CHAPTER 1.10

Geometallurgy

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TERMINOLOGY

Geometallurgical concepts are as old as the Bronze Age, but the term *geometallurgy* is one of the newest mining words. According to Jackson et al. (2011), the first mention of the term *geometallurgy* in literature was in 1968 by McQuiston and Bechaud. The oldest paper in the OneMine database to use the term geometallurgy (Martens et al.) is dated 1998. In the late 1990s, geometallurgy was adopted for general usage in the mining industry. Like most language additions, the term was used and understood within some localized mining communities, such as the Chilean copper industry, before it was adopted as a globally understood term.

Apparently, even in its earliest context, the term embraces geology, mining, and metallurgy (not simply geology and metallurgy as the name implies). The *geo* part of the term refers to the ore in the ground and the way that ore is removed from its natural surroundings so it can be delivered to the process plant. The *metallurgy* part refers to all the processes that are applied to the ore to generate a revenue stream for the organization.

Geometallurgy has been loosely applied as a general term to describe geological measurements that relate to metallurgical outcomes. In its simplest (and least successful) form, these measurements are ones that have been routinely performed by geologists and have simply been used to describe properties of geological ore types. This is a low-cost and hopeful geometallurgical approach.

Similarly, it is insufficient to have a process engineer dictate a new set of measurements to geologists and new planning methods to miners. Geometallurgy requires teamwork between the disciplines and a shared understanding of all aspects of the project. Rigorous implementation of geometallurgy in a project requires a detailed understanding of the project cost, risk factors, and revenue drivers and how the mining and processing influence these drivers. The following is a useful definition of *geometallurgy*: "Geometallurgy is a scientific discipline in which geological data, mining data, and processing data are co-analysed to generate useful information and knowledge to optimize resource profitability" (David 2014).

The important points in the definition are that all disciplines are involved and that the outcomes should be real in monetary terms, not simply nice-to-know information. When developed and implemented in an appropriate way, geometallurgy unlocks value in a resource that is kept hidden by the silo mentality that often exists in most modern operations. The knowledge silos of geology, mining, and metallurgy become interactive and inter-responsive, revealing limitations, realities, and opportunities, some of which have lain dormant for years or decades. For those who have successfully implemented geometallurgical principles at sites, geometallurgy is much more than something to add on to the business to help improve profitability—it is the heart of the business.

ARTISANAL MINING

A modern mining operation employs hundreds and often thousands of people, and they all carry out individual important functions. The result is that a resource that nature has provided is converted into a steady revenue (and sometimes profit) stream. The revenue stream supports all the employees, their families, the managing company, the shareholders of the company, the economy of the region in which it is located, and the various levels of government with responsibility for the locality. Mining projects are usually big and expensive, and it is almost impossible for individuals to grasp the complexity, let alone optimize it. It is much easier to describe a mining operation at the other end of the operations scale by examining the artisanal (often illegal) miner, working alone or in a small group, who can be thought of as the embodiment of geometallurgy.

It is useful to walk in the shoes of an artisanal miner to appreciate the breadth of realities that must be managed just to secure an ounce of gold. Imagine you have a small operation centered on a vein of gold-bearing sulfides that have supported you and your family for three years. However, the vein has split in two directions, and you need to make a choice. Going one way looks like the right direction for the highest grade, but the ground is bad. Your own son could become one of the next forgotten victims of the goldfield if you chase this vein. Going the other way, the ground looks good, but the grade is lower and the ore is harder. You also know that what you can see today does not hold any guarantees for tomorrow.

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You decide to get some help from the bad-ground specialist on the field (at the cost of a few ounces) while chasing the safe vein with your team in the short term.

Although you are poor, mining is in your blood, and you have been working this goldfield for a decade. You are rich in knowledge and hoping that life's lottery will favor you and your family one day soon. From knowing the vein to follow, the pain of seemingly endless time at the face, the dust and danger of hand drilling, the small-scale blasting you do every couple of days, hauling the ore out in a wheelbarrow, breaking the big lumps with a sledgehammer followed by hours of crushing in a mortar and pestle (that is big enough to hold 5 kg at a time)—only then is the ore ready for you to get the gold out. You have taught your daughter to manage the cyanide, zinc, and mercury, which is something you did for six years on another field before you were strong enough to work the ore. Sometimes the cyanide followed by zinc (or iron scrap) provides the majority of the gold, but sometimes it is the mercury when the gold is coarse.

You also know the financial pressure from the richest miner on the field—the one that supplies you with the explosives, cyanide, mercury, and even the new wheelbarrow trays you need every three months. You also know where the best and worst prices are to be found for the gold you extract. As an illegal miner, you can only sell on the black market at the prices they offer. All the time, there is the threat of prosecution or being shut down by the government, and it is necessary to keep on the good side of the local authorities.

To keep the gold revenue flowing, there is no real alternative for the artisanal miner who is fulfilling the roles of exploration geologist, production geologist, mining engineer, crushing and grinding supervisor, gold extraction supervisor, health and safety officer, process supervisor, purchasing officer, marketing manager, and government liaison officer. In short, the artisanal miner must be a geometallurgist (and more), and some of those who are less experienced and are working the ore in the team will also be learning essential geometallurgical skills on a daily basis.

GEOMETALLURGY IN EXPLORATION

The discovery of ore bodies is the job of exploration geologists, and it is an unfortunate reality that most exploration efforts end in failure. Only a few exploration targets progress to be considered as mining projects, and even fewer make the full journey to become producing mines.

The prospects that progress to the point of economic assessment have (by definition) many of the attributes that are essential for success. These include the most fundamental, such as deposit size, grade, location, and the ability to mine it economically. However, it is also necessary to be able to process the ore using proven methods, in an economic fashion, in an environmentally acceptable way and produce something a customer will want to buy. A prospect that advances to the point where diamond core drilling has been approved has already passed many geological hurdles but may have passed no metallurgical hurdles. Basic application of geometallurgy late in exploration is about placing metallurgical hurdles in front of the project at an early stage, and it is also about making sure that all ongoing geological data collection is relevant to potential metallurgical outcomes.

A potential consequence of early geometallurgical intervention is that exploration on a prospect falters, or even fails, because of a metallurgical hurdle. Typical metallurgical problems include low recovery values and the inability to generate a marketable product using conventional processing means. The hurdles are there to modify exploration and economic assessment behavior at the earliest possible time, because that is when the least money has been spent. The behavior change on that prospect could be as final as stopping the program and using exploration resources on a more attractive prospect. However, it could also be in the form of a change in the way drilling is targeted or modifications to the standard assay suite for the project core samples.

Nickel sulfide mineralization is particularly prone to the problem of lower than expected recovery. An exploration core that appears to have excellent grade, in terms of total nickel, may only have half of the nickel in a form that is recoverable by the separation process of choice—flotation. The unrecoverable nickel is bound up at very low grades within some of the silicate minerals. Early metallurgical intervention quickly identifies this relatively common problem and results in the introduction of appropriate assaying techniques that distinguish sulfide nickel from silicate nickel. The prospect grade assessment basis is then changed to sulfide nickel (rather than total nickel) and a realistic reassessment of economic prospects for the mineralization is performed. Not only are decisions being made on the appropriate basis, any future routine drilling on the prospect will be assessed correctly.

