

Conclusions

Responsible mine maintenance management is critical to operations. Mine and mineral processing equipment can fail in many different modes, and, in some cases, failure has a negative impact on safety, equipment integrity, and the environment. In this regard, maintenance management should be based on the analyses of equipment functions, aiming to identify safety, environmental, and economic priorities, and concentrate on the maintenance strategies for those pieces of equipment and systems that are critical from the reliability, safety, environment, and production point of view.

CASE STUDY: GRADE CONTROL SYSTEMS AT EL VALLE-BOINÁS MINE*

Grade control refers to the process of assessing economic areas during mining operations. The objective is to determine which geological resources become reserves and thus can be mined at a profit. Grade control also involves the supervision of the mining operation in order to make sure that economic ore is sent to the plant and waste to the waste dump.

The scope of the grade control function varies with the characteristics of the mine; therefore, each mine will have its particular grade control system. Furthermore, for a specific mine, the grade control function cannot be defined before mining startup; it will be only after operation begins and several steps have been taken that the initial grade control system will reach its optimum state.

Depending on operational requirements, the grade control function can mean a simple visual delimitation of the ore–waste boundaries or a highly sophisticated process. In the present case study, in which visual delimitation of the ore–waste was not possible and grade variations were so vast, the ore–waste boundaries could only be defined through interpretation based in close pattern sampling and assaying.

This case study describes the scope and results of the grade control system that was implemented at the El Valle-Boinás open-pit gold mine in Spain, operated by Rio Narcea Gold Mines, S.A.

Project Goals and Objectives

The project had the following objectives:

- To establish a grade control system using state-of-the-art commercial software and hardware and also in-house software to optimize the process;
- To reduce the production cycle to a minimum to ensure that planned production was achieved;
- To implement the system gradually, thus avoiding taking the next step before the preceding one was verified and fully operational; and
- To assess system performance by means of reconciliation of mine and plant production data.

Work Methodology

The project methodology was planned along the following courses of action:

- Visiting similar mines in several countries
- Compiling existing public information on the subject, obtained from different publications and the Internet
- Starting with a system based on RecMin software, creating a simple and functional software application capable of exporting and importing information generated by other commercial mining programs, such as Datamine

* This section was written by C. Castañón.

- Defining the ideal work methodology for each unit of the production cycle:
 - Type of sampling
 - Drilling system (core drilling, direct or inverse circulation)
 - Sample sizes
 - Sample preparation process
 - Sample assay methodology
 - Calculation and interpolation method
 - Criteria for the definition of ore–waste boundaries
 - Staking and demarcation of ore–waste boundaries in the open pit
 - Loading/haulage systems
- Implementing quality control systems for each of these units

Geographical and Geological Framework

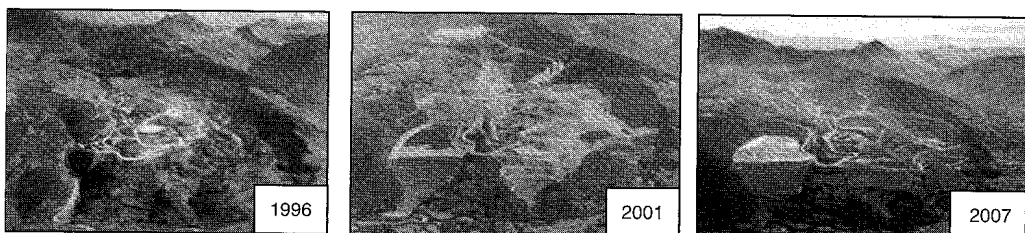
The El Valle-Boinás mineral deposit is located in Asturias (northwest Spain), approximately 65 km west of the city of Oviedo, in a hilly region with heights ranging from 300 m to 750 m above sea level.

The deposit is located in the Cantabrian zone, the outermost area of the Variscan orogen of the peninsular northwest, and is located in the core of the Asturian arc. Its occidental boundary is the Narcea antiform, which separates it from the west Asturian-Leonese zone and marks the transition to the internal areas of the Orogenics. All the Paleozoic systems are represented in the Cantabrian zone, although there are important stratigraphical differences among units. The structure is epidermic and is essentially made up of connected thrusts and folds.³⁰ The internal deformation is small and cleavage is only present in a few areas. Evolution of most of the Cantabrian zone took place in diagenetic conditions, and only a few areas underwent a low-grade or very-low-grade metamorphism.

El Valle-Boinás Mine

The El Valle-Boinás gold mine started operations in 1997 by open-pit mining from three pits located relatively close together. Today, open-pit ore has been mined out and mining continues underground. The treatment plant was originally designed to treat 500,000 t/yr of ore. Plant capacity was increased to 750,000 t/yr after a series of improvements in the process and in the plant facility.

During the 7 years of open-pit mining, a total of 4,600,000 t of ore were mined with an average grade of 5.8 g/t of gold. The aerial photographs (Figure 9.14) show the evolution of the mining works.



Courtesy of Rio Narcea Gold Mines

FIGURE 9.14 Evolution of the El Valle-Boinás open pit

Mineral Deposit Model

The Au–Cu mineral deposit in El Valle-Boinás can be considered of the skarn type,³¹ because approximately 90% of the gold and 100% of the copper contained in its ore reserves are of skarn genetic origin. On top of the skarn, there is an epithermal mineralization, which is associated with the existence of acid porphyritic bodies. These bodies essentially produce the silicification and argillization of the pre-existing rock and also produce gold, which is estimated to represent 10% of the gold found in the deposit. The resulting complex mineralization results in areas with very high variations of gold grades, with or without the presence of oxidized and nonoxidized copper. In addition, the ore is often composed of an irregular mix (breccia) of very soft clay rocks in contact with very hard silicified materials. All of these aspects have a very relevant influence on the selection of the steps of the grade control process.

Block Model

Block models are an essential tool when studying a mineral deposit for the estimation of geological resources and reserves. The technique is based on modeling the area of interest by parallelepipeds (blocks) of the same dimensions, each of which will be assigned a record in a database. Information about grade and any other properties that are required for calculations and studies will be recorded (e.g., lithologies, densities, analysis data, geotechnical data, and hydrogeological data).

