Mine to mill reconciliation

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Mine to Mill Reconciliation – Three Case Studies

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ABSTRACT

Fluctuations in quarterly reconciliations between the mine and the mill are an expected and common experience in mining and processing operations. These fluctuations are caused in part by errors in the estimates of tonnage and grade of the mined material as well as errors in the mill's assessment of the material processed. Keeping these errors small and unbiased is an important aspect of an efficient operation whose primary goal is to derive maximum benefit from the exploitation of the mineral resource.

These fluctuations in quarterly reconciliations can also mask subtle but economically important problems which may take years to identify and resolve. Such problems can result from a poorly-conceived model of the mineralisation which affects both the spatial distribution of the ore as well as the tonnage – grade distribution over the range of cut-off grades. Poorly conceived models of the mineral resources and ore distribution can arise as a result of insufficient and inadequate drilling and sampling of the mineralisation, inappropriate application of geological constraints in the estimation process and the use of inappropriate methods to estimate the block grade distribution.

This paper presents three case studies from gold and copper mining operations, two of which experienced significant problems with their mine to mill reconciliations over extended periods of time. For the gold mine in case study 1, sampling problems were ultimately considered to be the main cause of the poor reconciliations between grade control predictions and the mill which persisted over a period of 18 months. In addition, incorrect application of resource modelling methods contributed significantly to a mismatching of the mill capacity with the mine at the beginning of operations. For the underground copper mine in case study 2, a change in the approach taken to resource modelling brought marked improvements in ore definition and more accurate predictions of ore grade and better production forecasts.

The third case study describes the background to the expansion of production to exploit a large tonnage of lower-grade stockwork mineralisation in an open pit gold mine. Several years of favourable Mine to Mill reconciliation data from the mining and processing of higher grade and lower grade stockwork styles of gold mineralisation provided the confidence in the approach to grade control to support the expansion plans during a period of low gold prices.

The paper is intended to complement the discussion of mineral resource estimation methods and risks in Section 6.1.3 of the *Mine Managers' Handbook* to be published by the Australasian Institute of Mining and Metallurgy (Schofield, 2012).

INTRODUCTION

Mineral resource estimation and uncertainty

Financial success in the mining of a mineral deposit depends critically on the quality of the estimate of the mineral resource: its location in the subsurface and the value or grade of the ore minerals at that location.

Many mining people believe that an estimate of lesser quality implies a greater chance of financial failure or at least some reduction in the expected financial return if mining proceeds. This view of resource estimation places strong emphasis on 'getting the estimate right' and tends to ignore several fundamental facts underlying the problem of resource estimation:

- All mineral resource estimates carry with them a range of uncertainty which arises from the fact that each mineral deposit is a unique and complex entity and that only a very limited knowledge

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of it is ever possible, even after mining has begun. The principal source of knowledge of the deposit is usually drill hole samples, the locations and grades of which are subject to sampling and measurement errors. In general, some component of the total uncertainty surrounding a resource estimate is irreducible.

- Improving the quality of a mineral resource estimate requires increasing amounts of money to be invested in the project prior to production.

Approaches commonly used to reduce uncertainty surrounding the resource estimate are not uniformly effective. Increasing the density of drill hole samples and improving the quality of those samples tend to be far more effective, but also more expensive, than placing greater emphasis on more detailed geologic interpretation based on the current drill hole information. The quality of the resource estimate is fundamentally limited by the amount and quality of the sampling upon which it is based.

One of the most important sources of uncertainty in mineral resource estimation lies in the underlying and generally strong positive dependence between the magnitude of the local grade and the local variation of the grade, usually referred to as the *proportional effect*. A direct consequence of this dependence is that in areas of higher-grade mineralisation in most mineral deposits, local grade variation is much larger and the estimated grades of mineable units carry larger estimation errors ie greater uncertainty. Some resource modelling methods are more effective that others at mitigating the effect of this source of uncertainty.

The uncertainty in mineral resource estimates introduced by errors in sample location, sample measurement and sample quality can be devastating to the success of a mining project. Drill hole surveying errors or the lack of surveys cause serious problems for medium and long term planning and can induce a complete lack of confidence in the resource estimates. These kinds of errors tend to increase with drill hole depth resulting in deterioration in the quality of planning and scheduling with depth. Such errors arise more often in projects which are subjected to accelerated development time frames or under-funding.