Some copper sulfide mineralization is unable to generate a saleable product. One example is where a deposit shows good copper grades but has unusually high arsenic levels. The majority of the copper and arsenic report to flotation concentrates in early testing, and all efforts to reject arsenic also reject large quantities of copper. The problem is almost certainly that a large amount of the copper is contained in the mineral enargite with the formula Cu_3AsS_4. Copper and arsenic are chemically bound together in a mineral that floats as readily as chalcopyrite or bornite. Enargite is rich in copper (almost 50% Cu), but it is also 20% arsenic. It may be impossible to generate a copper concentrate with less than 5% arsenic, which is a contamination level that prevents the concentrate from being shipped to many countries, including China, and a level that is unacceptable to copper smelters. A metallurgical solution to a problem such as this would be expensive on-site concentrate processing using chemical (rather than physical) separation to make a saleable product. Immediately, the problem of enargite is recognized, and the economic and environmental assessment criteria for the prospect change significantly.

These two examples are somewhat uncommon, but both have been encountered on many occasions in the author's own experience. A more common occurrence is early or late identification of the refractory nature of a gold deposit. Early identification through appropriate metallurgical intervention allows geologists to set up a routine assay suite suited to potential processing options. Late identification occurs when scoping or prefeasibility study (PFS) testing identifies the refractory problem, but the only assays performed on the entire drill library are gold and (maybe) silver. For a refractory gold ore, the most important assay is sulfur. If core or drill chips have oxidized in storage, they will not be suitable for determining sulfur content retrospectively. (To assess oxidation, photos of the core when drilled can be compared with the core currently in the trays.) The only solution is to drill new core. If the core has not oxidized, then all mineralized core intervals need to be resampled and assayed for sulfur, which is achievable but also expensive and time consuming.

Other metallurgical problems best identified at the earliest possible time include extremely fine-grained mineralization (below 10–20 µm) that is difficult to liberate; high levels of manganese in sphalerite, which could make an unsaleable zinc concentrate; or extreme comminution properties.

The ideal process engineer to assist an exploration team is someone experienced in sampling and testing, comminution and physical separations, and with a comprehensive knowledge of flow sheet development techniques for the commodity in question. In ores where it is obvious that the flow sheet will be complex (such as nickel laterites or copper leaching), expertise in both the physical (ore handling, comminution) and the chemical (hydrometallurgy) is needed at an early stage, and this usually requires two individual experts.

With the increasingly proscriptive public reporting requirements, such as National Instrument NI-43-101 in Canada and the Joint Ore Reserves Committee (JORC 2012) in Australia, an early metallurgical intervention can also avoid the release of false information that may need to be retracted in a later release and may even leave the owner and report signatories open to legal proceedings.

It is preferable that problems with projects be identified (and acted on) as early as possible. Once the author asked an experienced metallurgist what he would do with the difficult ore body being worked on. The learned response was "find another ore body." Time, and a lot of misspent company money, proved him right.

GEOMETALLURGY IN PROJECT DEVELOPMENT

Geometallurgy is best applied during project development by understanding its ultimate purpose and source of value in the project. In the description that follows, the time progression of the project development is reversed so that the end point is understood first and the steps taken to get to that point are developed in that context. The steps are outlined in Figure 1.

Operational Phase

In a geometallurgically functioning operating environment, the mine site will seamlessly speak the same language: from the geologist carrying out grade control activities, to the short-term mine planners devising this week's blending tactics, to the process superintendent needing to get an extra 50 t/h

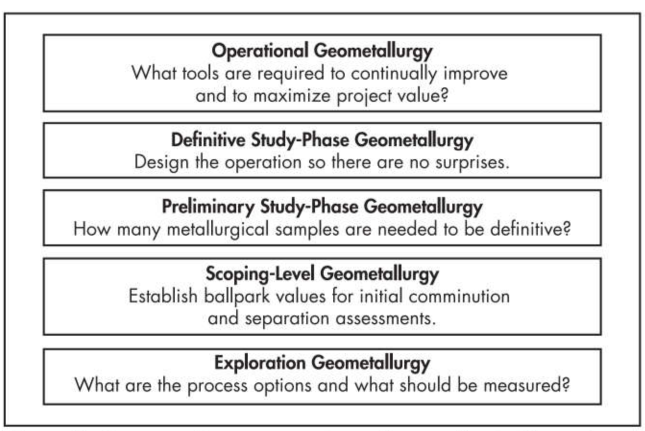


Figure 1 Geometallurgy steps through a project's life cycle

(metric tons per hour) from the plant at the end of the month. For this to be a reality, the geologists need to have ore type definitions that feed through to the mine planning process, and the mine planning outputs must indicate key factors, such as likely throughput rates, recoveries, and product qualities for the mix of ores. Finally, the process engineers must have a reasonable level of faith that the process information in the mine plan is reliable.

This sort of environment may exist on many sites worldwide, but it is certainly not normal practice. More typically, the process engineer, wanting more throughput, would ask the geologist if there was any soft ore about. The geologist would consider soft ore to be a particular geological type that has been globally assigned a single work index and would accordingly identify where such ore was in the pit. When the mining engineer delivers it to the plant, the throughput mysteriously drops by 50 t/h because the ore is hard. The *geological* ore type was not a *geometallurgical* ore type and, consequently, the mine planning information about ore from that particular location was unreliable.

The latter case is encountered on the majority of sites and is often characterized by poor relationships between the disciplines. The geometallurgically driven site is rare with one of the best published examples being the work done at Batu Hijau in Indonesia (Wirfiyata and McCaffery 2011). Other published examples of sites that are working geometallurgically are Cerro Corona in Peru (Baumgartner et al. 2016), Quebrada Blanca in Chile (Chait and Schiller 2016), and Cripple Creek and Victor gold mine in Colorado, United States (Leichliter and Larson 2013).

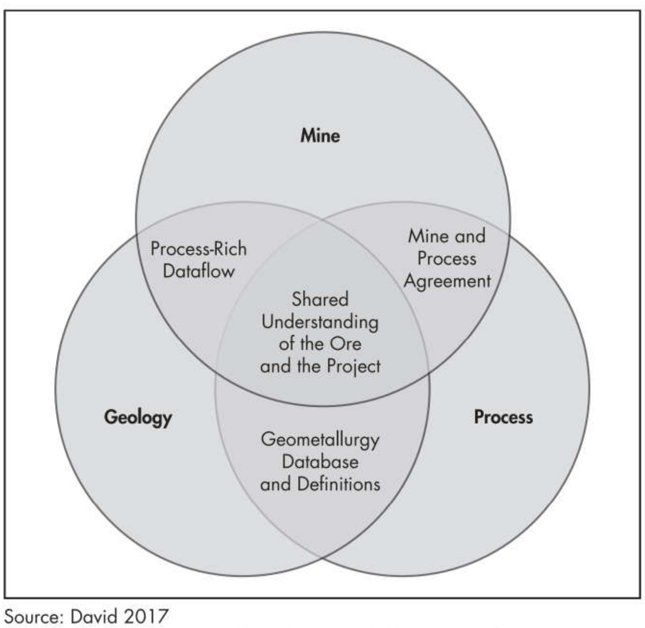
To underpin a successful operational phase, the following geometallurgical tools need to be available:

1. There need to be established and reliable relationships between easily obtainable measures (from tests conducted on the ore) and the metallurgical performance of the plant. This can be as simple as a recovery relationship based on head grade or more complex, such as a concentrate grade prediction based on mineralogical mixture. In the comminution area, this could be semiautogenous grinding (SAG) throughput predicted using a simple measure of ore competence or ball mill performance based on a grinding work index. All such relationships are recorded, along with their proofs, in a single document (typically a spreadsheet) termed the *mine and process agreement document* and owned by the site process engineers and mining engineers.
2. Measurements on the ore to develop the preceding relationships must have been performed in the past (during exploration and infill drilling) and must be continually conducted in a program of testing upcoming ore (using samples from grade control drilling) in time to be useful for production planning.
3. A mechanism is in place to enter this metallurgical information into the geological database, including assignment of the ore to a particular geometallurgical domain.
4. The geological model recognizes geometallurgical domains, and these are likely to be different than geological domains.
5. The geometallurgical domains and the metallurgical properties are carried electronically from the geological database through to the mine planning procedures, providing properties for ore blocks.