The main advantage of a block model is the simplicity it brings to the digitization of the orebody and the application of computerized methods for interpolations, simulations, and the planning and design of open-pit or underground mining operations.

The block model of the orebody was developed using Datamine software. For this purpose, a borehole core database was created with the following data:

- Borehole collar coordinates
- Hole deviation data
- Lithological description of core
- Assay data of mineralized zones

The lithological information was obtained by geologists from borehole logging and by grouping lithologies according to significant changes in density, alterations, and mineralizations. Lithology was used to generate the geological sections and for modeling the orebody. Finally, the assay data, the orebody topography, and the rock type's densities were incorporated.

Computer modeling work consisted of digitizing into the Datamine software the different geological plans and sections of the orebody as interpreted by the geologists. After the sections with the different rock types were created, they were three-dimensionally connected to form a wireframe model. This is a 3-D envelope containing all the blocks of a given type of rock. The geological model of the orebody integrates the wireframe models for all the different lithologies.

Ore-bearing and waste-rock lithologies were determined through the statistical correlation of grades and rock types present in the deposit. When mineralization is strongly controlled by lithology, then lithology wireframes can be considered as ore envelopes. Otherwise, ore wireframes must be designed crossing the boundaries of the different lithologies.

To standardize on a fixed unit core length, the samples of different lengths were used to compose 4-m core length intervals by weighted average methods (composites). The composite length of 4 m was selected to match the block height and also the planned mining bench height.

The block model (with 4 m × 4 m × 4 m blocks) was generated using two methods: (1) nearest neighbor, and (2) inverse distance.

In the nearest neighbor method, the grade assigned to a block is equal to the grade of the nearest composite interval falling within a "cylinder of search" that is parallel to the general trend and to the dip of the orebody, always within the limits of the ore envelope that has previously been defined. The geology of each block is assigned using the same method.

In the inverse distance method, the grade assigned to a block is calculated as a weighted average of the grades of the composite intervals from several boreholes. This method takes into account the internal dilution of the block and therefore is more precise. In this case, a lentil-shaped "ellipsoid of search" was used.

Each block in the block model was identified with its x, y, z coordinates; ore grades for Au, Ag, Cu, As, and Bi; and other types of information, like resource classification, lithology, and so forth.

Specific Issues in Gold Mining

Gold mining introduces additional complexities to the grade control process:

- Working with very low grades, measured in parts per billion (ppb), makes assaying very time-consuming and expensive.
- The density of gold is very high (19.3 kg/L), which means sample preparation methodology must follow a very thorough procedure to avoid segregation during sample grinding and splitting processes.
- The nugget effect—that is, the gold grade variance associated with large gold grains (nuggets)—can be significant, and this entails very fine sample grinding in order to limit sample preparation errors. In addition, splitting the finely ground samples requires the use of specific equipment and work procedures.

In view of these factors, the design of the sample preparation method requires the prior completion of very detailed statistical studies aiming to evaluate and control the error. These studies must be repeated for the different zones within the orebodies because they can differ in their characteristics (e.g., degree of oxidation, alterations, grain size).

Open-Pit Bench Height

Typically, open-pit mines operate with different bench heights for the ore—where grade control is necessary—and for the waste.

The higher the ore bench, the lower the operating cost, but also, the less controlled the grade and subsequently the higher the dilution of the ore blocks grade with the surrounding waste blocks. This dilution effect will be higher for more irregular orebodies, such as where orebodies are irregular both in shape and dip. To determine the optimum bench height for the mining of the ore, the following constraints have to be taken into account:

- High benches are more productive. For a given production plan, there will be a minimum bench height below which the required production rates would not be achievable.
- Size of the mining equipment. The smaller the equipment is, the higher the unit cost will be and the lower the performance rates achieved. Ideally, the best approach is to use the same equipment sizes for both ore and waste material.
- The factor limiting the production rate for a given work area is the "productive cycle," that is, the total time required for drilling, blasting, sample preparation and assaying, and for getting the grade control information ready to define the ore–waste boundaries for the next cycle.
- The number of work areas, that is, the different mining areas in which work can be done independently from each other.

Ore Types

Ore types are defined by the following characteristics:

- The shape of the ore–waste boundaries in the ore zones. The higher the regularity of these boundaries, the easier the grade control function will be. The regularity in the vertical direction is most important, because it significantly affects the choice of bench height.
- Ore hardness. This is a very important property because soft ores are easier and cheaper to blast and the amount of blast displacement—an important source of error when staking ore–waste boundaries after blasting—is low. When blasting produces large ore displacements, it might be necessary to blast the waste in a first stage, followed by separate ore blasting, considerably delaying the productive cycle.
- The variability of the grades. If there are important changes in the grade in a short distance, much more detailed grade control is needed. It is also important to determine if there are any waste areas isolated within the ore.

Mine Production Startup

When a mining operation begins, the initial grade control concepts are normally revised and optimized as discussed previously.

In the block model case, the initial bench height was 4 m, which was selected to be equal to the height of the model blocks. Also, the elevations of the benches were designed to coincide with the elevation coordinate (z) of the block base in the model to allow for an easier updating of the block and the borehole model databases as new information was generated.

As mentioned previously, it was not possible to delimit the ore–waste boundaries visually, which meant that ore sampling and assaying was required.

