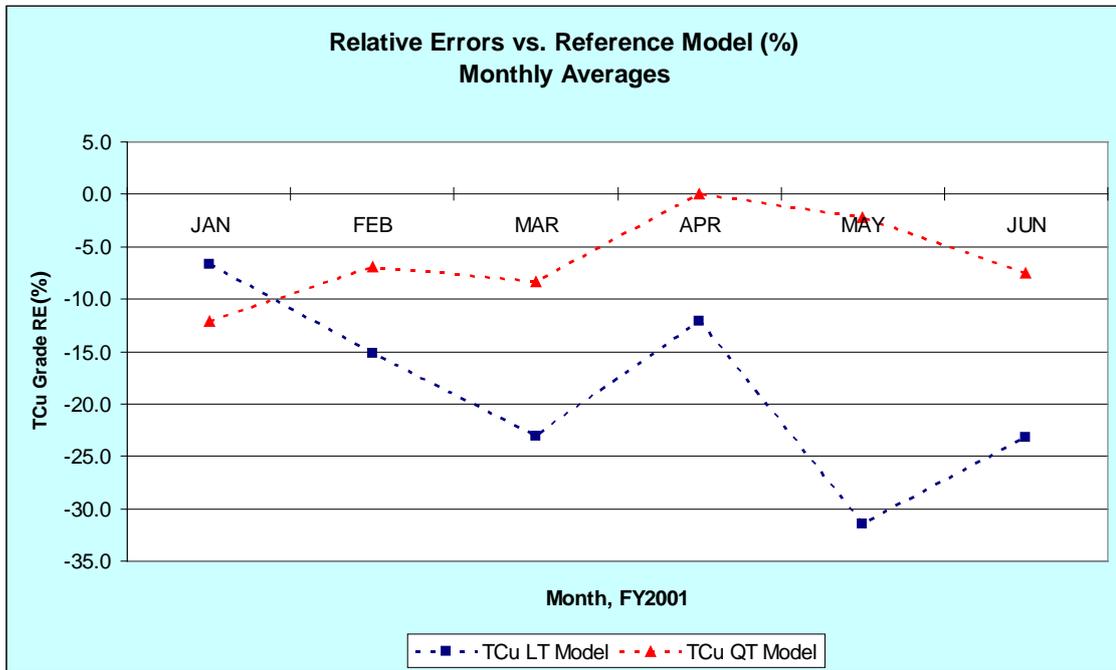


Figure 13.5: Relative Errors, Long-term (LT) and Quarterly (QT) models vs. Reference Model calibrated with production data.

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 important decision at the mine whereby 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 important because the resource is finite, and the time of selection is often the only and last opportunity that the mining company has to realize its expected revenue. It is also used to maximize resource recovery, or more frequently to optimize recovered dollar value. Also, the processing plant usually works better when a constant grade and tonnage is fed to it, although there are sometimes other options to avoid fluctuating grades and tonnages. There are four considerations in grade control. They are classification, cutoff grade, loss functions for grade control, and the approach for free selection.

Classification is the process of deciding what material to take where. A block is selected as ore if the revenue from processing it as ore exceeds the cost of mining as waste. As discussed in Chapter 7, the calculation of cutoff grades is complex and site specific. Many different costs and variables may come into play. One possible definition of a processing (also called marginal) 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 equation, costs such as General and Administration (G&A), and mining costs are not considered, only the additional costs that may exist of mining ore as opposed to waste. This cutoff grade is applicable when the operation has already committed to moving the material, and the only remaining decision is where it should go, which is the case when grade control is performed.