6. The geometallurgical domains, metallurgical properties, and metallurgical predictions are a standard component of mine planning reports.
7. An ongoing program of geometallurgical verification, reconciliation, and improvement is in place.

Putting such a system in place after commencement of operations is expensive and is not trivial. First, dedicated personnel who intimately understand the operation, but who are removed from day-to-day operational responsibility, need to analyze historical data to show that relationships exist between easily measured parameters and plant performance. Second, a clear economic case needs to be developed for implementing an extensive ore measurement system and linking this to mine planning. Third, the geometallurgical system needs to be institutionalized as standard practice at the site (parallel to existing grade control), rather than being an occasional measurement and adjustment system. Not surprisingly, the sites where implementation after start-up has been successful have required major changes from the status quo, accompanied by site-wide cultural changes. The preferred pathway to achieving an effective operational geometallurgical mindset is through developing a geometallurgical focus in the early project phases, commencing with the first geologists, miners, and process engineers to work on the resource at about the time where the discovery is considered to have potential. From that point onward, a model, such as the one shown in Figure 2, can be used to keep the project geometallurgically on track.

Recently, it has been shown (in confidential studies by the author) that geometallurgical prediction based only on data collected during the definitive design phase can provide useful, but limited, operational guidance. A metallurgical data set of about 200 samples measured for multiple comminution properties has provided useful plant throughput predictions. These predictions are well correlated (±10%) when the prediction time frame ranges from two to four weeks. On a daily or shift basis, the scatter in the actual-versus-predicted throughput graph is large (equivalent to the full span of throughput rates experienced by the plant), and these predictions should be ignored.



Source: David 2017

Figure 2 Simple operational geometallurgy model

If operational metallurgical predictions are required for periods shorter than two weeks, then there are two choices. The first is to test many (thousands) spatially distributed samples during the design period, similar to the program instituted at Olympic Dam in South Australia (Liebezeit et al. 2016; Liebezeit 2011; Turner et al. 2013). The second is to test hundreds of samples so that medium-term planning is geometallurgically informed and then institute an ongoing geometallurgical testing and verification program.

Definitive Phase

The preliminary phases must set the scene for the definitive phase. The definitive project phase is the last phase before a project moves into implementation and is also called the *feasibility phase* (JORC 2012). Typically, a single option for taking the project to implementation is designed and costed out to a level of accuracy suitable for company boards to approve implementation, for banks to make decisions on lending, and for equity partners to evaluate the investment potential. By referring to this stage as *definitive*, there is much more than a suggestion that the project, as presented, will achieve the stated outcomes. From a process perspective, definitive means that the operation will be implemented with the projected capital expenditure level; it will achieve the nameplate annual throughput; and it will achieve the design metallurgical outcomes at the estimated operating cost.

Process designs that fail by a margin of more than about 5% to achieve stated physical performance have damaging business outcomes and often fail financially (unless the metal price gods smile at the ideal moment).

The key question arising from this discussion is "How much metallurgical test work is necessary to make a design definitive?" As with all things metallurgical (and indeed, all things connected to the development of mineral projects), the answer is "It depends!" The factors it depends on include resource size, number of geometallurgical domains, variability of properties within the geometallurgical domains, flow sheet complexity, and product-quality requirements.

Many owners (especially those in a hurry) present a case for conducting a feasibility study (FS) to an engineering group without conducting the preliminary work (i.e., a PFS), and while an active exploration campaign is ongoing on the resource, they wish to evaluate definitively. This situation is a massive contradiction, and it fails the tests that will be imposed in attempting to meet reporting codes such as NI 43-101 or JORC (2012). Invariably, the exploration work will expand the resource during the term of the FS, and the owner will demand the FS now be based on the new larger resource. However, the inputs to an FS need to be frozen at the 10%–15% elapsed time mark if it is to be completed in the agreed schedule. Of course, it is possible to allow for a full FS rework at the 50% effort mark, but the risk is high that much of the work done up to that point will be wasted if there is a relatively simple change to the ore delivery schedule in terms of annual ore tonnage, feed grade, ore type, or key ore properties such as hardness. It is much better for the owner to be improving mineralization status during an FS with infill drilling than expanding the resource and changing the FS fundamentals. In the worst-case scenario, the owner discovers a new nearby deposit during the FS and switches to it as the only ore source on the assumption that it has similar properties to the one the FS is based on. This is an actual case example in which all geometallurgically based risk reduction was lost, the ore was not the same, and the project quickly failed.

The main message is that once a decision is taken to go to the FS stage, then expansive exploration activity should cease and work should shift to infill drilling with a target of finalizing mine planning and process selection at the 10%–15% mark in the FS. The FS should be allowed to run its course and either be rejected as uneconomic or accepted based on the physical limits imposed on the mine and the test sample selection before FS commencement. If exploration is ongoing and identifies a better prospect during the FS then there are two paths: either cancel the FS and return to the PFS on the new prospect or continue the FS on the existing prospect and commence a separate PFS on the new discovery.

Fundamental to the concept of being definitive in design is understanding the inherent variability in the most important (and most variable) process input—the ore. Given the importance of this topic, it is dealt with in some detail in the following sections.

Ore Variability

A common statement heard from less-enlightened geologists, miners, and process engineers is that ore type X is all the same and we have tested it. Further investigation usually reveals that ore type X is dominated by one lithology and, in that limited sense, it is all the same. A few tests may have been conducted on this lithology and the answers came out close to each other. Consequently, the average property measurement has been assigned to that particular geological ore type. This is not an adequate basis for assigning a design point to an ore type, and not only because the average is an inappropriate design point. For an ore type to be metallurgically confirmed as being "all the same," an adequate number of tests need to be conducted on spatially disparate examples of the one lithology to determine a reliable standard deviation for the property in question. If there are no metallurgical design implications arising from the plant being fed ore at the low end of the property spectrum (average minus two standard deviations) or the high end of the spectrum (average plus two standard deviations), only then can the ore be considered all the same with respect to this property. A common example of this is specific gravity (SG), which has a very low relative standard deviation in many (but certainly not all) ore bodies. The band of values encompassed by two standard deviations either side of the mean accounts for 95% of the expected values (for a statistically normally distributed property). Many ore properties, especially within a lithology, are either normally distributed, log normally distributed, or close enough to these distribution types. Provided enough samples are tested, it is possible to determine the standard deviation of the measure in question.

Therefore, the starting point for having a definitive design basis is understanding ore-body variability. Unfortunately for the impatient, this is a two-step process. The first step is estimating the inherent degree of variability by testing an adequate number of samples. The second step is testing adequate samples to confidently define the variability envelope for design purposes. These steps sound almost the same but the difference between them is crucial to this discussion.