Taking into account that mining activities take place above the ore zones that will be mined with descent from one bench to another, there are several possible options for ore sampling:

- Collecting samples from the top surface of the bench where mining is taking place. Samples can be collected from panels, channels, or short inclined boreholes drilled in close pattern. The main problem with this system is the sample contamination caused by heavy equipment working on top of the bench. Furthermore, this method is not advisable for soft ores, when crushed rock fill might be required to repair working surfaces after heavy rains. With harder ores, it might be an acceptable system.
- Sampling from boreholes. In this case, three types of borehole drilling can be distinguished:
 - Core borehole drilling. Without doubt, this is the best sampling method because the entire core of rock drilled can be seen and it can be assayed in part or in full. However, the high price and the longer time required makes this method seldom used for grade control.
 - Reverse circulation drilling. This is a valid option when samples at different depths are sought in order to get assay data from several benches in advance of mining. In this drilling method, drill cuttings circulate inside the drilling pipes, thus eliminating contamination from the hole walls, which can be important in gold mining. However, the cost of this drilling method is around seven times higher than direct circulation drilling.
 - Direct circulation borehole drilling. This is the most used method, mainly because samples can be taken from blastholes with no additional drilling cost, but also because it allows for a closer drilling and sampling pattern wherever needed at a lower cost. As a drawback, the quality of the sample is less than when using inverse circulation methods,

especially when attempts are made to take samples in the same borehole at different depths. However, this disadvantage is balanced by the possibility of reducing the sampling error by drilling and sampling with much closer patterns at a lower price.

Grade Distribution

One of the first steps to take at the beginning of the mining in one area, or whenever the ore type changes, is to perform enough testing and assaying to determine the distribution of gold values within mineralized zones. This can result in a significant change of grade control procedures that might require higher or lower levels of complication for each ore type.

The objective is to determine whether the gold contained by the orebody is distributed in a more or less homogeneous manner or, on the contrary, is concentrated in certain zones of the orebody. These can be small faults or alterations, contacts between different lithologies, specific rock types, and so forth. In the first case, grade control would be easy because it would consist of just delimiting the external boundaries of the orebody. In the second case, it would be more complicated because as ore is concentrated in some zones in the orebody, the amount of information and the sampling density must be increased to allow for the definition of ore–waste boundaries and to be certain to avoid sending ore to the waste dump or waste to the plant.

The procedure used at El Valle-Boinás was to define a volume of 27 m × 5 m × 5 m within the ore zone to be mined in the first bench of the mine and to obtain the following information from it:

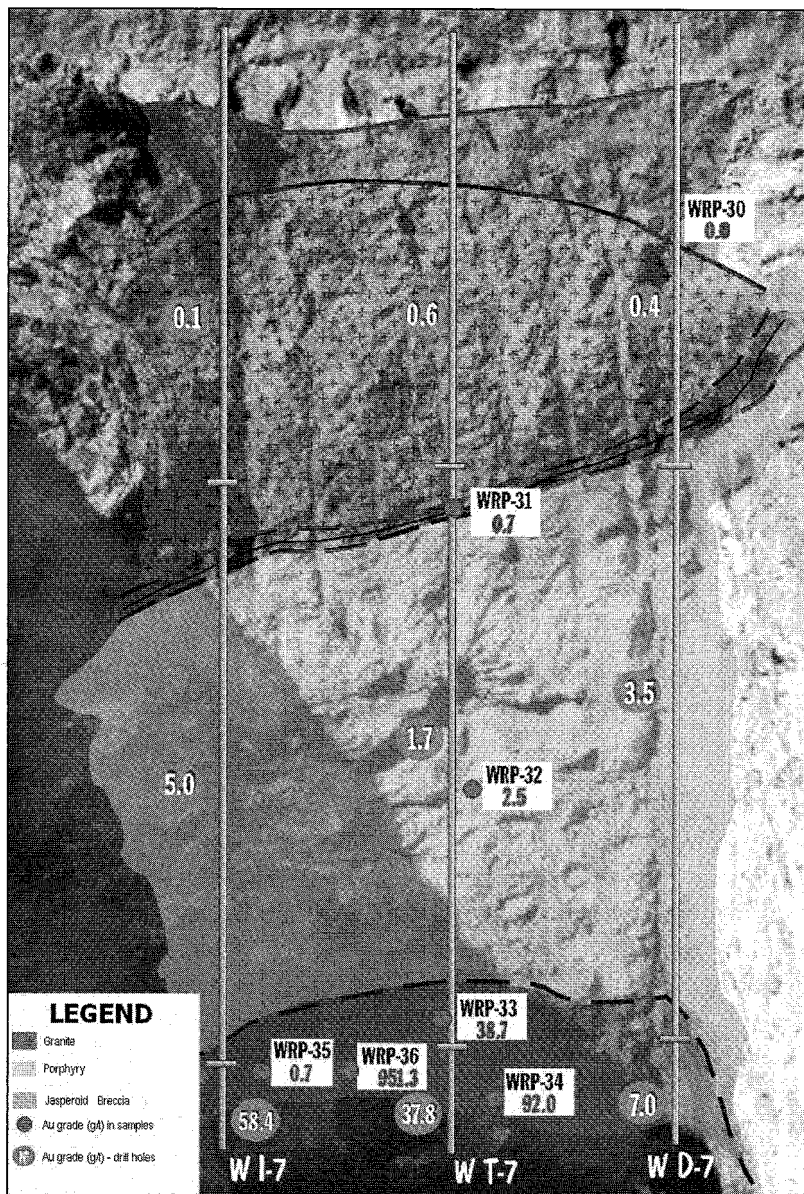
- Lithological information and assay data from three lines of boreholes, each drilled with a different method (core holes, reverse circulation, and direct circulation);
- Detailed geological interpretation from several mining faces corresponding with longitudinal and cross sections of the volume under study, including type of alteration, joints, small faults, lithologies, and so forth; and
- Sampling from the said mining faces, using different sampling methods, such as panels of different sizes, linear channels, and spot samples from fault fill material, zones of alteration, lithological contacts, and so forth.

Figure 9.15 shows a section with the detailed geological information, the boreholes, and the sample assays taken from both the boreholes and the face samples. It was mainly concluded that there could have been small local zones with very high grades, as much as 950 g/t. Although this can be seen in borehole samples, it is clear that there is no homogeneity in the grades. This means that there must be a borehole sampling pattern as tight as possible and that benches of the minimum possible height must be mined.

Work Cycle

As mentioned previously, considering the great variability of gold grades and the small size of the open pits, which only allowed for a maximum of two independent work areas, the grade control system had to be based on systematic sampling and on the minimum bench height that would allow achievement of the planned production rates—which were around 14,000 t of ore per week using medium-sized mining equipment (the plant treated an average of 2,000 t/d, 7 days a week).