In the present day mining and mineral exploration industry, two approaches to reducing the uncertainty in mineral resource estimates appear to be more commonly practiced. A popular and deceptively less costly approach is to create an illusion of greater certainty through detailed geologic interpretation of the drill hole data – a wire frame model of geology/mineralisation. The JORC Code (JORC, 2004) lends credibility to this approach where it requires the establishment of continuity in geology for the reporting of Measured and Indicated resource estimates. The more expensive, more objective and decisively better approach to reduction in uncertainty attached to resource estimates is to increase the sample coverage of the mineralisation with more and regularly spaced drilling. With this approach to reduction in uncertainty, the problem is to decide when to stop drilling: at some point, the benefit brought by the additional samples is less than the cost of acquiring them.

Forecast and actual production

In most mining operations, medium- to long-term production forecasts are made using the resource model and short-term production forecasts are based on the grade control modelling. Typically, the resource model and the grade control model are constructed from completely different data sets. This is an important aspect of the operational process for problem solving because it allows two potentially independent views of the mineralisation through sampling and modelling.

Uncertainty in mineral resource and grade control estimates contribute directly to differences between forecast production and actual production at both long-term and short-term scales. The relative magnitude of the contribution obviously depends on problems in other areas of the mining operation such as changes to the areas that were planned for mining in the current production period, and recovery problems in the plant for example.

The experience of the past 20 years suggests that errors in sampling and problems in the application of geological modelling to resource estimation are the two main contributors to the poor performance of resource and grade control models in forecasting actual production.

In the case of sampling, there is an unfortunate willingness to dismiss downhole surveying as superfluous, and quality control in sample collection and processing as an unnecessary burden because of an incorrect belief that sampling errors are averaged out in modelling (Schofield, Gatehouse and Carswell, 2011). In fact, high error rates in sample locations and sample grades ultimately make production planning and scheduling largely futile.

The use of detailed geologic interpretation incorporating some form of grade threshold applied to drill hole samples to 'constrain' which samples are used to estimate block grades and which volumes will be estimated, is the other major cause of biased resource estimates and poor reconciliation of forecasts and production.

Section 6.1.3 of the *Mine Managers' Handbook* (Schofield, 2012) presents an overview of the most important issues affecting the quality of mineral resource estimates and the way in which these issues are handled with modern resource estimation methods. One goal of this paper is to illustrate some of these issues through a number of interesting case studies which exhibit the kinds of problems that can arise as a result of inadequate quality control in sampling and inappropriate application of modelling methods. Another goal is to illustrate the benefit in terms of confidence that is derived from a history of favourable mine to mill reconciliation data when planning operational expansions.

Case study 1 – The Damang Gold Mine: Ghana

The following narrative is a summary of the sequence of events that occurred during the period in which Neil Schofield was involved in undertaking and advising on mineral resource and grade control modelling at the Damang Gold mine in Ghana, West Africa. Post feasibility and prior to production, he became involved in advising on grade control modelling and subsequently was involved in revisions of the mineral resource model. Figure 1 shows drill hole cross-sections illustrating the distribution of the main geological units and drill hole gold grade in the eastern limb of the Damang Anticline.

