Estimating Mining Factors (Dilution and Ore Loss) in Open Pit Mines

R Bertinshaw and I Lipton

ABSTRACT

Dilution and ore loss have an important bearing on the success of open pit mining projects. Their impacts on production costs, revenues and production rate require that they are correctly modelled in the long-term mine plan. The estimation of dilution and ore loss is therefore an essential part of any Ore Reserve estimate or mine schedule. Large open pit mines are driven by the economies of scale that are achieved at the cost of less selectivity in the discrimination of ore and waste. Planning for these mines is facilitated by the use of block models for mine design, scheduling and the conversion of Mineral Resources into Ore Reserves.

This paper reviews various approaches to the estimation of dilution and ore loss, as implemented in computer block models. In the case of orebodies with well-defined sharp boundaries, rather than irregular stockwork mineralisation, the geological interpretation is a critical factor. Variability in the geometry of orebodies is common within large open pits and must be taken into account when estimating dilution and ore loss. A method of regularisation, controlled by the wireframed geological model, is demonstrated, using examples from large open pit stratiform copper and magnetite deposits. This can be carried out without special computer software modules.

INTRODUCTION

The estimation of dilution and ore loss that will be incurred in a mine is important on many accounts. Not least of these is that it is an essential aspect of Ore Reserve estimation (JORC, 2004).

Dilution results in the processing of material for which the revenue generated is less than the targeted economic limit or, in many cases, less than the cost of production. Mines generally aim to minimise dilution unless profitability can be improved by taking a less selective approach to mining. Dilution has a critical role in large open pit mines because these mines are commonly driven by economies of scale and operate with narrow profit margins. If dilution is higher than planned, profit margins may be eroded. High target production rates and large-scale equipment may limit the ability of these mines to respond to unforeseen dilution problems.

Dilution is lower than economic cut-off grade material (waste) that is taken with the ore as part of the mining process. It is dependent on the style of the geological interpretation and the method of modelling of the Mineral Resource. For example, if a stockwork of mineralised veins is interpreted as a series of discreet, very narrow ore zones, dilution, for a given mining method, will be much higher than if the mineralised veins are modelled as a bulk, low grade package.

The sources of dilution are diverse and can be divided into four types:

- **Type 1: dilution due to geometry.** This dilution occurs due to the incompatibility between the geometry and operation of the excavation equipment and the geometry of the ore boundaries. It is related to the size of the excavator bucket, bench height, and the strike and dip of the ore contact. It includes waste material taken at the external boundaries of the orebody and internal waste within zones that are too small to be selectively mined. This has been referred to by Shaw and Khosrowshahi (2004) as ‘planned dilution’.
- **Type 2: dilution due to uncertainty in knowledge of the in situ ore boundary.** This dilution occurs due to uncertainty or lack of precision in the sampling and assaying, geological interpretation or grade estimation. It is inherent in the Mineral Resource model. It may be reduced by improved methods or more detailed sampling but it can never be entirely eliminated.
- **Type 3: dilution due to blast movement.** This is the mixing of waste which occurs as the result of throw and heave during blasting. It is an expected consequence of blasting but is difficult to predict or measure due to the large number of variables that control blast fragmentation.
- **Type 4: dilution due to mining errors.** This dilution occurs at the time of mining and is, in some way, an unintended consequence of mining procedures. It includes inaccurate mark-out due to poor survey control, geotechnical failures, inaccurate digging (operator error), misdirected trucks, spreading of waste as road-base on pit floors, etc.

Type 1 dilution may be addressed by changing the mining method or direction or by reducing the size of the mining equipment. Type 2 dilution may be reduced by increasing the sampling density and improving sampling and analytical techniques. Type 3 dilution may be reduced through the use of devices for the monitoring of blast muck pile movement. Types 3 and 4 dilution may be addressed by improved training and supervision, and by using regular reconciliations to ensure that poor mining practices do not creep in.