Mining equipment can operate at an average production rate of 300 m³/h when loading ore. However, it's important to remember that loading ore is not as productive as loading waste with the same type of equipment due to the need to mine ore and waste selectively at the same working faces.



Courtesy of Rio Narcea Gold Mines

FIGURE 9.15 Detailed geological section

Bench height was fixed at 4 m, coinciding with the block model. Benches of less than 4 m would lead to loading problems and difficulties in achieving the desired production rate, among other problems.

Mining equipment works in 10-hour shifts, so a minimum of 3,000 m³ must be staked for loading at the beginning of the shift. This means that for a bench height of 4 m, a minimum working area in ore of 750 m² for every shift is needed.

TABLE 9.2 Time chart of a complete mining cycle

Operation	Time, h
Drilling and sampling	8
Sample drying	20
Sample preparation	8
Sample assaying—ore grade	8
Sample assaying—plant samples	2
Staking ore zones by surveyors	2
Loading and haulage	10
Total	58

Source: Data from Rio Narcea Gold Mines

For grade control drilling and sampling, direct circulation borehole drilling was employed, using the same holes used for the blasting but increasing the sampling density by adding a hole in the center of the blasting pattern. As explained later, the resulting drill pattern was $3\text{ m} \times 2.5\text{ m}$, which means a total of 100 boreholes for the defined work area (750 m^2). Therefore, there were 100 samples to collect, prepare, and analyze.

Sample preparation was estimated to require 28 hours—20 hours for drying the ore, and 8 hours for sample preparation (grinding and splitting). Processing the samples at the laboratory was estimated to take 8 hours of work.

For data processing and the definition of ore–waste boundaries, a computerized system based in RecMin software (Mineral Resources) was arranged. The system processed all the information on hole survey data, lithology of drill cuttings, and sample assays data required for the interpretation and interpolation processes and the definition of the ore–waste boundaries in 2 hours.

The time for staking of ore–waste boundaries by the surveyors was also estimated at 2 hours. The total time to complete each production cycle is summarized in the timetable in Table 9.2.

Although the total duration of each cycle was estimated at 58 hours, not all the processes work on a 24-hour-per-day schedule, and some of them—like sample preparation and assaying—must be shared with other needs like exploration drilling, plant assaying, and so forth. Therefore, the drilling equipment was scheduled to work from 8:00 AM to 6:00 PM, Monday through Friday; sample preparation and laboratory worked 24 hours per day, 7 days a week; and geology and surveying worked from 9:00 AM to 6:00 PM, Monday through Friday.

The timetable shown in Table 9.2 outlines a complete example cycle, from the initial drilling in the work area to loading, transportation, and bench preparation for the initiation of a new cycle. Three days were necessary to complete each cycle; therefore, three times the calculated area for each cycle was needed to be able to keep the loading and transportation teams working. Therefore, $3 \times 750 = 2,250\text{ m}^2$ was needed.

Given the use of two loading and transportation teams working in independent areas, a total of two areas was needed, each with $2,250\text{ m}^2$ of available surface to achieve the planned production of $6,000\text{ m}^3/\text{d}$ that—at an average density of 2.2 t/m^3 —would yield around $13,200\text{ t/d}$. Therefore, it was necessary to drill and sample a total of 200 grade control boreholes per day.

Within the total surface area that is sampled and assayed for grade control, considering that only about 25% is defined as ore, the grade control system would have to allow for an ore production of around $3,200\text{ t/d}$, 5 days a week; this is a total of $16,000\text{ t/week}$.

The case described here is an ideal one. In reality, plans typically call for intensifying the work during the summer to increase the stock of ore, then slowing down in winter, when fewer natural-light work hours are available and more downtime can be expected because of weather conditions.

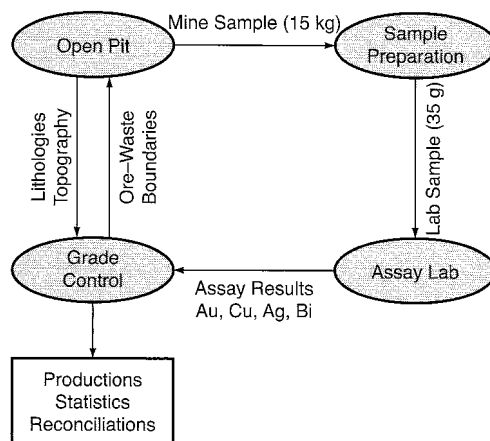


FIGURE 9.16 Grade control information flow

Information Management

Managing around 200 samples per day means that approximately 3,000 kg of samples must be transported, dried, prepared, and assayed every day and also that the grade control team must process the lithological and analytical information. This represents generating a great deal of information, such as borehole coordinates, borehole and sample coding, lithological information from each sample, analytical data from several elements in each sample, and so forth. In total, 3,200 pieces of data must be processed each day, not taking into account the information generated later when processing this data in terms of interpolation with the block model, marking ore–waste boundary lines for the economic areas, and so forth.

The diagram in Figure 9.16 outlines the type of information that is generated and where.

To allow managing all this information in a fast and safe manner, a system for coding, reading, and generating the information using barcodes was implemented and linked with the RecMin software. This system interconnects all departments and work places using a local intranet with wireless connection with the open pits, thereby allowing all the information to be centralized in a single database on the server. Bar coding was used to eliminate the need for each department to duplicate the process of keying in codes, thereby eliminating time losses and reducing the risk of making mistakes.

The coding was designed in such a way that the reports generated are automatically transferred to the corresponding place in the database. Sample codes were designed to be read from right to left, so that the last item to be read is the borehole code that represents where the sample was taken. It was designed that way so that it is the name of the borehole that varies in size; the code of the sample also begins with the name of the borehole, which makes identification easier.

In addition to the grade control sample codes, other barcodes were designed for other types of samples—like exploration boreholes, treatment plant samples, and so forth. All these allowed for easier information management and reduced operating errors.

Drilling and Sampling

The grade control boreholes were drilled with the same drilling equipment as the blastholes, that is, hydraulic drilling machines with a hammer at the head. These machines were modified in some areas so they could be used to collect samples.