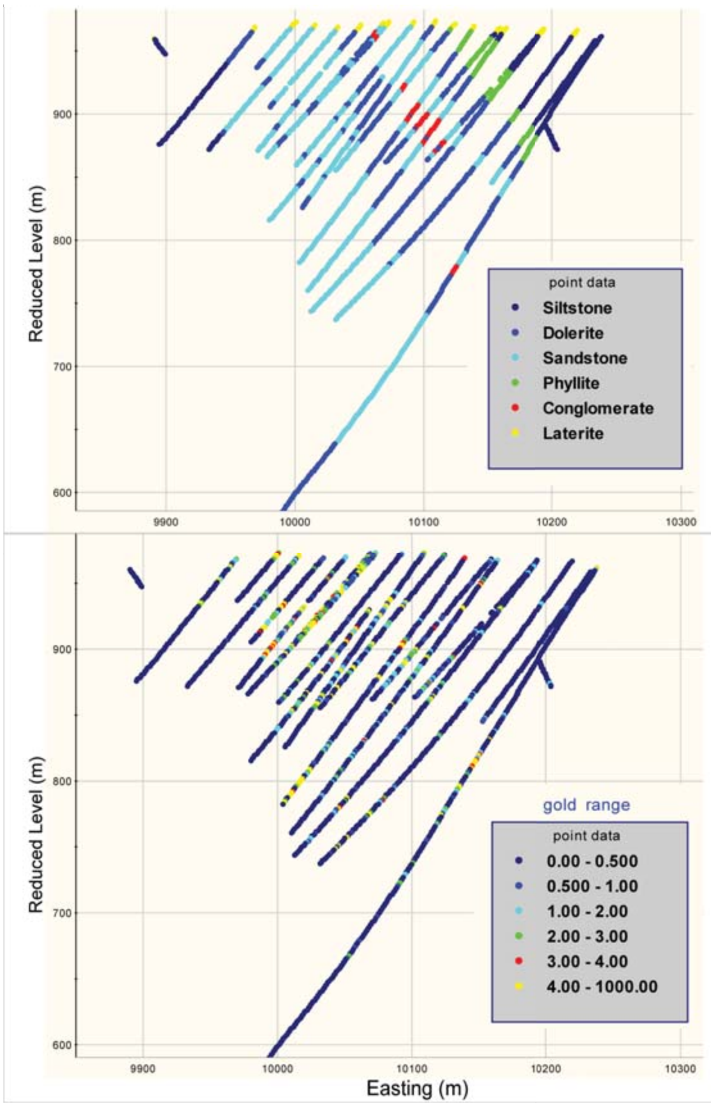

FIG 1 - Drill section showing geology and grade distribution, Damang Gold deposit.

Preproduction

The open pit gold mine came into production in late 1997 after extensive exploration and resource definition drilling followed by a feasibility study. The gold mineralisation occurs along an extended strike length of around 4 km in Proterozoic meta-sediments including quartzites with conglomeratic lenses and accompanying siltstones and phyllites. The genesis of the gold mineralisation was considered to be both sedimentary, in the conglomerates and sandstones, as well as hydrothermal associated with extensive quartz veining (Tunks *et al*, 2004).

The resource model which formed the basis of the feasibility study was generated from both diamond (DD) and reverse circulation (RC) drilling spaced at around 20 m across the strike and 40 m along the strike of the mineralisation. The area of the start-up pit was drilled more densely to 20 m spacing. All resource drill hole samples were assayed by fire assay. The resource model was strongly constrained by an interpreted geological model of the mineralised sediments. The method of multiple indicator kriging (MIK) (Deutsch and Journel, 1992) was used for resource estimation.

The feasibility study indicated a reserve of 22.9 Mt with an average grade of 3.1 g/t gold at a cut-off grade of 1.0 g/t. An audit of the resource and reserve model was undertaken and no significant problems were identified. The planned capacity of the plant was 3 Mt/a with the mine operating to a 1.0 g/t cut-off grade producing around 299 000 oz of gold per year.

Prior to production, extensive grade control drilling and sampling to a number of fixed levels was done in the area of the startup pit to provide a check on the predictions of the resource model as well as to provide a good basis for short- to medium-term planning in early production. Due to assay laboratory limitations, a large proportion of the preproduction grade control samples were processed by the Aqua Regia method rather than fire assay.

In the area of preproduction grade control drilling, grade control modelling indicated a mineral reserve of 1.67 Mt at a grade of 2.08 g/t (112 000 oz) at a 1.0 g/t cut-off. Over the same volume, the feasibility reserve model predicted 0.72 Mt at a grade of 3.56 g/t (82 500 oz) at the same cut-off grade.

Production to end of March 1998

The first three months of production supported the grade predictions of the grade control modelling. A cut-off grade of 1.6 g/t was required to achieve a head grade in excess of 2.8 g/t. Grade control to mill reconciliations in this period indicated a five per cent short fall in gold production; startup problems in the mill was considered the likely cause. The grade control modelling was entirely independent of the resource modelling: only grade control sampling was used and interpreted geologic wire frames were not used.

An extensive review of the feasibility resource model was undertaken after around four months of production by external consultants Hellman and Schofield. A revised model, based on the same sample data and utilising the MIK modelling method, but without hard boundary wireframe modelling of the geology, provided results consistent with those of the grade control models.

Around the end of the first quarter of production, the mine changed the assaying method for grade control samples from the Aqua Regia method to the LeachWELL method – an accelerated cyanide leach method.

Production April to June 1998

During this period, reconciliation of the grade control modelling with the estimates of the revised resource model supported those of the revised resource model over the feasibility resource model. However, grade control to mill reconciliation began to deteriorate from a short fall of five per cent to an average of around 12 per cent.

Production June to December 1998

In this period, resource to grade control reconciliation continued to perform well but the grade control to mill reconciliation remained a problem. Management began to lose confidence in the grade control modelling and suggested introducing polygonal grade control modelling. Sampling and assaying quality was not investigated.

Grade control modelling was shown to be consistent with the grade control sampling data. It was demonstrated that the grade control modelling could not be the cause of the shortfall in gold production. A decision was taken to introduce cutting of the grades of grade control samples to lower the predictions of ore grade to the mill, with the expectation that better reconciliations with the mill would be achieved.