Mining loss is that part of the Mineral Resource that is above the economic cut-off grade and was intended to be mined as ore but is not sent to the mill or placed in an ore stockpile, ie it is lost to waste. Mining recovery is related to mining loss but is expressed as ore recovered as opposed to material lost. The material lost is usually considered to be post dilution.

If procedural improvements are made to reduce dilution, such as greater sampling density, ore loss will also be reduced. However, more commonly, planned ore mining boundaries are deliberately modified to reduce dilution, at the cost of sacrificing additional ore loss.

**SOME APPROACHES TO ESTIMATING DILUTION IN OPEN PIT MINES**

Some approaches to the estimation of dilution, as implemented using computer block models, are summarised below.

**Multiple indicator kriging**

Multiple indicator kriging (MIK) is a geostatistical method of resource estimation that accounts for dilution and ore loss when applied to mineral deposits in which the internal and external ore boundaries are probabilistically defined (Isaacs and Srivastava, 1989; Grovaerts, 1997). Several case histories have been published (eg Hill, Mueller and Bloom, 1998; Lipton, Gaze and Horton, 1998). The MIK method builds an approximation of the distribution of sample grades in the neighbourhood of each
block. A change of support correction is then applied to the local distributions of sample grades to produce an approximation of the distribution of grades at the scale of the chosen selective mining unit (SMU). The SMU is taken to be an approximation of the minimum practical mining unit, which might be for example, of the order of 10 m³, compared to a sample size of perhaps 0.1 m³. Since the variance of the grades of SMUs is much less than that of small samples, the support correction compresses the distribution. In doing so, the portion of the distribution above a selected cut-off grade changes. This alters the tonnage and grade above the cut-off grade and may be considered to reflect the impact of ore loss, dilution and expected mining recovery, so that these are built into estimates of the resource for blocks of the selected SMU size.

The support correction is derived from the continuity of grades, as modelled by the variogram. However, the correction derived in this way does not include the impacts of Type 2, 3 and 4 dilutions. Consequently, the correction factor is commonly adjusted, either based on experience or on Chain of Mining studies, to more fully account for each of the sources of ore loss and dilution (Shaw and Khworsahli, 2002).

MIK has several advantages as a resource estimation method, including that it allows explicit modelling of the continuity of grades over a range of cut-off grades and that it deals much better with highly skewed distributions, such as in typical gold deposits, than estimation methods such as ordinary kriging and inverse distance weighting. Importantly, it produces estimates that are matched to the SMU size and it builds in ore loss and dilution based on the local data, rather than global factors. The MIK method is particularly well-suited to stockwork or disseminated styles of mineralisation including porphyry copper deposits and large low-grade stockwork gold systems; however, it is only suitable for use within a geological domain that is reasonably homogeneous (one that satisfies the geostatistical condition of ‘stationarity’). It cannot be used to correctly model the dilution that may occur across the boundary between one geological unit and another. It is therefore unsuitable for modelling dilution at the boundaries of deposits with strongly defined stratigraphic, lithological or grade boundaries.

MIK is also not ideally-suited to deposits where multiple elements are to be modelled because the technique only models the distribution of a single variable. Unless all the variables are strongly correlated, it is not possible to evaluate a second or third variable against a cut-off grade specified for the primary variable. MIK is therefore not well-suited to iron ore deposits which typically require estimation of variables including Fe, SiO₂, Al₂O₃ and P.

A further practical difficulty with MIK models is that they are more complex to use as an input to open pit optimisation, mine scheduling or detailed mine design because each block carries an approximation of the local grade distribution and the exact location of ore boundaries is not specified by the model. For these reasons, mining engineers commonly prefer to revert to simpler models with a single grade estimate in each block.

### Addition of average grade diluent

Dilution of Type 1 (planned dilution) may be addressed by using information about the ore geometry. In the following sections, we examine some of these methods.

Adding an average dilution across the whole orebody is perhaps the method most commonly used by mining engineers. It involves the estimation of a diluent grade which is then added as a percentage of mining production. It is typically added as part of the mine scheduling process, prior to completion of the Ore Reserve estimate.