Grade control attempts to minimize miss-classification. The basic issue is shown in the schematic below, where a scatterplot of unknown true values for each block are plotted against the corresponding estimated grades:

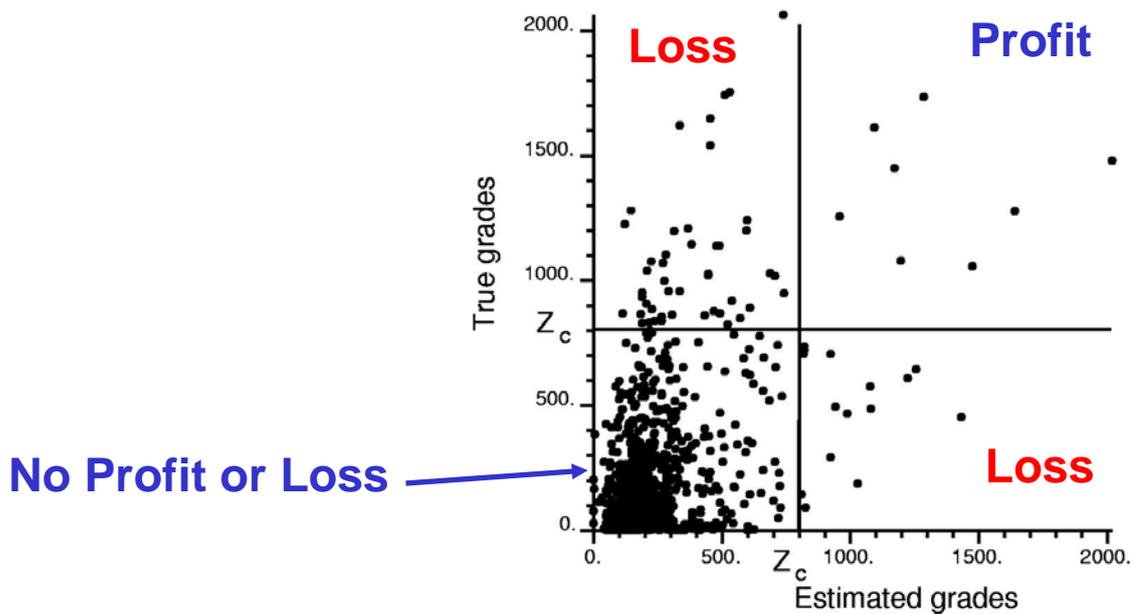


Figure 13.6: Loss function for grade control

Grade control can be referred to as a problem of reducing miss-classification, i.e., avoiding as much as possible sending material to the wrong destination. Section 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 in-situ values. Perfect selection is thus impossible, if only we can never be sure of the true in situ grades. There are also other practical (operational) factors affecting the decision, including how exactly the ore/waste markers are 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. 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.

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 the importance of operational procedures and constraints, generally through a combination of mining methods, mining practices, and culture.

First, data collection and data quality are always important, but it becomes even more critical when operational constraints limit the time and availability of samples, and thus may impact the overall quality of information used to make the decision. Secondly, the samples are modeled to provide for effective grade control involves some kind of prediction of either grades or block values, allowing for the actual selection of ore and waste to occur. And third, all related operational procedures should be considered. This is akin to the discussion on recoverable resources, i.e., how much dilution should a resource model (not a reserve model) include. In a similar manner, the grade control method chosen to predict in situ block grades or values should consider the type and possible limitations of available sample data and also operational constraints that may render any given grade control method impractical.

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. Occasionally, and particularly for vein-type deposits, dedicated reverse circulation (RC) 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. This is a fairly common practice in gold mines in Western Australia and parts of Africa, and generally applicable if ore is of high intrinsic value (such as high grade Au) and if the preferential higher-grade distribution is sub-vertical. However, the practice may not be beneficial if it costs more to implement it than the added economic benefit derived from improving grade control.

In the case of underground mines, the mining method is always much less flexible and therefore there is generally little 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, i.e., to perform 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 are indicated as the better options, since the ore and waste selection is dependent more on the variability of the grade distribution than on its average. Kriging-based methods can very easily fail (as can the more conventional methods) because of its characteristic smoothing effect, which for grade control is generally a problem.

Grade control models can sometimes be 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 be able to capitalize on that opportunity. It may be an overkill to develop and implement such a method, if the mining method is not selective enough to take advantage of the additional level of detail.

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 Chapter 3. Here the more common industry practices are discussed.