Production January to May 1999

In this period, resource to grade control reconciliation began to deteriorate with the resource model predicting consistently more tonnes of higher-grade. An investigation of the grades of deeper resource RC and DD samples showed that RC samples were biased high compared to DD samples. The magnitude of the bias suggested this as an explanation of the deteriorating resource to grade control reconciliation.

Grade control to mill reconciliation deteriorated even further to an average of around 15 per cent shortfall in gold production with some months exceeding 20 per cent. A police investigation failed to identify any activities involved in large-scale gold theft.

Production post-June 1999

In the June to July 1999 period, grade control to mill reconciliation changed suddenly from a short fall in gold production of about 15 per cent to around a 20 per cent over-production of gold. No logical explanation could be found for this change at the time. Resource to grade control reconciliation remained an ongoing problem.

After the cutting of grade control sample grades was discontinued, grade control to mill reconciliation settled down to an overproduction of gold of less than ten per cent and generally less than five per cent. Much later, the grade control to mill reconciliation problem was linked to sampling problems in the preproduction and early grade control period, in particular wet sampling in swampy areas. Also, a significant number of early grade control holes were believed to be incorrectly located.

Case study summary

The significant underestimation of the tonnage of economic ore at the 1.0 g/t cut-off in the feasibility study led to the construction of a mill that was not matched to the production capacity of the mine. As a consequence, large tonnages of profitable ore had to be stockpiled while the mine operated at a much higher cut-off to achieve the gold production anticipated by the feasibility study. The processing capacity of the mine was increased to a more appropriate level over time, but the combination of Mine to Mill mismatching together with the poor reconciliations experienced as a result of errors in the feasibility resource modelling and sampling problems during the first 18 months of production detracted significantly from the overall performance of the operation.

Case study 2 – The CSA Copper Mine, New South Wales, Australia

The CSA mine is situated in the Lachlan Fold Belt of New South Wales where copper and minor lead and zinc have been mined underground at the rate of about 1 Mt/a with shaft hoisting since the early 1970s.The case study refers to a period of mining from the late 1980s to the early1990s when John Carswell held the position of principal geologist at the mine.

Copper occurs almost exclusively in chalcopyrite along with other sulfides which include sphalerite and galena, and is hosted in dilation zones in a number of en echelon, steeply dipping quartz filled shear zones. The shear zones are well defined and characterised by coplanar cleavage, strong silica-chlorite alteration and have lateral and vertical extents of well over 1 km. The ore textures vary from massive poddy sulfides to stockworks and breccias. The drill hole cross-section in Figure 2 illustrates the structure and continuity of the QTS North copper lodes at the CSA mine.

The ore zones are commonly 20 - 50 m long, 5 - 15 m wide and have a steep northerly plunge and down dip/plunge dimensions of up to 500 m. Stopes were designed at a two per cent copper cut-off grade, dilution was about ten to 15 per cent and stopes were back filled with development waste or uncemented tailings. Mining was by long-hole stoping from 30 - 40 m spaced sublevels and the stopes were up to 60 m long.

Final ore definition drilling was at a spacing of 10 m along strike and 20 m down dip with LTK46 (36 mm diameter) coring with whole core assaying. Ore outlines were for some time based on 10 m sectional interpretations and later manual 3D wire-framing to mitigate potentially significant projection errors in sectional interpretation due to the plunge of the ore shoots. Grades estimations for the sectional and wire-framed estimates utilised area/volume of influence techniques. Based on historical annual reconciliations, dilution was applied at rates of five to 15 per cent depending on the