The advantages of this simple approach are:
- the method is intuitive and simple to apply, and
- it works equally well for multiple grade variables.

The disadvantages are:
- It uses global values and so is not necessarily correct locally. There may be no factual basis for the estimation of the grade and percentage of dilution.
- It takes no account of geological information, particularly the geometry of the mineralised zones, which controls Type 1 dilution.
- It leads to incorrect selection of ore and waste in economically marginal parts of the deposit.

A variation of this approach is the addition of a percentage of diluent material as dilution having zero grade. This has become a common method as it is used by many open pit optimisation packages such as the Whittle Optimisation program. However, as is noted in the Whittle manual (1999), the preferred approach is to deal with dilution as part of the model construction process. In addition to the disadvantages noted above, the assumption of zero grade dilution is highly conservative when applied to gold and base metal deposits. Conversely, when applied to ferrous metal deposits, this approach results in underestimation of the grades of deleterious components such as SiO₂.

It is arguable that with the almost ubiquitous availability of powerful computers and sophisticated mining software at mine sites, such a simplistic approach should no longer be acceptable to stakeholders.

### Standard regularisation

Many mining software packages allow the creation of subblocks, smaller than the parent or regular blocks, as a means of improving the resolution of the model at geological boundaries. Alternatively, the software may use only regular blocks but also record the proportion of a block lying within a specified geological zone. These techniques are designed to maximise the resolution of the in situ boundaries of the mineralisation in the Mineral Resource model.

In large open pit mines the use of front end loaders and large shovels means that it is unnecessary to maintain subblocks in the vertical direction, since the mining equipment effectively mines the full bench height. For mine planning it is necessary to reblock the subblock models to blocks of regular size which match the mining bench height. This reblocking process is known as regularisation. Ideally, the regularisation would reblock the model to a block size that represents the mining selectivity. Thus, subblocks of less than the bench height that were used to represent the dip of the orebody would be combined to reflect the geometry and size of the bench height.

The regularisation process creates blocks that cut across the ore-waste boundaries, thus adding dilution to the ore. This also drives some of the regularised blocks below the cut-off grade and these become ore loss and are removed from the Ore Reserve estimate.

The advantages of this approach are:
- the method is intuitive and visually satisfying to the mining engineer;
- it is easy to implement;
- the dilution is applied locally, no global factors are applied;
- it works equally well for multiple grade variables, including the grades of deleterious components;
- it is particularly suited to large disseminated deposits;
- it locally reflects the interaction of the mining bench height with the width and dip of the orebody; and
- it creates a fully diluted mining model that can be directly used for planning purposes.

The disadvantages are:
• Reblocking can only be applied where small block sizes or subblocks have been used to represent the geology.
• The method commonly does not work well for deposits with sharp ore boundaries which are amenable to some visual control during mining, as it tends to add too much dilution.
• The amount of dilution is uncontrolled since it depends only on the position of the ore boundary within the regular block. So, some blocks are highly diluted, some moderately diluted and some blocks are not diluted at all. Changing the origin of the block model or the regular block size changes the dilution in each block locally, although similar results are expected globally.

Functions to reblock subblock models are a common feature of mining software. At least one mining software package implemented a sophisticated variation of this method which allows subblocks to be aggregated according to a set of rules that define the SMU size, the mining direction and the limits of acceptable dilution to meet product specifications. This algorithm was developed to address the problems of mining bulk commodities (such as iron ore) to meet strict multi-element product specifications.

**Controlled regularisation**

For stratiform or strongly lithologically controlled deposits where there is potential for good visual control of mining at the ore contact but where some allowance for internal and external dilution is still required, standard regularisation may add too much dilution at the edges because regularisation follows orthogonal axes that are generally not parallel to the ore boundaries. Without additional control, standard regularisation underestimates the ability of the mining method to follow the strike, and to a lesser extent the dip, of the geological boundaries. For many stratiform orebodies mined in large open pits, such as iron ore deposits, the location of the ore-waste boundaries can be estimated accurately by geological mapping, logging of blast hole chips and/or geophysical logging. This information can then be used to plan the direction of mining that minimises ore loss and dilution within the wider practical constraints of access to mining benches, working area and haul road location.