Unfortunately, even after major technological advances in many aspects of grade control including geostatistical modeling, most companies still do not fully appreciate the importance of grade control, and devote little resources and thought to this task. The flexibility that open pit mines generally enjoy is not always fully utilized, since many operations work with very simple methods that are not optimal with respect to what they could achieve with very little additional effort. This is also true, to a lesser extent, 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 the more recent geostatistical advances of the last 10 or 15 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 prediction.

Other commonly used methods include the nearest-neighbor method and inverse-distance methods 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.

KRIGING-BASED METHODS.

During the 1980's (mostly) kriging-based grade control became more popular. Different types of kriging algorithms were used, but most commonly ordinary and indicator kriging were applied, particularly in gold mines in Northern Nevada.

In the case of ordinary kriging, the application of 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 Chapter 8 and include the minimization of the estimation variance. In practice, kriging has not been much more successful at grade control compared to conventional methods because of the inherent smoothing, the minimization of the estimation variance which is not necessarily optimal for grade control, see for example Srivastava (1987), and the use of inadequate kriging plans.

Multiple variants of the indicator kriging approach have been used as well. Beginning in the 1980's several gold mines in western United States began pioneering the implementation of indicator kriging-based grade control methods. Most commonly, the 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, once again at a larger-than-necessary grid spacing. Occasionally, block kriging may be done while 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, see Chapter 9. Nonetheless, the practice is to analyze equal-probability contour lines for several values and decide based on visual observations which one adjusted better to prior production. Commonly, in many operations, probabilities of being ore of about 30 to 40% are used.

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

The BEI grade control method uses a combination of both indicator and grade kriging, whereby 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 krig an ore grade $Z_o(x)$ for the location x . Similarly, the waste-grade blast holes are used to krig 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:

$$E(R) = P_o \bullet R(Z_o) + (1 - P_o) \bullet R(Z_w) \quad (13.1)$$

The revenue function is traditionally calculated as $R = (\text{gold price}) \cdot (\text{metallurgical recovery}) \cdot (\text{grade}) - (\text{costs})$, where "costs" generally include metallurgical processing costs only, which defines the in-pit cutoff, without including mining, general administration, and other additional costs, in this case normally considered sunk costs. The method does offer the flexibility of adding additional costs if desired, to work on what would amount to a higher ore/waste cutoff grade.

From Equation 13.1, if the expected revenue 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 leading to a low value of probability of ore. In this case, the ore pays for large amounts of waste, which ensures all high grade ore is recovered, and hence a low probability of ore is required. Alternatively, if the ore grade is low, the revenue will tend to zero and the estimated probability of ore will be close to 1.0. Then, the adjacent ore will not pay for much overbreak, and 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, $E(R)=0$:

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

The method should be applied on a small scale (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. 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 or small, for which an improved grade control would be expected.

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, and therefore the estimated grades of the area.

EXAMPLE.

A comparison of several grade control methods was performed for the copper-molibdenum Ujina open pit mine, operated by Compañía Minera Doña Inés de Collahuasi (CMDIC) in Northern Chile. The deposit is a typical 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, since the grade distributions are not very variable. However, there are other factors that make grade control a very complex process, and thus, CMDIC wished to explore alternative methods to perform grade control.

The differences observed among the methods tested will be larger if the grade distributions being modeled are more variable, or if there are many different possible destinations for the ore and waste material, which charges the grade control process with increased complications, since 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 specific, and result in narrow grade bands and other characteristics affecting the destination of the material.

The methods tested included the inverse distance cubed (ID^3) used 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 to illustrate the differences that may be encountered among different grade control methods, even in deposits with relatively low grade distribution variability.

Table 13.1: Material type classification and destinations, Ujina Mine, as of 12/1999.

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

Figure 13.7 shows a small area of Bench 4270 with the Total Copper (TCu) blast hole grades and selection panels as defined by ID³, 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, Figure 13.9 shows the comparison for the same of the panels defined by both methods. In this area only sulphide material was present, corresponding to destinations (Stocks) 1 through 6 in Table 13.1. The intent of these figures is to demonstrate that, locally, the differences among the different grade control methods can be significant, even for a more homogeneous porphyry copper deposit.