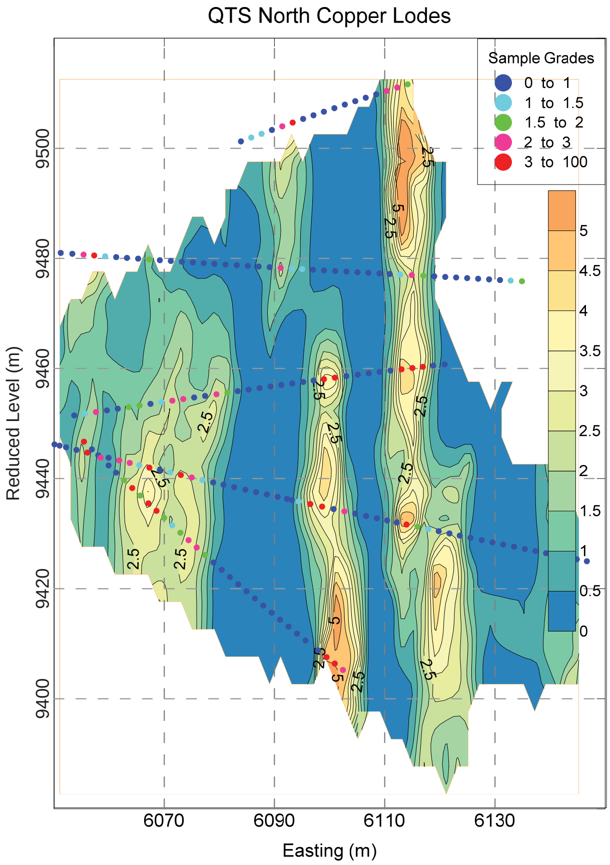

FIG 2 - Drill section showing sample grades and modeled copper lodes, QTS North mineralisation.

geotechnical domain. A correction of ten per cent was applied to copper estimates due to consistent overestimation of copper grades of these stopes.

Weekly reconciliations were introduced to test and monitor the change from 2D to 3D interpretations and the long-term dilution and correction factors. The reconciliations also attempted to correct for significant variations in bulk density between ore domains due to differences in sulfide mineral content. The reconciliations demonstrated that:

- some overestimation of copper grades was associated with overestimating tonnage from low bulk density stopes; production from high bulk density, mainly lead/zinc deposits, was underestimated
- real dilution rates in copper stopes was less than previously estimated
- estimation error was reduced and ore waste definition improved with 3D wire-framing but significant overestimation still occurred in high-grade stopes particularly those with extremely variable copper grade and strongly skewed grade histograms.

In 1990, geostatistical estimation methods were trialed in a number of high grade copper stopes using multiple indicator kriging (Carswell and Schofield, 1993). The results of these trials showed:

- a much improved definition of the ore-waste boundaries compared to the wire-framing methods
- more accurate but more conservative predictions of stope grades and an increase of about ten per cent in stope ore tonnages.

Following these trials that were undertaken with the assistance of the senior author, geostatistical methods were introduced across the mine and coincided with the transformation of the geological database from hardcopy plans to totally digital.

Case study summary

Throughout the period of the case study, the mill was under-utilised and the extra tonnage even at marginal grade was profitable. The case study demonstrates the benefits of using the most appropriate and advanced estimation methods, which include better selectivity and more accurate grade estimation, leading to better production forecasting. The improvements came not only from the change from wireframing to geostatistical techniques but were supported and validated by improved mine to mill reconciliation.

Case study 3 – the Macraes Gold Mine, New Zealand

Introduction

At OceanaGold's (OGL) Macraes open pit and underground operations, gold mining is centered on the Hyde-Macraes Shear Zone (HMSZ), an extensive low-angle shear zone that can be traced for over 30 km of strike (Daniels and Mascini, 2011). The shear zone strikes NW-SE, dips at approximately 15° to the NE, and is typically about 100 m thick. Gold mineralisation is most intensely developed along the top of the HMSZ, which is defined by the hanging wall shear. The hanging wall shear comprises deformed schist, mylonite-cataclasites, quartz veins and breccias, and from start-up (1990) until 2001, had provided the majority of mill feed. It is however, the mineralisation developed beneath the hanging wall shear, historically termed 'stockwork', that is the focus of this case study. The 'stockwork' mineralisation comprises subvertical arrays of quartz veins as well as numerous erratically developed shears and is typically more broadly developed than hanging wall mineralisation (Figure 3).

By 2001, OGL (then GRD Macraes Ltd) faced the challenges of increasing stripping ratios, longer haul distances and rising costs. OGL had also identified grade biases in earlier campaigns of RC drilling that had been sampled under wet conditions and had assessed their impact on what were already narrow profit margins. It was in this context that the 'stockwork' conversion project was initiated by OGL.