When using controlled regularisation there are two decisions to be made. The first is the expected margin of error, when mining to the ore contact. This will be a combination of the uncertainty of the knowledge of the location of the boundary (Type 2 dilution), the uncertainty due to blast movement (Type 3 dilution), and mining errors (Type 4 dilution). The second decision will be whether it is better to accept ore loss or dilution. Accepting more ore loss reduces the dilution and vice versa. These two assessments are combined and expressed as the width of the zone of uncertainty, in metres.

Control of the regularisation is applied by expanding the boundary of the ore zone by the width of the margin of error, or zone of mixing of ore and waste, that is expected to occur during mining. The extra material included in the ore zone is the diluent. Several commercial modelling software packages provide functions to achieve this; however, it can also be carried out using normal mining software without any special modules. The model is then regularised but with the modification that the diluted grade and proportion of the ore block which lies within the expanded ore zone are recorded. In this way neither the recoverable tonnage nor grade are further diluted by the regularisation process.

Table 1 shows a simple one-dimensional example of the difference between conventional regularisation and controlled regularisation. Here a column of 2 m high blocks is regularised to create a number of 8 m benches. It has been assumed that there will be 1 m dilution added at the orebody contact on both edges and that a 0.2 per cent Cu cut-off will be used.

From Table 1, it can be seen that if the mineralised zones could be mined without dilution, then 18 m of ore at 0.90 per cent Cu could be achieved. Standard regularisation would give 24 m at 0.71 per cent Cu while controlled regularisation would give 22 m at 0.76 per cent Cu.

![Table 1](image-url)

**TABLE 1**

Controlled regularisation.

<table>
<thead>
<tr>
<th>Block height (m)</th>
<th>Grade Cu%</th>
<th>Non-diluted</th>
<th>Regularised</th>
<th>Controlled</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>Ore (m)</td>
<td>Grade Cu%</td>
<td>Ore (m)</td>
</tr>
<tr>
<td>2</td>
<td>0.07</td>
<td></td>
<td></td>
<td>1</td>
</tr>
<tr>
<td>2</td>
<td>0.07</td>
<td></td>
<td></td>
<td>2</td>
</tr>
<tr>
<td>2</td>
<td>1.00</td>
<td>2</td>
<td>0.70</td>
<td>2</td>
</tr>
<tr>
<td>2</td>
<td>0.80</td>
<td>2</td>
<td>0.80</td>
<td>8</td>
</tr>
<tr>
<td>2</td>
<td>0.15</td>
<td></td>
<td></td>
<td>2</td>
</tr>
<tr>
<td>2</td>
<td>0.70</td>
<td>2</td>
<td>0.70</td>
<td>2</td>
</tr>
<tr>
<td>2</td>
<td>0.90</td>
<td>2</td>
<td>0.90</td>
<td>2</td>
</tr>
<tr>
<td>2</td>
<td>1.20</td>
<td>2</td>
<td>1.20</td>
<td>8</td>
</tr>
<tr>
<td>2</td>
<td>1.30</td>
<td>2</td>
<td>1.30</td>
<td>2</td>
</tr>
<tr>
<td>2</td>
<td>0.90</td>
<td>2</td>
<td>0.90</td>
<td>2</td>
</tr>
<tr>
<td>2</td>
<td>0.60</td>
<td>2</td>
<td>0.60</td>
<td>2</td>
</tr>
<tr>
<td>2</td>
<td>0.15</td>
<td>2</td>
<td>0.15</td>
<td>8</td>
</tr>
<tr>
<td>2</td>
<td>0.07</td>
<td></td>
<td></td>
<td>1</td>
</tr>
<tr>
<td>2</td>
<td>0.06</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>2</td>
<td>0.08</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td><strong>Totals</strong></td>
<td><strong>18</strong></td>
<td><strong>24</strong></td>
<td><strong>8</strong></td>
<td><strong>22</strong></td>
</tr>
<tr>
<td><strong>Metal</strong></td>
<td><strong>16.20</strong></td>
<td><strong>17.00</strong></td>
<td><strong>16.75</strong></td>
<td><strong>16.75</strong></td>
</tr>
</tbody>
</table>
The advantage of the controlled regularisation method is that the amount of dilution that is added is related directly to the interpreted geology. The dilution is therefore based not only on the local grades of waste but also on the local geometry of the ore boundaries. The estimates can be easily fine-tuned if experience shows that the estimated width of the zone of uncertainty or mixing needs to be altered, for example if blast movement is greater than initially predicted. The regularised model is in a format which is ideally suited to open pit optimisation and mine scheduling.