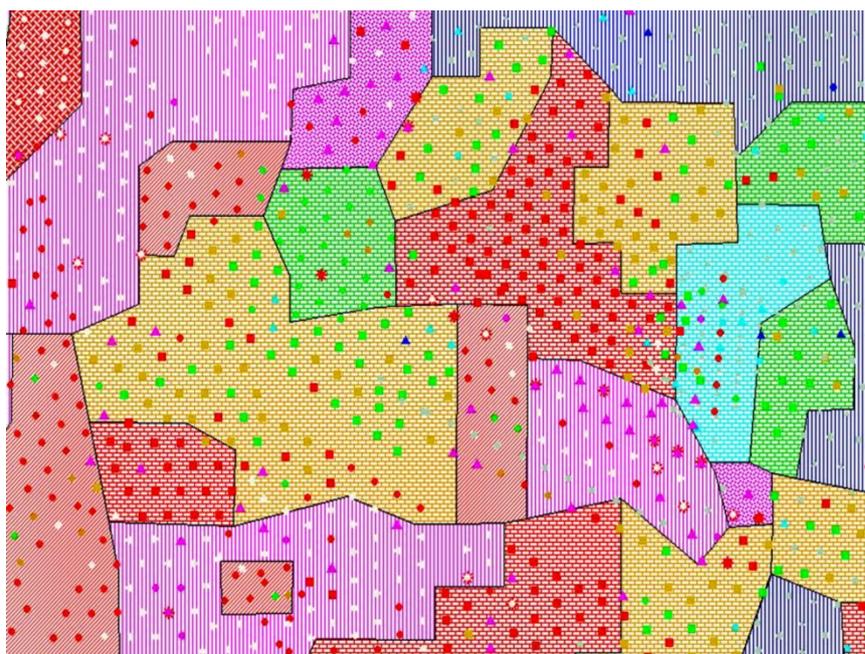


Figure 13.7: Blast holes, color- and shape coded by destination, and grade control panels based on ID³ interpolation. Blast hole spacing is approximately 8 x 8m, and the area is 250m per side. Blast holes and panel hatching represents Stocks 1 through 6 in Table 13.1.