The drilling pattern was defined as 2.5 m $\times$ 3 m, with 3 m in the longitudinal direction of the mineralization and 2.5 m in the transversal, with the objective of better limiting the transverse ore limits with more boreholes.

The holes were drilled vertically, 3.5 in. in diameter and 4 m long, which is equal to the bench height. In some areas, two benches were drilled in a single drilling pass, collecting two 4-m samples where each bench was mined separately. By doing this, the work cycle was shortened, but the quality of the sample from the lower bench might be reduced because it could become contaminated by the walls of the first 4 m.

The total quantity of drill cutting in each sample ranged between 52 and 68 kg, depending on the density of the material. In order to reduce the size of the samples, which was too large for handling at an acceptable cost, a sample splitter was prepared and placed under the cyclone of the machine. This method allowed separating a sample from 13 to 17 kg (one-fourth of the total cuttings) directly into a bag.

Part of the drill cuttings are lost and eliminated by the drill through another duct. It makes up around 5% of total cuttings and contains the finest part. To make sure that the error made discarding it is acceptable, tests were conducted comparing both the cyclone cuttings and the fine discarded cuttings. The results gave grades sufficiently close to consider that the error was not significant.

Geological Description and Sample Labeling

The drilling of grade control boreholes is subject to continuous monitoring by a geologist with the following functions:

- To check the 3 m $\times$ 2.5 m borehole pattern. This ensures it has been correctly staked and marked on the ground.
- To tag the stakes with the borehole codes according to the established order. These stakes will be next to the borehole permanently and will be used to identify the lithological and topographical information and the samples correctly.
- To tag the bags with samples that are produced by each drilled borehole with the corresponding codes. A copy of the code is placed inside the bag so it can be later used in the drying, preparation, and laboratory processes.
- To define the lithology for each sample. This information is entered in the database.
- To make sure that the boreholes are surveyed after they are finished.

Considering that samples are made up of drill cuttings, their description by the geologist is greatly assisted by the experience acquired from visual inspection of the ore drilled at the time of loading.

To enter the data, the geologist has a RecMin module in a personal digital assistant that allows for easy and quick input of the information, as well as for wireless transmission from the pit itself to the central database.

Sample Preparation

The preparation of samples for analysis in the laboratory is a delicate process for several reasons:

- The high density of gold (19.3 kg/L) facilitates the segregation of gold from ore. Because of this, sample handling and splitting processes must be done according to very specific rules.
- Sample grades are measured in parts per billion (ppb) of gold, and these are very small amounts, which require working with samples as large as possible in order to reduce sampling errors.

- The nugget effect is also important for sample size selection. For a given sample size, when gold is present in small grain sizes, sampling error is reduced; but conversely, if the gold grain sizes are large, sampling error will be larger and can only be reduced by taking larger samples, which in turn, would mean a more costly and labor-intensive sample preparation process.

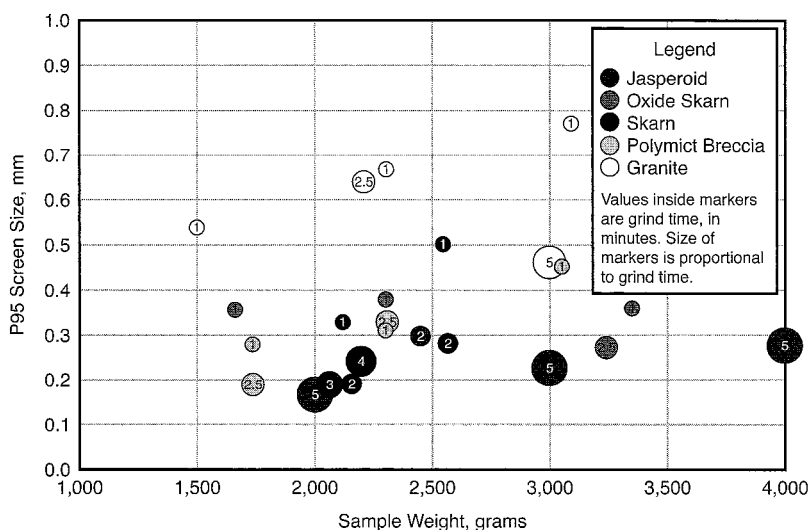
If one imagines drilling a borehole every $2.5 \text{ m} \times 3 \text{ m} \times 4 \text{ m}$ for a total of 30 m^3 , or around 66 t, it becomes obvious that by sampling and assaying all 66 t, the sampling error would be nil. Given that this is impossible, then the sampling problem consists of taking a sample of a smaller size that is representative of the entire 66-t block of ore. Obviously, the smaller this sample is, the greater the chance of error.

The error is also dependent on the size of the grains of gold. The bigger these are, the larger the error. If all the gold is in just a few large gold grains, it will be very difficult to get a representative sample. Taking this idea to the extreme, if all the gold in the block was in the form of a single nugget, then the sample taken would either contain the nugget or would contain no gold. Conversely, if the grain size is small, the number of grains will be larger and any sample taken will be more representative of the whole.

To be able to determine the best sample preparation and assaying method, some comparison tests must first be performed to define the maximum margin of error that would be acceptable at a reasonable cost.