EXAMPLE – A LARGE MAGNETITE DEPOSIT

The controlled regularisation method has been applied to the evaluation of mineable ore zones in a large magnetite deposit. Ore-grade magnetite mineralisation occurs in two stratigraphic units separated by thin horizons of subgrade sediments (Figure 1). The deposit occurs in a syncline. Drill core indicates that the ore grade units are easily visually identifiable. The mineralisation is evaluated in terms of the whole rock Fe, SiO2 and Al2O3 grades, the proportion recovered by magnetic separation using a Davis Tube (DTR per cent, expressed as mass per cent), and the Fe, SiO2 and Al2O3 grades of the magnetic concentrate (DTFe per cent, DTSi per cent, etc).

Controlled regularisation was achieved by expanding the wireframe triangulation outlines of the mineralised zone into the diluting areas by 1.0 m. The model was then reblocked to 10 m E x 12.5 m N x 12 m Z blocks. This allowed a controlled amount of dilution to be added.

Table 2 compares the in situ resource estimate and the diluted ore estimate at a DTR cut-off of 20 per cent. The results show a predicted net reduction in ore tonnage, after dilution of about 2.7 per cent. Fe grade is reduced by only 0.1 per cent and SiO2 and Al2O3 grades are increased due to the dilution. The loss of resource tonnage is due to dilution dropping some material below the cut-off grade. Mining loss has to be applied in addition to the dilution discussed above.

EXAMPLE – A COPPER DEPOSIT

Controlled regularisation was applied to a large copper deposit, consisting of two bodies of stratiform mineralisation. Mine production was planned to be by hydraulic shovels on 8 m benches.

The geological block model was diluted by expanding the ore zone wireframe models by 1.0 m. The block model was then converted to create two diluted mining models, firstly by regularising the blocks to 12.5 m x 12.5 m x 8 m and secondly by regularising the model to 12.5 m x 12.5 m x 4 m. These two models represent different mining bench heights or different mining selectivities. So although the 4 m high model might be mined on 8 m benches normally, dozing or flitching at the ore contacts might be used giving a greater selectivity. The original parent blocks were 25 m x 25 m x 4 m. However, as the estimated subblock grades were the same as the parent block grades for each domain there was no conditional bias introduced by the regularisation.

Table 3 compares the in situ resource and the diluted mineable ore at a cut-off grade of 0.2 per cent Cu for the two models. Absolute figures and factors are shown.

---

**TABLE 2**

<table>
<thead>
<tr>
<th></th>
<th>In situ resource</th>
<th>Diluted mining resource</th>
<th>Difference (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Tonnage (Mt)</td>
<td>1883</td>
<td>1832</td>
<td>-2.7</td>
</tr>
<tr>
<td>Fe%</td>
<td>35.94</td>
<td>35.90</td>
<td>-0.1</td>
</tr>
<tr>
<td>SiO2%</td>
<td>43.86</td>
<td>43.90</td>
<td>0.1</td>
</tr>
<tr>
<td>Al2O3%</td>
<td>1.12</td>
<td>1.14</td>
<td>1.7</td>
</tr>
<tr>
<td>DTR%</td>
<td>43.62</td>
<td>43.50</td>
<td>-0.3</td>
</tr>
<tr>
<td>DTFe%</td>
<td>69.99</td>
<td>69.96</td>
<td>0.0</td>
</tr>
<tr>
<td>DTSi%</td>
<td>1.94</td>
<td>1.96</td>
<td>0.9</td>
</tr>
</tbody>
</table>