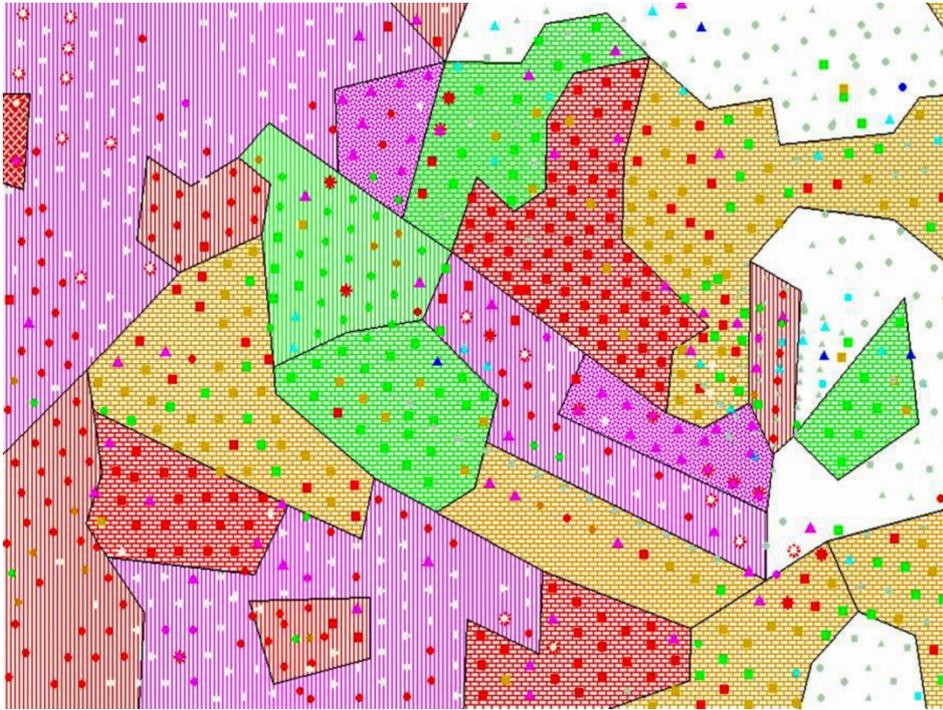


Figure 13.8: Blast holes, color- and shape coded by destination, and grade control panels based on the BEI method. Same area as Figure 13-7.

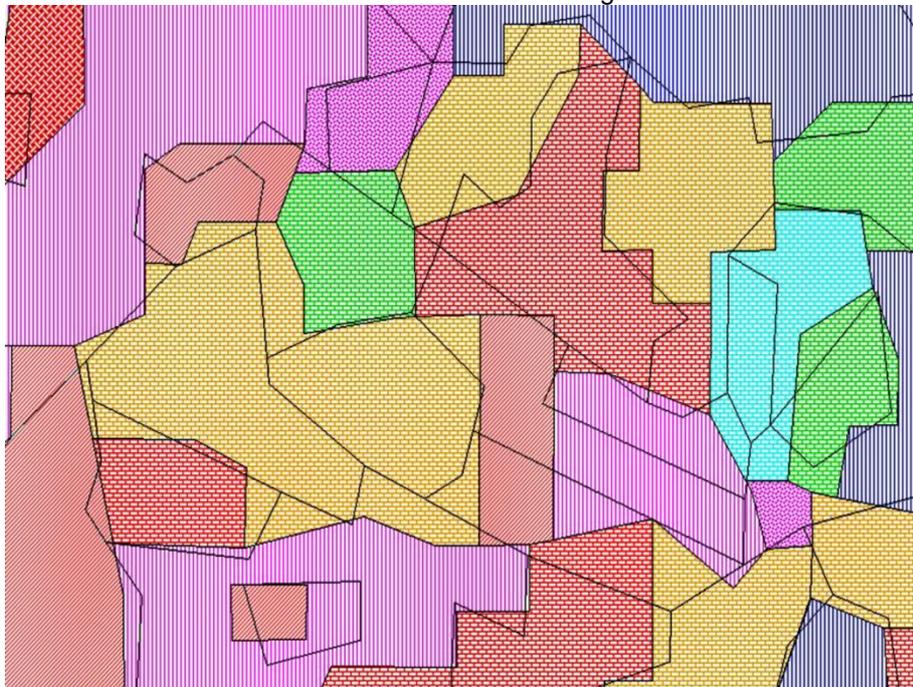


Figure 13.9: Comparison of grade control panels according to the ID³ used by the mine and the BEI methods, same area as Figures 13-7 and 13-8. Note the sometimes very different selection panels.

The comparison among the different methods was based on a reference model corresponding approximately to 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 detailed 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 ID³. For the sake of simplicity, all methods were evaluated against the SGS reference model, which was considered representative of approximately the past two year's production.

Tables 13.2 and 13.3 present the results for the ID³ and BEI methods only, as in this case the other two methods did produce marginally worse results. The Tables shows the relative performance of each 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 reference model, and, by extension and within the approximations of the reference model calibrations, actual production. 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 0.01 difference between the two methods represent 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, that 0.01 difference in contained Cu represented about US\$ 16 Million. In most cases, even though the differences in percentage points may be small, they may represent significant economic improvements given the size of the operation.

Table 13.2: Evaluation of the ID³ method by destination. All numbers correspond to TCu performance factors relative to the SGS reference model.

<i>Destination</i>	<i>Tonnage (Dest./Reference)</i>	<i>TCu Grade (Dest./Reference)</i>	<i>Cu Metal Content Cu (Dest./Reference)</i>
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: Evaluation of the BEI method by destination. All numbers correspond to TCu performance factors relative to the SGS reference model.