The tests were performed using the following procedure:

1. The first step involves looking at the size of the gold grains based on one or several representative samples. The gold grains are separated using gravimetric methods, then studied in the laboratory under a microscope in order to define the larger sizes that can be expected. With these tests, the size of the sample and the degree of grinding required for a sample as homogeneous as possible within a reasonable cost can be defined.
2. The 15-kg sample that is collected from the drill cuttings undergoes a process of drying and assay sample preparation, which yields a final sample of approximately 30 to 100 g, ground to a size of approximately 100 microns. This process entails three grinding processes, each followed by a sample splitting process. To optimize the sample preparation process, testing must be undertaken that involves comparing similar samples ground at different sizes and with different sample size reductions in each step. Using a statistical comparison of the results, a process can be designed that yields an acceptable error at a reasonable cost.
3. The drying process consists of drying the sample in an oven at 90°C for around 20 hours. This step aims to prevent contamination between samples caused by sample grains getting stuck to the grinding rollers and disks. Also, a sample of gold-free silica is ground between consecutive ore samples in order to clean the grinding mill.
4. The sample preparation laboratory has one roller mill, one disc LM5 mill, and one LM1 or LM2 disk or micronizing mill. In each of these mills, the size reduction tests necessary for the sample to be ground need to be conducted. The results will depend on the hardness and the size of the ore samples. If different types of ore are used, different processes must be defined.
5. The drill cuttings samples are smaller than 6 mm (>95%). For the roller mill, after it is adjusted, the size reduction required and the average amount of time needed for the preparation of the sample will be tested. With respect to the LM5 and the LM2 or LM1 disc mills, different grinding tests for each type of ore must be performed, varying the grinding



Courtesy of Rio Narcea Gold Mines

FIGURE 9.17 Sample preparation tests: size versus sample weight and grinding time

times and the attained sizes. In this way, graphical representations, as shown in Figure 9.17, can be obtained in which the size reductions in relation to the sample weight and the grinding time can be seen.

- Finally, after the results of the tests have been studied, several sample preparation scenarios will be defined. The lower the cost of the preparation process, the larger the error that will be incurred. At the end, the scenario that yields the lower error at a reasonable cost will be chosen.

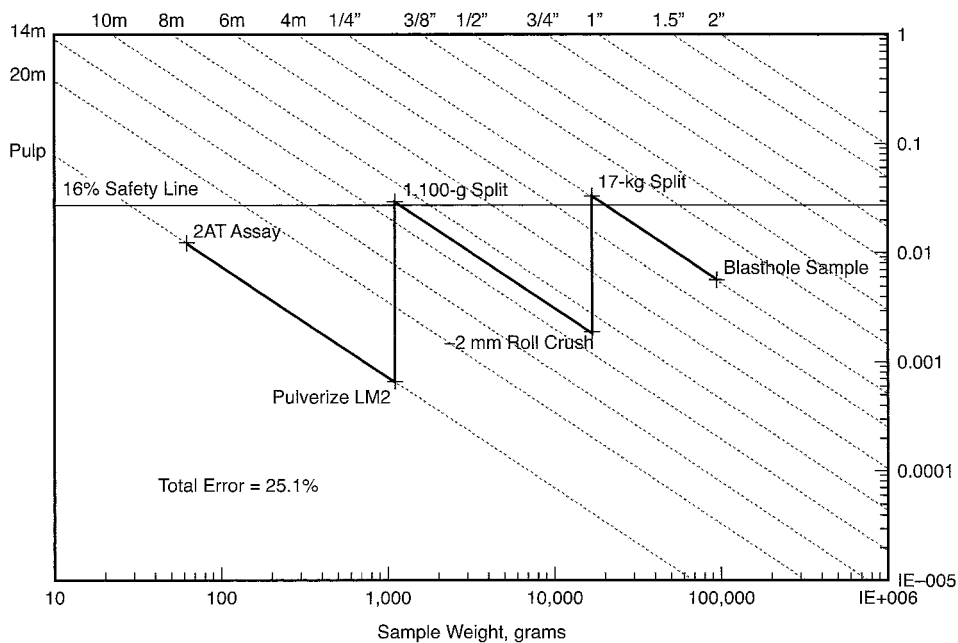
In the graph in Figure 9.18, the amount of error incurred in one of the cases studied can be seen. As shown, the process began with a large sample, approximately 90 kg, which was then split at the drill down to 17 kg. After drying, it was ground with the roller mill down to 2 mm maximum grain size (>95%). Then, the sample was split-reduced to 1,100 g, ground and split-reduced again to achieve a sample of 60 g. As shown in the graph, the total error incurred was around 25.1%, calculated as the sum of the errors obtained in the three grinding–splitting processes.

The sample preparation method finally selected is shown in Figure 9.19. The final sample size is 35 g and, as can be seen later, it will be divided in the lab into a 30-g sample for assaying of gold and a 2-g sample for assaying of copper and other elements. However, the sample bag that is sent to the laboratory with the prepared sample usually weighs around 100 g, of which the laboratory separates 30 g for gold and 2 g for copper and other elements; the rest is stored for several days in case it is needed for additional assays or if any assays need to be duplicated.

Sample Assaying

The size of the sample that will be assayed in the laboratory will be established after several tests with different sample sizes and a comparison study of the error incurred.

Usually, assay sample sizes will range between 30 and 100 g. In the case of gold, the assay procedure requires costly fusion and copellation processes; therefore, reducing sample size is important in reducing the final cost.



Courtesy of Rio Narcea Gold Mines

FIGURE 9.18 Sample preparation protocol required for acceptable accuracy on 4-m blasthole sample versus sample error