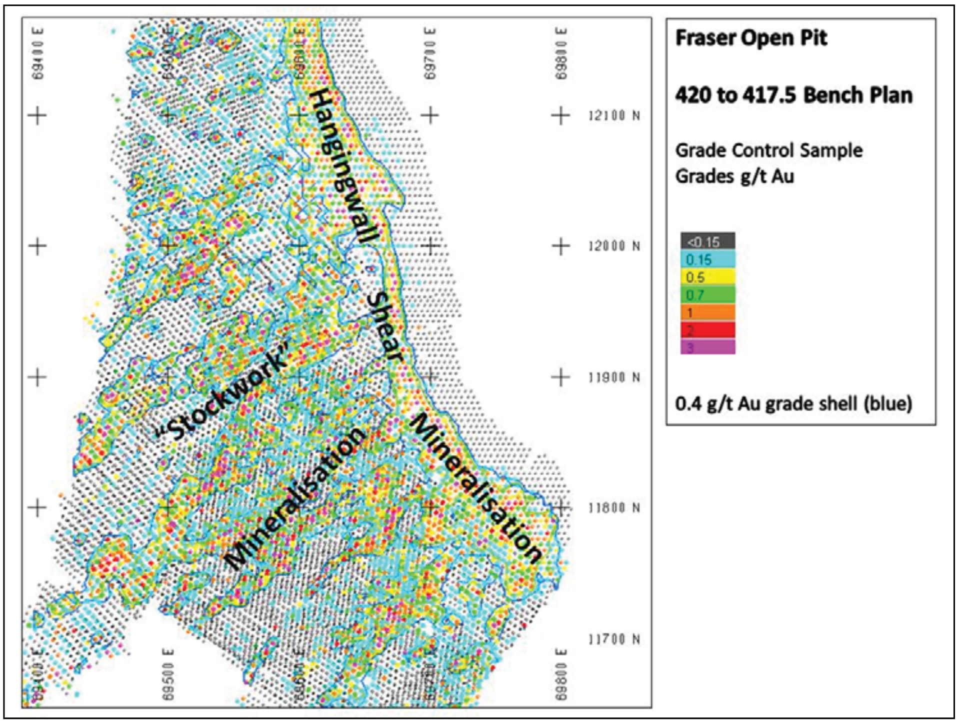

FIG 3 - Bench plan with grade control sample grades, showing both mineralisation styles, Macraes Gold Mine.

The project sought to assess the extent to which this complex style of mineralisation, which at the time was sparsely drilled (typically at drill hole spacings of between 50 m × 25 m and 50 m × 50 m) could be included in the Frasers open pit reserves. Approximately 95 per cent of the drilling was reverse circulation, precluding meaningful lithological/structural interpretation. Neither the gold price of around $US 300 /oz nor the average grade of the stockwork mineralisation, at around 1.0 g/t, was particularly attractive at the time.

Eleven years on, over 25 Mt of the stockwork mineralisation have been profitably mined and processed and this resource continues to provide around 40 per cent of open pit mill feed.

The stockwork conversion project

The aim of this project was to bring a substantial proportion of 'stockwork' mineralisation within the Frasers open pit into reserves based on the drilling available at that time. After considerable review, a number of risks and opportunities were recognised.

The opportunities were:

- large volumes of this mineralisation were anticipated
- only moderate incremental waste stripping (below the mineralised hangingwall shear) would be required and therefore mining risk would be lowered
- if large, broad pit stages could be mined, the resource estimation risk would be aggregated and therefore lowered.

The key risks were:

- the structural complexity, poor spatial continuity, and the low average grade (marginal profitability) of the mineralisation
- very little stockwork mineralisation had been mined in Frasers open pit at the time, so OGL's geological understanding was limited
- approximately 95 per cent of the drilling was RC, precluding meaningful logging of the visually subtle rock types and structure
- the broad drill hole spacing was typically between 25 m × 50 m and 50 m × 50 m
- most of the drilling was vertical, while much of the mineralised veining dipped at between 70° to 90°
- the concern, based on practical levels of resource drilling, that the resource model would not provide a sound basis for short to medium term (less than three months) scheduling of this mineralisation.

Grade control

In 1998, polygonal grade control estimation was replaced with conditional simulation – based grade control. Conditional simulation modelling is used in conjunction with trench mapping and ore spotting for ore block design and control. Since the introduction of conditional simulation modelling, annual grade control estimates have generally been within a few per cent of the back-calculated mill estimates with the exception of 2001 when the difference was nine per cent.

Figure 3 shows a typical grade control bench with blasthole sample grades on an approximate 4.5 mE × 4.5 mN pattern. Sample lengths are 2.5 m if vertical, or adjusted to 2.5m flitch height if inclined (commonly 70°) to better intersect vein packages.

Resource modelling

Up until 2002, OGL were using small block (5 mE, 5 mN, 2.5 mRL) E-type resource estimates, via multiple indicator kriging, with substantial volumes of the mineralisation being modelled within grade-based wireframes. The resource predictions via this approach were found to be severely under-estimating in some areas of stockwork; in the cases of Innes Mills Pod B and Southern Pit stage IV mining respectively realised 2.5 and 1.5 times the predicted contained gold.