<i>Destination</i>	<i>Tonnage (Dest./Reference)</i>	<i>TCu Grade (Dest./Reference)</i>	<i>Cu Metal Content Cu (Dest./Reference)</i>
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

In addition to the added economic benefit brought about by the BEI method (with virtually no additional expenditure, since all operational practices remain the same), the panel drawing process is facilitated by the use of smaller blocks, and the lesser amount of sharp corners, compare Figures 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.

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, in addition considering that, as seen above, the ore category may include several destinations of recoverable material, including blending of material 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. An insightful discussion on this topic can be found in Srivastava (1987).

In open pit and underground grade control, the optimization criterion should always be maximizing the economic value of the recovered material. The basic intention is to select the material that will be metallurgically processed that provides the maximum possible economic benefit, given the operations mining constraints. Other possible optimization criteria, such as maximizing resource utilization as opposed to maximizing net present value of the mine (long-term vs. shorter-term economic views), is not applicable in the case of grade control, since the decision is short-term in nature, and aims at making basic make the most out of the current operation, thus maximizing daily the overall value of the operation.

Loss Functions can be used to optimize based on pre-determined functions that assign value to estimates, or equivalently, costs to mistakes, see for example Journel (1988), Isaaks (1990), and Goovaerts (1997). Conditional simulation can be used to assess local uncertainty, as discussed in Chapter 12, providing 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 through different variants with success in several open pit operations.

The basic idea of the free selection approach is to calculate the expected profit if we classify the material as ore and to calculate the expected profit if we classify it as waste. We choose the classification that maximizes profit. The expected profit calculation for each type of classification is shown:

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

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

establish c_{lo} by calculating revenue if it were milled $rev = prz^{(l)}(\mathbf{u})$:

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

MAXIMUM REVENUE GRADE CONTROL METHOD.

The Maximum Revenue grade control method is a two-step procedure, first outlined by Isaaks (1990). Initially, a set of conditional simulations is obtained from the blast hole data available; these conditional simulations provide information a uncertainty model for the grade at any specific point within the blast. Second, an economic optimization process using loss functions is implemented, to obtain the economically 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 full histogram and modeled measures of spatial continuity of the original, conditioning data. Therefore, they honor the spatial characteristics of the deposit as represented by 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 connectivity of low and high grade zones. These are fundamental variables for the optimization of the ore/waste selection procedures, which are likely to depend most on a correct prediction of the variability of high to medium-grade and to waste grade transitions.

Typical grade control simulation grids can be a 1m by 1m by bench height (corresponding to the full sampled blast hole column), which is generally small enough to allow the point simulation to be 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 very sensitive to departures from original assumptions, most importantly regarding stationarity, it is critical that they be controlled as much as possible by geologic models. Changes in the characteristics of the geologic ore controls require the separation of the data into different populations, as would any geologic or lithologic boundaries. This in turn 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 typically 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, refer among others to 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. If too many high grade simulated points are obtained, the contained metal in the simulation model will tend to suggest larger selection panels than otherwise. Often, the better method to resolve the issue is through calibration with existing production data for the corresponding geology.

Preferably, a large number of simulated images should be used to describe the posterior cumulative conditional probability curve (model of uncertainty), perhaps 100 for each simulated point. However, often a much smaller number, perhaps as small as 10 or 20 simulations in the case of grade control, can be used as an initial approximation. This may be necessary because of practical limitations, since grade control is a process that has to be completed in a short period of time, but also it may not be necessary of images to describe the model of uncertainty. In the case of a grade distribution with low variability (coefficient of variation) and the availability of a large amount of closely-spaced conditioning data, such as blast holes, the simulated values for each grid node will not vary significantly.

The model of uncertainty, or posterior conditional cumulative frequency curve, can be described as:

$$F(z; x | (n)) = \text{Prob}\{Z(x) \leq z | (n), \alpha = 1, \dots, n\} \quad (13.3)$$

where $F(z; x | (n))$ is the model of uncertainty for each point x of the simulated grid and obtained using the (n) , Conditioning blast holes. The model of uncertainty provides the probability of that node in the grid of being above (or below) any grade z .