Coefficient of Variance Concept

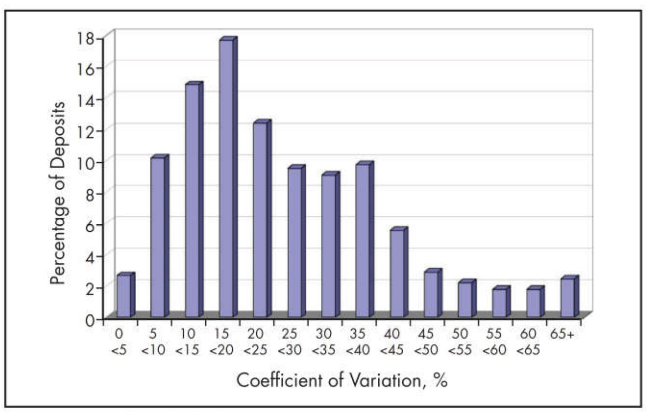
The principles of understanding variability in respect to comminution properties have been outlined by Morrell (2011), and this section will follow Morrell's conventions. First, the concept of coefficient of variance (COV) must be understood. This is sometimes called the *relative standard deviation*. COV is simply the standard deviation of a set of values divided by the mean of the same set of values and expressed as a percentage. For example, a data set with a mean of 25 and a standard deviation (SD) of 5 has a COV of 20%. For such a distribution, the band containing 95% of the measurements (±2 SDs from the mean) spans from a low value of 15 to a high value of 35. This means the maximum expected value (35) is much more than twice the expected minimum (15). If this is a critical design parameter (such as a grinding work index), then this high level of variability must somehow be handled by the process design or mitigated by an ore blending strategy.

A more benign case might be a siliceous ore with a mean SG of 2.5 and an SD of 0.05, giving a COV of 2%. The 95% band of SG values will be from 2.4 to 2.6. Most plant designs that are required to treat such an ore could be safely based on the average 2.5 SG without allowances for variability.

Morrell has examined the distribution of COV for the ore competence measurement (drop weight index [DWI] in kilowatt hours per cubic meter) of 650 ore bodies within his database of SAG mill competency (SMC) test results. The relationship between the impact energy imparted to a particle and the degree of breakage achieved in that particle is used to calculate the DWI. The results are useful in designing crushing and grinding circuits, and the test has been designed to be applied in geometallurgical programs. The graph showing the distribution of variability outcomes across the 650 ore bodies is reproduced in Figure 3.

Only 12% of the measured ore bodies have COV values of 10% or less, and half of the ore bodies tested have COV values greater than 26%. Competence is one basis for estimation of a SAG or autogenous grinding mill throughput capacity because, simplistically, the milling power requirement per metric ton is directly proportional to DWI.

At the median COV of 26%, the spread of DWI values expected is 100% of the average value (52%, or 2 × SDs below the average and 52% above). For example, an ore body with an average DWI of 5.0 kW·h/m^3 and an SD of 1.3 kW·h/m^3 (COV = 26%) will have ores with competence values ranging from 2.4 to 7.6 kW·h/m^3. The upper DWI value is more than



Adapted from Morrell 2011

Figure 3 Distribution of ore-body COV values for competence measurements

three times the lower value, and this is a range for which it is difficult to design a milling circuit without allowing for some blending. Note, however, that many ore bodies have well above average COV values and present even greater design challenges.

Coefficient of Variance Estimation

Asking how much information is needed to estimate the COV is the next obvious question. The answer is somewhere between 7 and 20 samples. The more samples, the better the estimate of COV will be. However, as COV should be estimated in the PFS stage, it will be covered in the "Preliminary Phases" section.

Assuming the COV of the measurement is approximately known (because a good PFS test program was conducted), it is then necessary to test enough samples to adequately define the distribution. The larger the COV, the more samples needed to achieve an acceptable level of confidence that the distribution average, and its limits, are understood. Of course, it is possible to design with a lower level of confidence, but this demands either acceptance of risk that the project will not perform or the inclusion of educated levels of overdesign to compensate.

Confidence Level in the Results

To assess the raw test data for variability, it is sensible to determine the average together with the SD and then calculate the COV. This is a simple matter in a spreadsheet software application, such as Microsoft Excel, which provides standard functions for these purposes. It is statistically correct to say that the more samples of ore that are tested, the more meaningful the statistics will be for the property being measured. An alternative expression for meaningful is *confidence*.

Mathematically (also available as an Excel function), the confidence level to which the mean or average is known can be calculated for any data set. A confidence calculation returns an absolute number (z), which is then used to define a range within which the true average should lie (measured average ± z). This range is the *confidence interval* (CI), and its very existence attests to the fact that any average from test work is only ever an estimate of an ore property.

One of the inputs to the calculation is the alpha value (α), and this is directly related to the confidence level to which you need to know the average. An α of 0.05, for example, is equivalent to a confidence level of 95%. At 95% confidence level, the answer you are looking for can be relied on to lie within the CI 19 times out of 20. At 99% confidence level, the certainty is 99 times out of 100 that the answer will be within the CI. To achieve this higher level of certainty (99% confidence), the interval calculation results in a much wider range within which the average value could reside.

How does the confidence level concept relate to the real world? When samples are chosen for metallurgical testing, they are usually selected from available core. If there is 1,000 kg of core available to sample, and each sample needs to be 25 kg, then (simplistically) there are 40 possible samples available for testing. Money has only been allocated for 10 tests, so only 10 samples are chosen, somewhat at random. Tests are performed and an average property value is calculated. However, if a different set of 10 samples was randomly chosen and tested, then the average value would have been different, but the method of arriving at the average has not changed. Calculating the CI for the mean using the 10 chosen tested samples provides the designer with a measure of what could have happened if a different random set of 10 samples had been chosen from the same batch of core for testing. For example, if the calculated CI ranges for the Bond ball mill work index (BWI) ranges from 8–12 kW·h/t, then the average value is not very well known at all, and the design basis (whatever it is) will carry a lot of risk. Had a different set of 10 samples been chosen for testing, then the average might not have been 10 kW·h/t but could easily have been as low as 8 kW·h/t or as high as 12 kW·h/t. However, if the CI ranges from 9.5 to 10.5 kW·h/t then the average of 10 kW·h/t is well known. It is highly likely that a similar average would have been determined for any other set of 10 test samples, so the design basis risk is low.

Apart from the confidence level being sought (α of 90%, 95%, etc.), the other two properties used in calculating the CI are the number of samples tested (n) and the SD of the data set (mathematically, CI = $f\,[\alpha$, n, SD]). Once sufficient samples have been measured to arrive at a credible SD, there is little change to the SD value as more samples are tested. Therefore, the only method of improving confidence in the mean (making the CI narrower) is to test more samples.

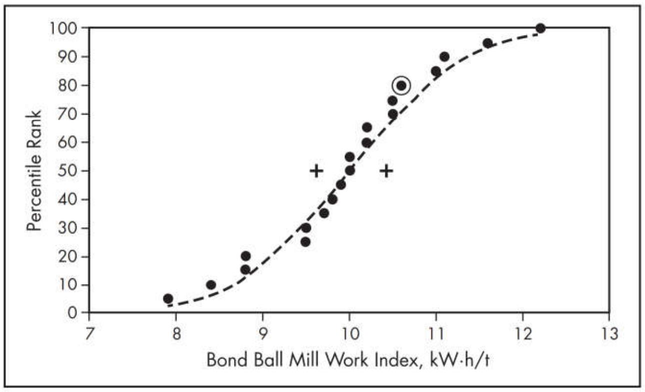
The concept of confidence must be understood by process designers because it is fundamental to knowing if sufficient tests have been performed to size equipment. The confidence a designer has in a design point translates directly to the confidence the designer has in the equipment selected.