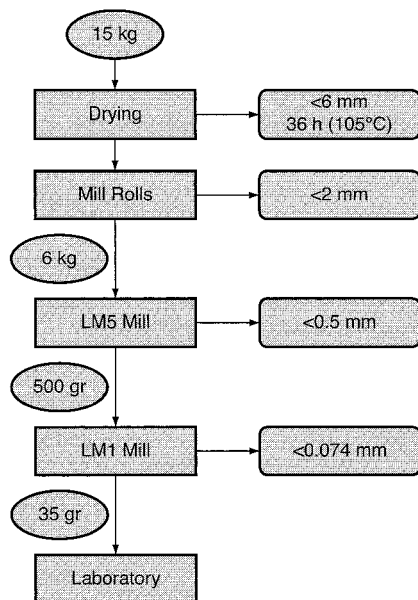
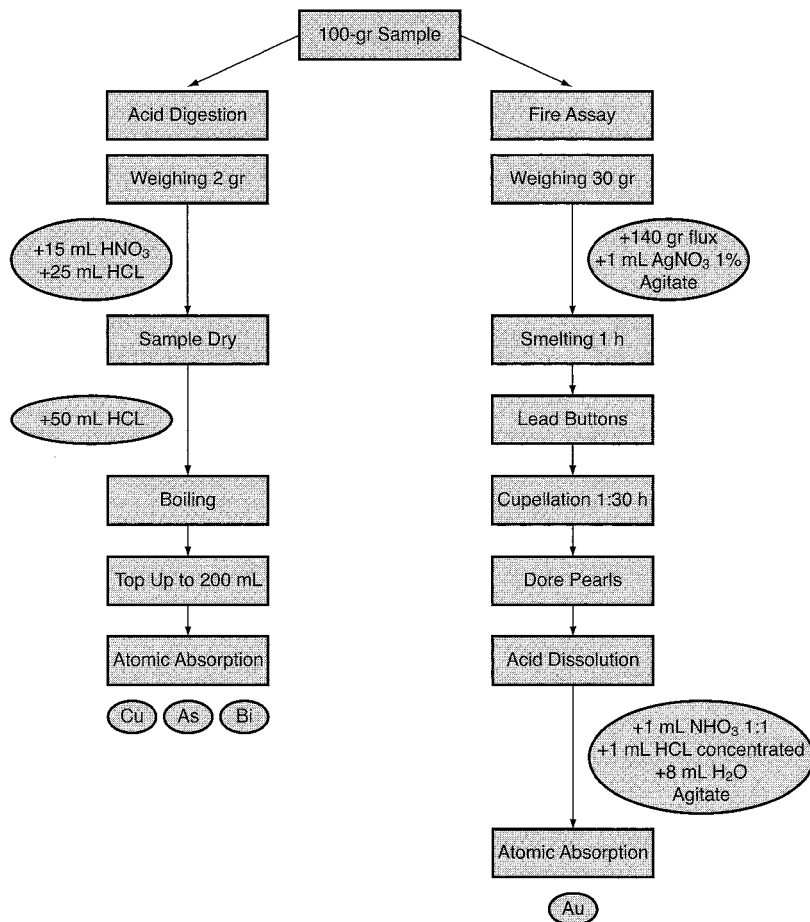


FIGURE 9.19 Sample preparation procedure



Courtesy of Rio Narcea Gold Mines

FIGURE 9.20 Grade control sample preparation procedure

The drawing in Figure 9.20 shows the process of analysis for the gold—which is the most important—and also for other elements like copper (Cu), arsenic (As), bismuth (Bi), and so forth.

From the results of the comparison tests, a standard process was defined for the gold assaying of 30-g samples. For high-grade gold samples and low continuity of the orebody, it was decided to perform two assays with two 30-g samples and calculate the average.

Data Interpretation

As mentioned before, all assay data is stored in a single database, which is managed using RecMin software. Every day, more than 3,500 data fields are input to the database and managed in a semi-automatic way by the grade control geologist, who also carries out the interpretation, the calculations, and the definition of the ore zones.

The definition of linking codes between the different data tables (topography, lithologies, assay data, borehole surveys, etc.) makes importing this information a simple and quick task. Graphic handling and interpolation are easily performed on the screen with color codes that simplify selecting the economic zones later and exporting the data to stake out positions in the pit.

The software program always works in three dimensions, which makes interpreting the information easier at all times.

Several methods were tested for defining ore zones. The simplest would be to draw lines to delimit the boreholes with grades above the cut-off grade and use those lines as ore–waste boundaries. In this case, ore grades would be calculated using the average grade value of the samples that fall within that drawn area. A more complicated method was attempted that involved doing a geostatistical grade estimation of block grades in a block model defined as $1 \times 1 \times 4$ m by applying a geostatistical interpolation method (Kriging) to the assay results, followed by the use of this block model to define the ore–waste boundaries.

Finally, interpolation by the inverse distance to a power was chosen. An ellipsoid of search was used to select which samples—from the bench being calculated and the bench below it—would be included in the interpolation.

The calculation parameters, like the power of the inverse distance and the size of the axes of the ellipsoid, are defined by means of geostatistical interpretation. These parameters are updated once or twice a year as more information becomes available. As an average, the power used at El Valle-Boinás was 3 and the distance from the main axes of the ellipsoid is 6 m for the principal axis, 3.5 m for the secondary axis, and 1.5 m for the third axis.

The spatial orientation of the ellipsoid of search is defined by the geologist on the basis of its position in previous benches and the dip of the orebody for each calculation area, given that it changes with the area that is being mined.

The definition of ore–waste boundaries is not as simple as drawing lines separating blocks according to a cut-off grade calculated from operating costs and metal prices. Other factors must be taken into account:

- There is an operational cut-off grade defining marginal ore that must be set aside for treatment at the plant when necessary. Given that a truck's load must be either taken to the waste dump or to the plant, the cut-off grade would be lower if just this cost of loading and transportation to the plant's stockpiles and the cost of treating it were considered. If the actual grade falls between both cut-off grades—the one that includes all the operation costs and the one just defined—this would be what is called marginal ore, and this must be separated and stored aside, because it contains enough gold to pay for the processing costs.
- In addition, the economic value of the ore is not solely from gold. The silver and the copper must be taken into account; for this reason, work is usually performed with a theoretical grade called *gold equivalent grade*. This grade is estimated as the grade of the principal element that alone would amount to the total net value of the block, taking into account the metal market value, the plant recovery, and the price paid by the smelter of the concentrate.
- The ore might also contain elements that, when present in the flotation concentrates above certain grades, are penalized by the smelter. This is the case for arsenic and bismuth. The grade control system makes it possible to prepare stockpiles with different concentrations of these elements so an "ideal blend" can be delivered to the plant to minimize penalizations. This is achieved, in some cases, by adding to the gold equivalent a negative gold value related to the grades of some of these elements. It could also happen that the hardness of the ore outlined by grade control is a problem for grinding in plant. In cases like this, it can be sent to a stockpile for future blending with softer ores to improve plant production.

- There are also some limits on the size of the ore zones. The minimum size is established by the grade control geologist, taking into consideration the size of the loader bucket, the direction of the load, and the geologist's experience in previous ore zones. Taking into account all this and the information available on the screen, the geologist adjusts the ore zone boundaries to a minimum in order to reduce dilution with the surrounding waste and, at the same time, achieve an acceptable loading performance.
- It is also important for the grade control geologist to have a precise idea of the value of each ore zone and to instruct the mining team accordingly. In some cases, there might be a small, very-high-grade ore zone (e.g., 400 g/t Au), which alone might be worth more than the rest of the ore in the bench. In cases like this, the grade control geologist must be able to convey to the mining team that the loading and haulage of the high-grade area must be done under the best possible conditions.