As part of the project, OGL transitioned to large panel (25 mE, 25 mN, 2.5 mRL), recoverable resource estimates with multiple indicator kriging. Other major changes accompanying this were to reduce the amount of mineralisation constrained by wireframes (to about 50 per cent), to reduce the number of geological domains, as well as to revise the resource classification scheme. Collectively these measures improved the resource estimates.

Resource classification

A three tier resource classification was implemented in which minimum numbers of sample data and minimum numbers of informed search octants applied:

- Tier 1 is geological; hanging wall mineralisation is treated with more confidence (requires less drilling) than 'stockwork' mineralisation, due to its grade continuity.
- Tier 2 is based on drilling density:
- for hanging wall 50 m × 50 m/100 m × 100 m for Measured/Indicated

- for 'stockwork' 32 m × 32 m/50 m × 50 m for Measured/Indicated.
- Tier 3 is grade based, is only applied to 'stockwork', and requires that the model panel has at least a 30 per cent probably of exceeding 0.5 g/t Au to be classified as Indicated.

Risk caveats

The resource estimates for 'stockwork' were premised on the mining of broad pit stages, and given the combination of sparse drilling, geological complexity and low-grade continuity, were not intended to provide reliable estimates at smaller scales (eg monthly mine planning). Because grade control drilled stocks rarely project more than two to three weeks ahead, scheduling for the one to three month advance window was anticipated to be, and remains a challenge.

At the time, some rules of thumb for checking of risks were recommended prior to inclusion of Indicated 'stockwork' in reserves:

- Optimised pit shells were to be reviewed, looking for areas of high risk, particularly areas with erratic floor topography and sparse drilling; for instance, shells coning down on modelled stockwork blocks, locally supported by only a small number of drill holes.
- As a proxy for uncertainty in modelled grade, optimised pit shells based on gold prices discounted by ten to 15 per cent were to be considered. Areas of Indicated 'stockwork', where these discounted shells retreated, were to be considered high risk.

In both instances, areas not passing these tests were to be either selected for infill drilling, or excluded from reserves.

Reconciliation and scale

Figure 4 presents scatterplots of the resource model estimates[4] versus grade control block model estimates, from left to right, for 25 mE × 25 mN × 2.5 mRL (an individual resource model panel), 50 mE × 50 mN × 5 mRL, and 100 mE × 200 mN × 20 mRL (256 resource model panels) volumes respectively. These areas are typically drilled to about 25 m × 50 m. The correlations between resource and grade control model estimates, while poor for individual resource model panels, steadily improve as the considered volume increases.

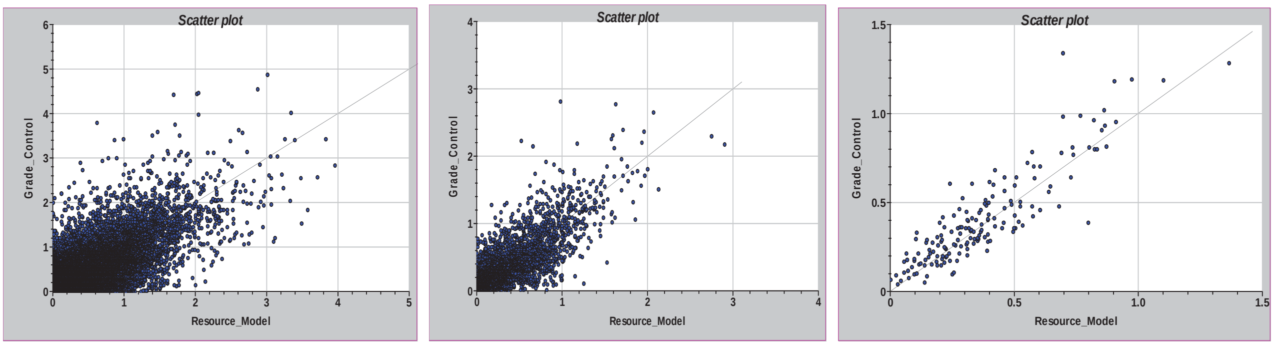

FIG 4 - Grade control ore proportions versus resource model proportions for increasing volumes of mineralization, Macraes Gold Mine.

The ore proportions within panels is on average about 40 per cent[5], so panels of dimensions 25 mE × 25 mN × 2.5m RL, 50 mE × 50 mN × 5 mRL, and 100 mE × 200 mN × 20 mRL represent about 2000 t, 15 000 t and 400 000 t of ore respectively. Based on analysis of the last (ie for mined packages of 400 000 t, which represent about one month's mining), there is a 20 per cent chance that mining of the package will result in a 20 per cent shortfall (or worse). For packages aggregated up to perhaps 1 000 000 t (two to three month's mining) the resource model provides less erratic estimates.