Being Definitive

Being definitive in design means that the resulting plant will achieve the intended performance with the assumed inputs, and within a stated tolerance of accuracy. It is sensible, then, to adopt a conservative design position once the data has been analyzed. Conservatism is normally achieved by selecting a value that is some known distance above the average but less than the maximum measured value. A typical approach is to use the 80th percentile value, but an alternative valid approach would be to use the upper confidence limit in combination with a second factor (e.g., the SD). In general, design factors and operating margins are also added to the calculated process requirement. One clear warning that insufficient samples have been tested is when the upper confidence level (UCL) of the mean is greater than the 80th percentile value in the data set. This is a red flag, and it implies that if another set of samples is chosen and randomly tested, then the average of that set could actually be greater than the so-called safe design point, being the 80th percentile value. The obvious solution is to randomly select enough additional test samples to at least double the data-set size and ensure that the new UCL is significantly less than the 80th percentile value.

An example data set is shown in Figure 4 with the CI for the mean shown as crosses on the 50th percentile rank line. The 80th percentile point of the data set (10.6 kW·h/t) is circled and is clearly close to (i.e., in conflict with) the largest potential value for the mean of the data (10.4 kW·h/t).

An alternative approach for selecting the design point from the data set in Figure 4 is to adopt the 80th percentile of the fitted normal distribution (the dashed line) rather than the 80th percentile data point. The normal distribution line places the 80th percentile value (10.9 kW·h/t) more than twice as far above the measured mean (10 kW·h/t) as the UCL (10.4 kW·h/t), and this would be considered a safe design selection. Testing more samples is also an option, but it is not warranted in this particular case.



Source: David 2015, reprinted with permission from the Australasian Institute of Mining and Metallurgy

Figure 4 Close proximity of average value and 80th percentile value

The ultimate message from this statistical discussion is that enough samples need to be measured to provide confidence that the average value is known and, at the same time, provide confidence that the design position is demonstrably conservative.

Measuring Variability

Given the correct tools, it is a relatively simple matter to statistically analyze a set of numbers. However, it is essential that the numbers being analyzed are the right numbers. In designing a plant, it is essential that ore variability is understood because this drives maximum equipment capacities and minimum flow requirements. All operating metallurgists will know that their plant will become throughput limited when a particular ore property (e.g., grindability, competence, or grade) approaches an extreme. They also know that this type of limitation can last from a few minutes to months. Therefore, what is a sensible time frame for understanding ore variability? Typical starting points are the operating day or the operating shift, because this is the time frame over which operational control is exercised.

An immediate implication of this typical starting point is that mine plans prepared on a monthly or annual basis do not contain variability information immediately relevant to design calculations. They effectively contain grouped average data that significantly understates variability. By extension, a more inappropriate basis for design is the use of life-of-mine ore-body average properties.

By definition then, the analysis of variability can be considered successful if the method used to estimate variability generates answers that approximate daily or shift fluctuations. To achieve such an answer, it is necessary to examine the sampling basis, the mining method, and the available short-term data.

The short-term data available for analysis refers to information available in equivalent daily (or smaller) parcels. The two data sets representing the shortest time frames on an operating mine site are usually the geological drill database and the mining block model. The drill database typically contains short interval (1–5 m) chemical analysis data together with longer interval geotechnical, lithological, mineralogical, and alteration data. A single 1-m interval will typically weigh 5–10 kg and will provide a representative analysis of only a fraction of a day's production, generally measured in minutes. A single metallurgical variability sample may represent 5–10 m of core, weigh up to 60 kg, and represent up to 30 minutes of production. Although the blocks in a block model do not represent physical samples, they are mathematically generated from the drill database (e.g., using kriging). Given sufficient raw data, block properties are probably the most useful data set available for design purposes. Typically, the ore represented by a mine block will represent between a few hours and a day of production. In a geometallurgical program, variability testing is performed on core samples with the aim of embedding useful metallurgical information into the ore blocks.

Metallurgical samples are usually selected from the geological core set and, if the ore-body geometry permits, represent single geometallurgical domains. To get the required mass for comminution and separation testing, it is common to take contiguous samples over lengths of 10–50 m, depending on the core diameter and how much of the core the geologists are able to release for testing. A single metallurgical sample can intercept anywhere between one and five mine plan ore blocks depending on the length of core selected. The variability of assays of metallurgical samples is typically between the variability present in the drill sample assays and the variability present in the block model.

To illustrate how variability varies with time frame, an analysis has been conducted on assays for a copper and magnetite deposit where drill data is available as is about six years of mine block model data. The mine block data set is arranged in a typical mining sequence and has been subjected to mine planning grade control practices in forming that arrangement. This allows the block data set to be subgrouped into shift, day, week, month, and annual data sets, all of which can be analyzed for variability. All intervals under 0.1% Cu (the cutoff grade) have been excluded from the drill data set and the block model data set.

The average, 80th percentile values, and the upper and lower 95th percentile bands were determined for each data set. A comparison is made of the variability levels within the data sets in Figure 5.

Variability within the "Year" set is very low and far less than the variability expected on a shift basis. This shows the

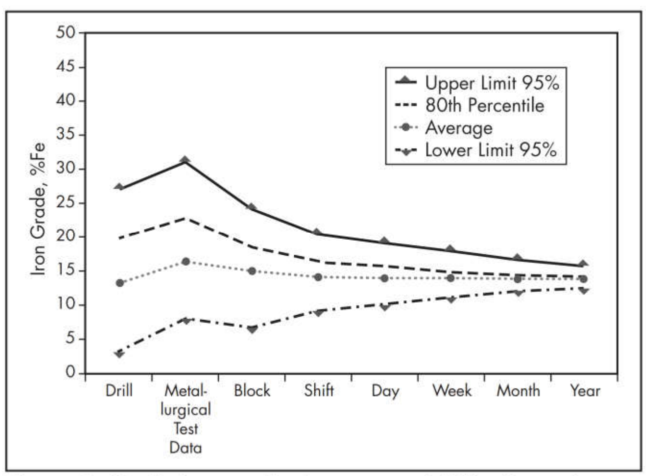


Figure 5 Single iron ore-body assay variability for various data sets

folly of using annual plan data to select a design point for any property. On an annual basis, the 80th percentile and the average are almost the same number (14% Fe). At the preferred design time frame (the shift data set), the 80th percentile value is much higher at 16% Fe while the average is effectively unchanged (as it should be) at 14% Fe. Figure 5 also demonstrates the overestimation, compared to shift data, of variability inherent in drill assay data (1-m interval basis) and in the metallurgical test data set. The average value for the metallurgical test data set is offset from the general average value because of the relatively small number of samples.