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 $L(e)$ (Journel, 1988) attaches an economical value (impact or loss) to each possible error, measured in, for example, dollars. By applying a loss function to a set of equiprobable simulated grade values (Equation 13.3), then the expected conditional loss can be found by:

$$E\{L(z^* - Z) | (n)\} = \int_{-\infty}^{+\infty} L(z^* - z) \bullet dF(z; x | (n)) \quad (13.4)$$

The minimum expected loss can then be found by simply calculating the conditional expected loss for all possible values for the grade estimates, and retaining the estimate that minimizes the expected loss. As described in Isaaks (1990), in grade control the expected conditional loss is a step function whose value depends on the operating costs, and the relative costs of mis-classification. 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. In Chapter 14.10 a more detailed case study is presented.

MULTIVARIATE CASES.

Grade control in the presence of multiple variables introduces additional challenges that can be fairly easily handled. For example, 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, and the presence of clays. Therefore, the multiple variables can be all mine products, or a combination of mine products, metallurgical performance variables, and contaminants in general.

Through revenue or profit function the relationships of each variable to worth is accounted for. In cases where there are spatial relationships between the variables of interest, then either co-estimation or co-simulation (Chapters 8 through 10) can be performed. This is most important when simulating for grade control, since interactions among different variables when obtaining the full model of uncertainty process are generally more consequential than the case when only an expected value is sought after, as in estimating individual variables. In Chapter 14 a multivariate simulation case study is presented in detail.

13.5 PRACTICAL AND OPERATIONAL ASPECTS OF GRADE CONTROL.

There are many operational aspects that need to be considered for an effective grade control. Some of these relate to 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, where daily production targets are the main drivers and generally allow little room for detailed and careful mapping and planning work, d) the appropriate staking of the ore/waste zones, e) the control of the extraction procedures, f) the control of the destination of each truck or load of material, and g) the accounting of material movement and overall reconciliations.

Each one of the aspects mentioned deserve a much more detailed discussion, but are highlighted here to remind the reader that adequate grade or production control should involve multiple areas of an operation, and cannot be developed in isolation from other aspects of the mine. Issues related to material accounting, particularly volumetrics or tonnage extracted, and mine-to-mill reconciliations are among the most important, since, as argued in Chapter 11, they are the basis for performance evaluations.

AUTOMATIC DIG LINES DEFINITION.

Optimally placing the dig lines on blast hole plans and then in the field is a task that can be now more easily developed and managed with newer technology. The process of automatically defining dig limits is always based on the same pre-defined operational and selection criteria applied in the definition of the grade control method. 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.

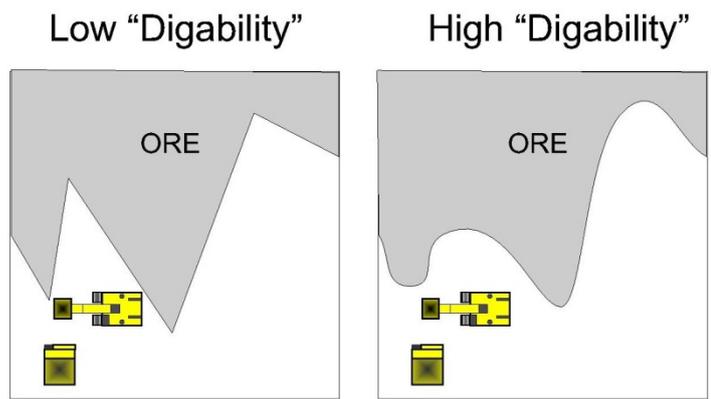


Figure 13.10: Comparison of two ore/waste dig limits. The left option is more precise, but less realistic, and impossible for the shovel to dig to. Therefore, a large amount of unplanned dilution would be expected. The right option is a smoother dig limit, easier to dig for the shovel, but that it may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

A method developed at the Centre for Computational Geostatistics of the University of Alberta is based on posing the optimal (in practice) dig limits as an optimization problem. Then, sequential annealing (see Chapter 10) can be applied by defining the objective function as:

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

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

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

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 Figure 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. This is an iterative process that can be carried out until convergence has been achieved. A vertex is randomly selected and moved within a small distance (see Figure 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.