*Comparison of geological and mining models.*

---

**TABLE 3**

<table>
<thead>
<tr>
<th>Model</th>
<th>Tonnage (Mt)</th>
<th>%Cu</th>
<th>Metal (kt)</th>
</tr>
</thead>
<tbody>
<tr>
<td>In situ resource</td>
<td>161.7</td>
<td>0.89</td>
<td>1438</td>
</tr>
<tr>
<td>Mine model (12 x 12 x 8)</td>
<td>181.7</td>
<td>0.79</td>
<td>1435</td>
</tr>
<tr>
<td>Mine model (12 x 12 x 4)</td>
<td>176.9</td>
<td>0.81</td>
<td>1433</td>
</tr>
<tr>
<td>Factors – Mine model/ in situ resource</td>
<td>Tonnage</td>
<td>Grade</td>
<td>Metal</td>
</tr>
<tr>
<td>Mine model (12 x 12 x 8)</td>
<td>1.12</td>
<td>0.89</td>
<td>1.00</td>
</tr>
<tr>
<td>Mine model (12 x 12 x 4)</td>
<td>1.09</td>
<td>0.91</td>
<td>1.00</td>
</tr>
</tbody>
</table>

*Comparison of mining and in situ models for a copper deposit.*
The results show that although the same assumption has been made about the width of the zone of mixing or uncertainty, the amount of dilution and ore loss are predicted to be lower for the more selective case. This confirms that the method produces results that are consistent with intuitive expectation. The particular combination of cut-off grade, ore grades, diluent grades, etc has resulted in the total amount of metal in the mining models being essentially the same as in the in situ resource.

ORE LOSS OR DILUTION?

As noted earlier, a mine has to decide whether it is better to accept ore loss or dilution. For a constant SMU size, accepting more ore loss reduces the dilution and vice versa. In general, the higher the value of the ore relative to the value of the diluent material, the more attractive it will be to minimise ore loss and, in doing so, accept higher dilution. The strategy may change from time to time depending on a range of factors including commodity prices, availability of equipment, effectiveness of grade control techniques, orebody variation, mining capacity and mill capacity. Powerful geostatistical techniques, such as conditional simulation, are available to evaluate dilution and ore loss under different assumptions about mining method, selectivity, and the preference for targeting grade or metal recovery (Khosrowshahi and Shaw, 2001). The results from studies using these techniques can be used to improve the calibration of mining models.

The use of actual data to check ore loss and dilution predictions is an essential but commonly poorly executed aspect of mining. Data for production reconciliation is relatively easily collected via grade control, survey and production monitoring systems. Reconciliations should be carried out at appropriate time intervals to check the performance of the resource and mining models and to provide a scientific basis for adjusting the procedure for estimating ore loss and dilution.

CONCLUSION

In many large open pit mines correct estimation of dilution and ore loss is critical to the successful implementation of a mine plan. These mines are driven by economies of scale and their high production rates and large equipment may limit their ability to respond to unforeseen dilution problems. Block models of in situ resources are generally enhanced by the use of subblocking techniques to improve resolution of the ore boundaries but for practical mine planning activities, such as open pit optimisation and mine scheduling, regularised block models are preferred, if not essential. Estimates of dilution and ore loss are commonly introduced into mining block models as part of the process of regularisation of the in situ geological model; however, the standard approach results in estimates that are poorly related to the local geology and do not satisfactorily quantify dilution or the local diluent grade. The method of controlled regularisation described in this paper uses the geological wireframe models to overcome these problems. It can provide more geologically controlled estimates of dilution.

REFERENCES