Out of 365 available days per year, the feed is expected to be within the 95% band in Figure 5 for 347 days. This means that there are nine days per year where the feed assay is expected to be greater than the upper limit (>20% Fe) and nine days where it is expected to be below the lower limit (<9% Fe). On a three-shift-per-day basis, this translates to 27 shifts of very high grades and 27 shifts of very low grades a year, about one of either instance per week. If the upper and lower feed grade expectations are set using yearly data, then for about 50% of the time, the feed % Fe will be higher or lower than these annual extreme limits suggest. Unless realistic expectations of shift operational variability are established by process engineers during design and operation, regular conflict can be expected between the process engineers, geologists, and mine planners on almost a weekly basis.

There is a valid alternative to accepting the shift variability outcomes as shown in the Figure 5 analysis. The alternative is to rerun the mine planning model with tighter limits on grade control. The disparity between the monthly variability and the shift variability will remain, but the "Upper Limit 95%" shift value may be reduced from 1% to 0.9% Cu or even 0.8% Cu, and this would lead to a design capacity reduction in the concentrate handling sections of the plant. Tighter grade control results in reduced plant capital expenditure but has a cost in terms of mining capital expenditure and operating cost (such as increased rehandling of ore and the possible use of smaller equipment). For each project, the design team needs to balance grade control cost, capital reductions, and plant operational flexibility. For demonstration purposes, this analysis has been limited to the iron feed grade. Similar consideration is necessary for each critical design input including, for example, comminution properties; impurity assays; acid consumption values for leach design; and concentrate mass yields for flotation, gravity, or magnetic separation circuits.

Importance of the Definitive Phase

It must be remembered that the definitive phase study (FS) design outcomes have the inherent potential to become an implemented project, often with associated process performance warranties for the designer and vendors. If the critical variability issues have not been measured adequately, then the implemented outcome will suffer financially. In many of these cases, the project may never achieve nameplate design, and some of these projects will fail to perform to the point where closure or major capital expenditure are the only options.

Although not wise, it is possible to complete a study with inadequate variability measurements, but such a study is far from definitive. A designer in this position must incorporate a prudent design contingency (i.e., select larger equipment) and argue the reason for it statistically, using experience and through appropriate benchmarking. A low level of confidence in a key design value has a real cost as higher capital expenditure.

It is only possible to achieve optimal definitive stage outcomes in a timely fashion if the preceding phases have been performed adequately. This includes thinking about how many individual test samples may be required for the definitive stage test program. This, in turn, influences the type of drilling conducted, the diameter of core being extracted, and even the way the drill samples are stored after extraction.

Evaluating the Adequacy of Outcomes

When evaluating definitive phase (alternatively bankable, or FS phase) outcomes, the key word is *definitive*. Are there undefined elements, such as the true variability of critical measures, which are missing from the evaluation and are not reflected in the design outcomes? All raw test data pertaining to these critical measures should be sighted, and both the adequacy of the raw data set and the logic behind the selection of subsequent design points must be assessed.

The following are key questions:

- Have enough samples been tested to establish a reliable value for the variability within the definitive ore body (as contained in the definitive pit shell or underground workings) and within major ore types?
- Do the sets of ore property data confirm that the various ore types have geometallurgical meaning?
- Do all predictive relationships (linking geology measures and process outcomes such as throughput or recovery) have strong enough correlations to be useful?
- Has the ore on which the plant will most likely be commissioned been evaluated as a set of separate samples?
- Has the ore that will be treated within the period in which the capital is repaid been evaluated as a set of identifiable samples?
- Are there any key process design aspects not supported by adequate test work?
- If a choice has been made to overdesign, rather than conduct definitive testing, this must be documented and be reflected in the equipment selections.

It is recommended that all samples to be tested are identified, collected, and in process before any definitive phase engineering commences. Testing at the same time as definitive engineering is proceeding is virtually guaranteed to negatively impact the definitive study schedule or the study quality. This often means that a definitive testing subphase should exist between the preliminary and definitive study phases.

Preliminary Phases

The preliminary project phases must set the scene for the definitive phase and, eventually, operations. Quantification of variability in the earlier phases allows informed decision-making in relation to the ultimate numbers of samples to be tested.

Variability Estimates for Each Critical Measure

A typical aim of a PFS phase is to evaluate many potential project development alternatives and select a best case to take forward to the definitive stage. A typical costing accuracy target at PFS is ±25%. To ensure that a believable comparison is achieved between options, it is necessary to have a relatively good understanding of the variability of the critical ore properties so that comparable design points can be selected for each option.

In some PFS comparisons, the key design decisions for the options being compared rely on totally different measures. An example of this for gold processing is comparing a flotation-based circuit with a whole ore leaching-based circuit. The flotation-based circuit needs to be understood with respect to concentrate grade, mineralogy, and recovery by flotation. The leaching circuit needs to be understood with respect to leach recovery, reagent consumption, and leach residence times (among others). When comparing these two options, it is tempting to take shortcuts, such as using the average values (rather than design values) when selecting equipment. This approach is likely to be misleading if the key parameters in each option have significantly different levels of inherent variability.

It is recommended that the prefeasibility test work aims to establish the variability levels in all critical design inputs (e.g., those that leverage more than 5% of the final capital or operating cost) by major ore type. In this way, the results provide a level of accuracy appropriate for prefeasibility and also guidance as to how much more testing is needed before a definitive study can commence.

It is normally recommended that general flow sheet development (such as establishing the flow sheet steps and the grind sizes for each stage) be conducted on composite samples. Once a reliable flow sheet has been developed, it is possible to test variability in a sensible way. For variability estimation, the recommended number of tests (as indicated previously) is between 10 and 20. For cost control, and to limit consumption of valuable core, select 10 variability samples and test each sample for every critical measure. Continuing the example of the gold ore where direct leaching is to be compared with pyrite flotation, the program for each major ore type might look something like the following:

- 10× SMC Tests for SAG sizing, general comminution circuit sizing, and comminution power consumption estimation
- 10× BWI tests for ball mill sizing and power estimation
- 10× Bond abrasion index tests to measure ore abrasivity and estimate consumptions of grinding media and mill liners
- A sensibly weighted composite of the 10 samples to develop a flotation flow sheet and develop leach conditions, including establishing separate grind sizes for optimal liberation
- 10× variability flotation tests to estimate all major flotation design parameters
- 10× variability leaches to estimate all major leach design parameters

When these results are available, it will be possible to compare the two circuit types at the required level of costing accuracy, and it will be possible to estimate how many more tests of each type are required before design can be elevated to the definitive level. The number of additional tests will vary from measure to measure. For example, it is possible that 40 additional SMC Tests may be necessary although only 10 more BWI tests are required to achieve acceptable accuracy and confidence.

Sampling needs for the definitive stage must be identified early in the PFS program so that any additional drilling or sample extraction can be conducted without delaying the project schedule. It is also essential that the sample selection locations be matched to likely mining areas. Sample location selection should be left until a mine plan is available that approximates the plan that will form the basis of the FS.

Essential Tests

A precursor to the preliminary test program is identifying what tests must be performed. The list of tests needed is a direct consequence of the various flow sheets and ore types being considered. Each flow sheet that is to be evaluated needs to be identified, and the types of equipment or processes being considered need to be listed.

The test type has geological implications because many tests require particle sizes that can only be delivered by core that is PQ size or larger. In the special case of pilot testing of fully autogenous milling, it is essential that either broken ore up to 200-mm topsize or large-diameter diamond core (PQ 83 mm and larger) is available.