Figure 5 provides a resource model to grade control reconciliation by bench. The chart presents entire benches and does not fully reconstruct the complex sequence of pit staging, nor does it reveal the challenges to mine planning that result from short term modelling uncertainty. These benches typically contain 300 000 t to 700 000 t, and on average are drilled to approximately 25 m × 50 m. The total reconciled tonnage is approximately 40 Mt, which includes about 50 per

4. Estimate of gold contained above 0.5 g/t Au cut-off.

5. That is, on average approximately 60 per cent of the tonnage for each panel falls below the 0.5 g/t Au cut-off.

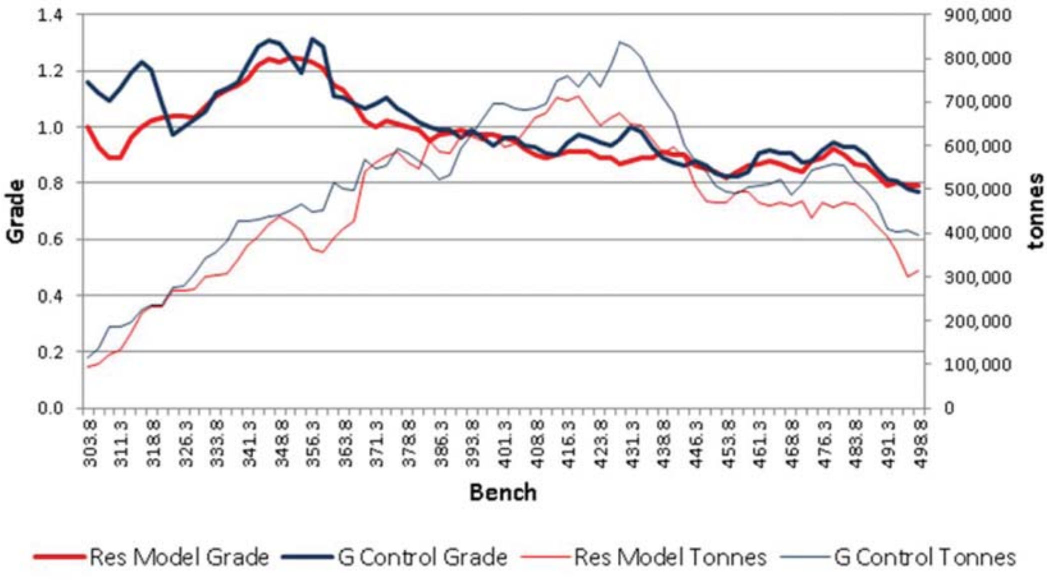

FIG 5 - Bench reconciliation of resource model to grade control model, Macraes Gold Mine.

cent hanging wall mineralisation. While some disparities occur on individual benches (particularly for tonnage), generally the model provides reasonable predictions when aggregated over 750 000 t. Local aberrations in the geological interpretation have led to periods of poor reconciliation from time to time; however the majority of reconciliation disparities have occurred in areas of sparse and/or flawed sampling.

Case study summary

For the Macraes open pit mine, the improvements in prediction and reconciliation which arose from changes to both grade control and resource modelling provided a basis of confidence for the mine management to consider the development of a much lower-grade but more extensive style of mineralisation at a period of unfavourable gold prices. There is no doubt that the increases in gold price that occurred in the ensuing years have contributed greatly to the success of the project.

CONCLUSIONS

Mine to Mill reconciliation is an important part of the routine checks required for efficient exploitation of mineral resources but it may not help to identify important underlying problems that can destroy economic value in an operating mine. Mine managers need to be aware of the ways in which a range of factors can affect reconciliation outcomes and how they can affect the economics of the mine.

All resource and reserve estimates are based primarily on drill hole samples and sampling errors are one of the principal sources of reconciliation problems. Routine monitoring of sample quality and investigation of aberrant behaviour is essential to the smooth and efficient running of a mining operation.

Resource and Reserve estimates are generated using estimation procedures which partition the samples into groups and generate averages from these groups. The common practice of geological wire-framing is often used incorrectly to partition the samples in a way which generates biased estimates, leading to poor planning decisions at both short term and long term scales and significant financial loss. More detailed geologic interpretation is generally not a useful substitute for more drilling to reduce the uncertainty associated with resource estimates.

On the other hand, well behaved reconciliation outcomes together with well-established quality assurance procedures for sampling and appropriate application of resource and grade control modelling methods can form the basis for a smooth and efficient operation and provide the foundation and confidence for significant investment decisions.

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