Summarizing all of these factors, the job of a grade control geologist is not limited to defining the ore–waste boundaries and where each ore zone is, but also to take into account other factors, like the ones described here. This makes experience a very important factor in the decision-making process.

Ore Loading

After the different ore types have been outlined and their destinations at certain stockpiles have been decided as described in the preceding paragraphs, the information is exported to the surveyors' team. These team members stake the ore zones on the ground and distribute the information to the operators and technical staff who are accountable for ore loading and haulage.

In addition to the usual operation tasks, special attention must be given to several other factors:

- Both the loader and the haulage truck units working with it must have a visible and easily identifiable flashing colored light, indicating whether a truck's load is intended as ore or as waste and the stockpile of destination. In the El Valle-Boinás mine, a red flashing light indicates that high-grade ore is being loaded and hauled to the plant's stockpiles, a green light indicates low-grade ore, and no light indicates waste. These security measures might not seem important, but they were decided upon after having experienced several cases where, perhaps because of fatigue, lack of information, or a desire to reduce truck cycle times, ore was dumped as waste, resulting in economic losses.
- Another way of controlling the movements of the ore would be to use a global positioning satellite dispatching system, linking trucks and loaders through a wireless network with the servers, so that information on the movements of the trucks, the type of ore, its destination, and so forth is available in real time. This system can also be complemented with the use of a warning signal for those cases in which the movement of the trucks and the truck dumping sites do not correspond to the type of mineral the truck carries.
- There must always be awareness of the need to handle zones of very-high-grade ore in a much more delicate manner, not only when loading and hauling the ore, but also when surveyors are staking high-grade ore zones, and when discharging the ore into the corresponding stockpiles.
- Careful monitoring and control of the elevation of the bench toe surfaces is critical. This can be done using topographic levels that are visible to loading operators, using automatic acoustic systems or manually. Toe elevation deviations of ± 20 cm could be acceptable, which for a bench height of 4 m is equivalent to $\pm 5\%$.



FIGURE 9.21 Typical ore zone definition and staking at El Valle-Boinás mine

The drawing in Figure 9.21 shows a typical example of the definition of ore areas for loading, marking in the pit, and the loading process.

Database and Report Generation

As previously mentioned, the block model is a database containing not only the information on the initial resource model but also all the data generated relating to interpolation, geology, type of mineral ore, calculations, destination, and so forth, which is entered as it becomes available.

All this information will allow for easy generation of all types of reports, including production, comparisons with expected outcomes, averages for certain periods, totals per mine, and so forth.

The statistical processing of all this information being generated allows for evaluation if the grade control system must be modified by comparing several scenarios or even conducting simulations.

Conclusions

This grade control system has evolved from the first year, adding improvements until the final state described in this case study was reached.

The main conclusion regarding grade control systems is that the concepts are far more important than the tools and the means. By this is meant that the mining concept of “what you want to achieve” is much more important than having sophisticated software systems without the concept. Attempts have been made to create very expensive custom-designed software applications for specific purposes that, in the end, were never used because they were too complex. Conversely, simple applications have been developed that were very good at accomplishing the functions for which they were created. In short, it is often difficult to explain to a software programmer what needs to be accomplished in mining with a given application because mining concepts are usually unfamiliar to them.

The development of this grade control system over these years has taught an important lesson: it is best to start with something simple that works before going on to more complex stages. Trying to implement a program with lots of objectives in a single step can easily turn into disaster.

A second important conclusion is that it is the actual mine operators and the users of software applications who must communicate what they need and determine whether an idea is feasible or not.

Perhaps the most important conclusion that can be drawn from all this testing is that money invested in grade control for this type of mineral deposit will make a profit through reduced dilution of the ore sent to the plant. Therefore, when increasing the amount of sampling and control, the limiting constraint is not cost but rather production cycle time, by increasing the time required to process grade control information and thus reducing production capacity.

Therefore, it is very important to have an automated grade control process, that is to say, to have all the information being generated processed by computer applications programmed with the logic of the process. For example, the atomic absorption machine, used for the last stage of gold assaying, initially operated independently from the rest of the lab equipment, so assay results were input manually and later were entered in spreadsheets and sent to the grade control staff. Today, when a package of samples arrives in the lab, these are entered in the database that links all the equipment using only the bar-code reader. Then the different machines, including the atomic absorption machine, link their results with that database through the interfaces in a way that eliminates the time needed for handwritten records and avoids the potential for input errors.

This example, just like others that have been mentioned, makes it possible to reduce the cycle drastically and to manage thousands of pieces of information efficiently and quickly every day.

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CASE STUDY: RELIABILITY ASSESSMENT OF A CONVEYING SYSTEM AT ATLANTIC COPPER*

This section describes a methodology for improving maintenance practices based on the application of reliability-centered maintenance (RCM) and mathematical modeling for the conveyor belt system at Atlantic Copper.

RCM is a systematic consideration of system functions, the way functions can fail, and a priority-based consideration of safety and economics that identifies applicable and effective maintenance practices. So, RCM allows the focus of the maintenance efforts to be on those functions whose priority, in terms of production, safety, and protection of the environment, are higher, leaving aside other maintenance tasks that are not strictly necessary. This will lead to an increase in the effectiveness of the maintenance and better management of safety and environmental risk, thereby reducing costs.

The other main feature of this section is how mathematical modeling is used to assist in optimizing the maintenance tasks considered in the RCM work. A mathematical model provides the decision-maker with a powerful tool for knowing where the optimum is located and how far the current maintenance practice is from that optimum point. Frequently, RCM is used only as a systematic procedure to filter out unprofitable maintenance practices. However, without the assistance of modeling, the engineer tends to make decisions according to his/her experience or

* This section was written by E. Crespo and M. Palacios.