If it is necessary to compare two technologies in the PFS (e.g., semiautogenous milling compared with crushing and ball milling), then tests appropriate to each technology must be performed for the comparison to be fair. It is not adequate to use Bond measurements for the ball mill case, for example, and then infer SAG performance from Bond measurements to provide the basis for the SAG design. In some cases, using the Bond inference works, but if the ore is high competence (DWI >8 or A × b <35), the Bond methods will typically underestimate SAG power and overall power for the case, giving a false comparison with high risk inherent in the SAG option alone.

Best Sample Size

Once the tests are known, then sample masses for conducting each of the tests can be obtained from the relevant laboratories. The overall sample mass requirement to conduct all tests, and prepare composites, is then used to determine how long the single sample contiguous core interval must be. The length will depend on the core diameter and how much is available for metallurgical testing (whole, half, or quarter core are the usual options). In virtually all instances, reverse circulation chips are unsuitable for metallurgical testing, especially comminution testing.

If the drill run intersections in the ore zone are shorter than the required length, then multiple holes at the same location (where the mineralization is shallow) or multiple wedge holes from the one parent hole (for deeper drilling) may be needed to provide suitable samples comparable to contiguous core. In all instances, the selection of metallurgical test samples must be a collaborative effort between the geologists and process engineers.

Optimum Testing Schedule

In the ideal world, PFS metallurgical test work should be completed before any PFS engineering design commences. This is relatively easy to arrange for standard comminution tests but much more difficult for concentrator flow sheet development work. If test work is ongoing while a plant section is being designed (using scoping design parameters), then there is a high risk that the results will change the flow sheet, the mass balance, and then the equipment selections. If any level of meaningful mechanical, civil, or electrical design or cost estimation work has been carried out based on the old parameters, it will all be wasted and have to be redone with the new inputs. When the test work cannot be completed in the suggested sequence and time period, a risk review must be conducted, the test work prioritized, and the PFS schedule rearranged to match the design timing to the availability of results. An alternative approach is to add design contingency, but it must be remembered that this will increase costs and reduce estimation accuracy.

Conducting PFS level testing then skipping the PFS rigor to commission a fast-track FS is a common mistake made by mining companies, both junior and multinational. The usual result (based on direct experience of many examples of this approach) is that the information set is far from adequate for conducting an FS, and additional un-costed testing and planning occurs during the fast-track study. At the end of the exercise, the FS is usually inadequate because it is out of date (e.g., more and better ore has been discovered), and the study schedule has blown out by months. These months would have been better spent conducting the PFS in an orderly fashion before deciding on the testing necessary for FS. The reality of a fast-track FS is that it takes about as long as a PFS followed by an FS, but for the entire time, an engineering team is consuming the owner's resources much faster because they have set in place FS staffing levels rather than the much lower PFS staffing levels. Worse, the delays shift the schedule for the preparation of the FS report to the extent that the promised expert resources (e.g., in estimating and financial evaluation) have moved on to another project, and lesser (but available) so-called expert resources are allocated to do this work. Where possible, one should strongly advise against conducting a fast-track FS and propose a professional PFS.

Evaluating the Adequacy of Outcomes

When evaluating PFS outcomes in preparation for a media release (e.g., acting as a Qualified Person under National Instrument NI 43-101, a Chartered Professional under JORC [2012], or in a similar manner under other jurisdictions), many key questions need to be answered, including the following:

- Has the test work been conducted on a mix of true variability samples and composites as appropriate?
- Can all the major design inputs and decisions be linked to both the average value and an estimate of its variability?
- Have the correct tests been performed to adequately assess the various processing options under consideration?
- Has a plan been prepared for additional testing ahead of the definitive phase?

Scoping Level

At scoping level, it is highly likely that no preceding metallurgical work exists, unless an old ore body is being reevaluated. The aim of a scoping study is to determine if this particular prospect is better than alternative prospects or investments vying for funding from one entity. As such, a sensible flow sheet of some sort needs to be evaluated at an accuracy of ±30% or greater. To select a sensible flow sheet for the major ore type(s), it is necessary to know how hard the ore is, according to the various comminution measures, and if a typical separation technique (such as flotation for copper sulfide mineralization) is effective. It is not necessary to optimize the flow sheet or compare alternatives, provided the selected flow sheet has a high probability of being successful. For example, if the evaluation is performed using a crush and ball milling circuit, and the investment is definitely attractive, this conclusion would not change (within scoping accuracy) if the same project were to be evaluated with a SAG or high-pressure grinding roll circuit. Where the fundamentals of recovery change (such as comparing a copper heap leach with a tank leach), it is legitimate to conduct both evaluations at scoping accuracy. Similarly, if a high-quality scoping evaluation shows the investment potential is marginal, then shuffling the equipment or efficiency deck chairs is not going to change this conclusion within scoping accuracy. Additional sampling and testing may show the scoping information was poor, and this is probably the best pathway to shifting the investment potential status. At the geological level, the best way to change the investment potential is to identify more ore. This leaves a marginal prospect in the situation where either more money is spent on exploring or testing, or it is shelved until conditions are more appropriate.

It is legitimate to work with composites at the scoping stage because there is little point in understanding ore variability until an ore body has been defined. In the author's experience, available scoping samples for a greenfield exploration prospect will have little relevance during PFS and FS evaluations because the resource knowledge and size expand exponentially (for a good prospect) in this time period.

Using Geology for Guidance and Sample Selection

If a prospect has reached scoping level, the geology should be well advanced. Geological domaining is the main source of guidance for metallurgical sample selection, and there is often a shortage of good core to test. First, the selection should be based on the major lithologies and five examples of each lithology should go into any test composite that constitutes more than 5% of the attractive mineralization. If a particular lithology is dominant (50% or more of the mineralization), then subdivision should be sought. Subdivision can be by alteration type, grain size, depth, and so forth. The aim is to ensure that no one scoping composite represents more than about 35% of the mineralized material.

These samples should be tested as if the most logical and common flow sheet for the ore type was to be implemented. If there is more than one obvious alternative, the test regime should be expanded to include the additional possibilities. In case the logical processing pathways become unattractive as a result of the initial test outcomes, there should be composite material in reserve.

When the test results are received, there should be initial indications if knowledge of the lithology (or another geological domaining basis) provides any guidance for metallurgical response. If lithology has no influence on metallurgical response, then another basis for understanding the mineralization geometallurgy will be needed. There will definitely be strong guidance as to how well the material responds to conventional processing, and this will provide a basis for estimating comminution and separation requirements to a scoping accuracy. If the separation test work was unsuccessful, there may also be grounds for selecting a more complex and expensive processing method with some confirmatory further test work.

Evaluating an Outcome

A scoping study should quickly arrive at a conclusion, indicating if the prospect is worth spending additional money on. A negative conclusion at scoping level is valuable as it allows the owner's scarce resources to be redirected to more favorable prospects. A strong positive conclusion is also valuable as it has identified real potential. With either of these definite scoping outcomes, the main point to evaluate is whether there is a strong plan for future work.

In the case of a strong negative outcome, the plan should relate to the conditions that may turn the prospect into a positive and how this should be addressed. This will typically be linked to a change in the metal price regime or future drilling success on the same prospect. The recommendations should include an ongoing geometallurgical plan and should be on a similar scoping level basis. There is no point planning for PFS level testing unless a positive scoping outcome is achieved.

In the case of a strong positive outcome, the evaluation should ensure that the correct preliminary test work has been conducted and the results have been correctly interpreted. The evaluation should also determine if a strong geometallurgical framework has been recommended for progressing the prospect to PFS level.