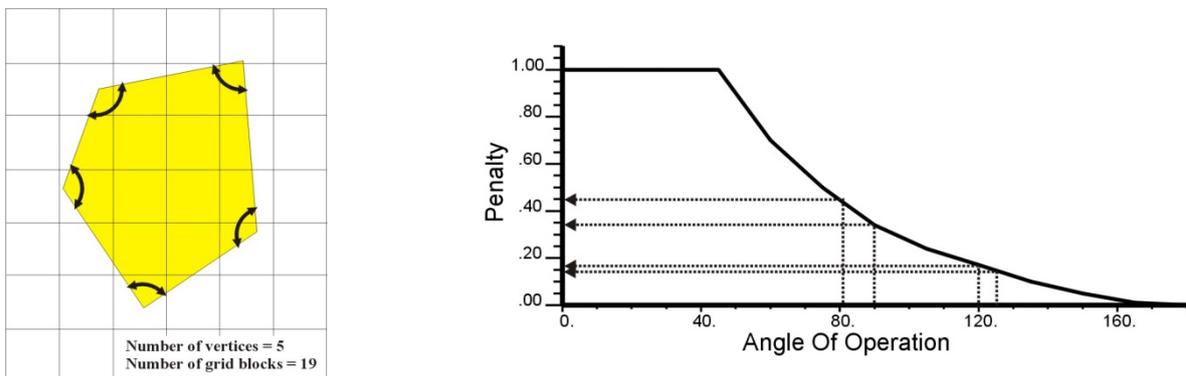


Figure 13.11: Example of an ore polygon, with 5 vertices and affecting 19 blocks. The penalty assigned is a function of the angle of operation of the shovel.

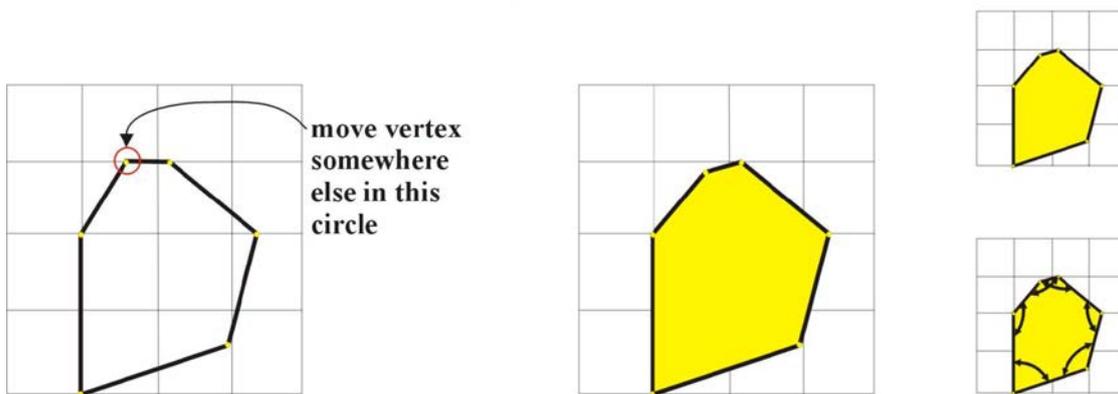


Figure 13.12: A vertex is randomly selected and moved, a new shape obtained and new profit and penalties obtained.

The dig limit selection algorithm can be made semi-automatic if the option of an additional constraint can be added manually, allowing for the technician accounts 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 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 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, and constant presence and control by the mine geologist in the field should minimize the probability of making gross mistakes.

Good practice of 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 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 should 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 are thus 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.