An unconvincing positive or small negative is problematic as, within the accuracy of the scoping study, the outcome could statistically be either a strong positive or a strong negative with deeper investigation. An evaluation of this type of inconclusive study should focus on methods of improving the evaluation accuracy with the minimal amount of test work or via consideration of alternative processing options. Again, an evaluation of geometallurgical issues, such as the representativeness of the samples that have been tested, requires close scrutiny. Often the best approach with such a prospect is to explore more and see if the resource itself improves before repeating scoping sampling, testing, and evaluation.

CONCLUSIONS

The last thing a project needs on commissioning is to encounter ore that was unanticipated or, at the least, not adequately designed for. Many projects have failed, or suffered a near-fatal setback, as a result of being unable to process the initial ore. Many other projects have faltered because the plant was designed for the average case, or the operational variability was considerably underestimated because variability for design was sourced from an annual mine plan. Still more projects operate in a state of semi-blindness because the mine planning outcomes do not provide the process plant operators with meaningful predictions of throughput, product grade, or other essential economic factors.

Actions to avoid these sorts of mistakes must begin as early as possible in the project study process and, if at all possible, in the exploration phase. It is the duty of qualified persons across the geological mining and metallurgical disciplines to understand what constitutes a strong geometallurgical project basis and what does not. The sooner that issues such as potential flow sheets, variability, and geometallurgical ore type definition are addressed within a study framework, the greater are the chances that the common operational problems will be avoided or that the owner will shelve the project at an appropriately early stage. It must be remembered that metallurgical test work is relatively inexpensive compared to drilling, and test work is a trivial cost compared to fixing an operating plant that fails to match the ore it is being fed.

Another important consideration is that the big-picture project is not all about the capital cost. Provided capital can be sourced, the real big-picture project is minimizing the time between accessing that capital and subsequently paying it back. Anything that can justifiably de-risk this payback period (and geometallurgy is at the top of the de-risking list) should be seen as an essential component of the study phases. The only asset a minerals project has is the mineralization, and all sensible measures should be taken to understand how value is generated by the asset so that it can (possibly) become a resource, then a reserve, and finally an operating project.

Conducting an effective geometallurgical evaluation of a project, especially at PFS and definitive levels, requires the correct metallurgical test samples and the performance of the correct metallurgical tests. This requires both appropriate time allowances in the schedule and the ability to access the appropriate geological sample types. Realistic allowances in the budget and schedule at, and between, the various study phases are a sign that such issues have been adequately considered in the project plan. Small test work budgets, inadequate time allowances, and, probably the worst of all options, the fast-track project that skips essential stages such as the PFS, are all geometallurgical red flags.

REFERENCES

Baumgartner, R., Gomez, P., and Escobar, G. 2016. Comprehensive mineralogical characterisation at the Cerro Corona Cu-Au porphyry mine—the fundamental key for geometallurgical applications. In *Proceedings of the Third AusIMM International Geometallurgy Conference (GeoMet) 2016*. Melbourne, Victoria: Australasian Institute of Mining and Metallurgy. pp. 221–230.

Chait, E., and Schiller, R. 2016. Adding copper recovery and acid consumption variables to the geological model of Quebrada Blanca. In *Proceedings of the Third AusIMM International Geometallurgy Conference (GeoMet) 2016*. Melbourne, Victoria: Australasian Institute of Mining and Metallurgy. pp. 257–266.

David, D. 2014. Geometallurgical guidelines for miners, geologists and process engineers: Discovery to design. In *Mineral Resources and Ore Reserve Estimation: The AusIMM Guide to Good Practice*, 2nd ed., Monograph 30. Melbourne, Victoria: Australasian Institute of Mining and Metallurgy. pp. 443–450.

David, D. 2015. Measuring and taking notice of orebody variability, an essential ingredient for reliable plant design. In *Metallurgical Plant Design and Operating Strategies—World's Best Practice (MetPlant) 2015*. Melbourne, Victoria: Australasian Institute of Mining and Metallurgy. pp. 10–26

David, D. 2017. An Introduction to Geometallurgy [presentation]. One-day course in geometallurgy.

Jackson, J., McFarlane, A.J., and Olson Hoal, K. 2011. Geometallurgy—Back to the future: Scoping and communicating geomet programs. In *Proceedings of the First AusIMM International Geometallurgy Conference (GeoMet)*. Melbourne, Victoria: Australasian Institute of Mining and Metallurgy. pp. 125–131.

JORC (Joint Ore Reserves Committee). 2012. *Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves*. Melbourne, Victoria: Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Minerals Council of Australia. Clauses 37–40.

Leichliter, S., and Larson, D. 2013. Geometallurgy for two recovery process operations at Cripple Creek and Victor gold mine. *Min. Eng.* 65(1):29.

Liebezeit, V. 2011. Geometallurgy data management—a significant consideration. In *Proceedings of the First AusIMM International Geometallurgy Conference (GeoMet)*. Melbourne, Victoria: Australasian Institute of Mining and Metallurgy. pp. 237–245.

Liebezeit, V., Ehrig, K., Robertson, A., Grant, D., Smith, M., and Bruyn, H. 2016. Embedding geometallurgy into mine planning practices—Practical examples at Olympic Dam. In *Proceedings of the Third AusIMM International Geometallurgy Conference (GeoMet) 2016*. Melbourne, Victoria: Australasian Institute of Mining and Metallurgy. pp. 135–144.

Martins, M.A.S., Medeiros, H., and dos Santos, B.Y.B. 1998. Geometallurgical studies for the Mineração Rio Do Norte Bauxites, Plateau Saracá III, Brazil. In *Latin American Perspectives: Exploration, Mining, and Processing*. Edited by O.A. Bascur. Littleton, CO: SME. pp. 155–162.

Morrell, S. 2011. Mapping orebody hardness variability for AG/SAG/crushing and HPGR circuits. In *International Autogenous Grinding, Semiautogenous Grinding and High Pressure Grinding Roll Technology 2011*. Edited by K. Major, B.C. Flintoff, B. Klein, and K. McLeod. Westmount, QC: Canadian Institute of Mining, Metallurgy and Petroleum.

McQuiston, F.W., and Bechaud, L.J. Jr. 1968. Metallurgical sampling and testing. In *Surface Mining*. Edited by E.P. Pfleider. Littleton, CO: SME-AIME.

National Instrument NI 43-101. 2011. *Standards of Disclosure for Mineral Projects*. Toronto, ON: Ontario Securities Commission.

Turner, N.L., Kuhar, L.L., Francis, N., Liebezeit, V., Ehrig, K., and Robinson, D.J. 2013. Development and testing of a small-scale uranium leach for geometallurgical applications. In *Proceedings of the Second AusIMM International Geometallurgy Conference (GeoMet) 2013*. Melbourne, Victoria: Australasian Institute of Mining and Metallurgy. pp. 345–350.

Wirfiyata, F., and McCaffery, K. 2011. Applied geometallurgical characterisation for life of mine throughput prediction at Batu Hijau. In *International Autogenous Grinding, Semiautogenous Grinding and High Pressure Grinding Roll Technology 2011*. Edited by K. Major, B.C. Flintoff, B. Klein, and K. McLeod. Westmount, QC: Canadian Institute of Mining, Metallurgy and Petroleum.

PART 2

Management and Reporting