Reconciliation principles for the mining industry

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Reconciliation involves the collection of tonnage, grade (quality) and contained metal (product) data from disparate and hopefully independent sources. Examples are exploration data, production sampling data from blast holes or draw points, and process plant data. These data may be compared by means of ratios (factors). The F1 factor usually relates short term (ore control) model tonnages, grades and metal content to ore reserves depleted. The F2 factor usually relates received at mill (measured by the mill) tonnages, grades and metal content to delivered to mill production tonnages, grade and metal content. The F3 factor is F1 $\times$ F2 and enables a comparison of a mine's (measured by mine) ability to recover the tonnage, grade and metal content estimated in ore reserves. The F1 factor measures the accuracy of orebody knowledge in the ore reserves to the demarcation of ore and waste by ore control (short term model). The F1 factor may be used to check and calibrate the selectivity of mineral resource models and/or planned dilution assumed in transfer from mineral resources to ore reserves. The F2 factor enables a check on unplanned dilution entering the ore stream between ore control and the mill. By using the factors it is possible to calculate a monetary value on improvements in the accuracy of orebody knowledge, selectivity and the effects of dilution and ore loss. Reconciliation should be an implicit part of the mining process, and reconciliation targets should be a key performance indicator for well run mines.

Keywords: Reconciliation, Tonnage, Grade, Contained metal data, Factors, Ratios

Introduction

This paper is an extension to Parker (2006) and builds on over 25 years of practical reconciliation experience gained in copper, gold, nickel and iron mining operations located throughout the world. Reconciliation is a key process that allows determination of the ability of a mining operation to produce the tonnage, grade and contained metal that were estimated in the ore reserve. This paper discusses the principles of reconciliation and the commonly encountered sources of error in reserve, ore control and mill head estimates by comparing the data collected by means of ratios or factors. Before developing the principles, it is necessary to see why reconciliation is important. A simple example is shown in Table 1, which shows a production schedule and cash flow statement for a copper mine, where the reserves are estimated to grade 0·6% copper.

One of the goals of ore reserve estimation is to make an unbiased estimate of the tonnage and grade. This is seldom true in practice. Errors will occur, and these will be related to inaccurate orebody knowledge at the time of ore reserve estimation and to the assumptions made as to the accuracy with which ore is selected from waste in the ore control process. It is generally accepted that in a base metals mine, a good annual reconciliation between mine and mill would be $\pm 5\%$, and for a precious metals mine, a good annual reconciliation between mine and mill would be $\pm 10\%$. Even so, it is useful to examine the impact of such seemingly small estimation errors on the cash flows and net present values (NPVs) of an operation. This is shown in Table 2.

The change in cash flow is three times that in grade; the change in NPV is six times that in grade. A small negative error can cause a mine to miss its cash flow targets. If these persist, unfavourable press releases must be issued to the marketplace. For longer term planning, mining companies weigh the NPVs of various alternatives when deciding to make acquisitions or determining the sequence in which they will develop their assets. Errors may result in incorrect decisions, and may even result in premature mine closure.

To understand and avoid sources of error, it is useful to gather reconciliation data at various stages of the mining process. To be valid, the reconciliation data taken for each stage should be independent, and generally, both tonnage and grade should be estimated or measured. The collected data are analysed in terms of ratios, herein called factors.

Sources of error

In most mines there are two main sources of error that will be considered here:

(i) inaccuracy in estimation of mineral resources and/or ore reserves (long range model)
(ii) inefficiency in the mining process to segregate ore and waste as planned (short range model) by the ore control staff.

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Table 1 Production schedule and cash flow statement for copper mine

Input variable																			
Mining ore/$/t		2·00																	
Mining waste/$/t		2·00																	
Milling/$/t		5·00																	
General and admin./$/t		4·00																	
Copper price/$/lb		2·50																	
Conc. Grade/%Cu		29·0																	
Conc. recovery/%		90																	
Freight/$/t Conc.		50																	
Smelting/$/t Conc.		80																	
Refining/$/lb		0·08																	
Payable Cu/%		95																	
Pounds/t		2204·62																	
Year	**−2**	**−1**	**0**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**
Production schedule																			
Ore mined/Mt				10	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20
Waste mined/Mt			10	20	40	40	40	40	40	40	40	40	40	40	40	40	40	40	40
Ore grade/%Cu				0·60	0·60	0·60	0·60	0·60	0·60	0·60	0·60	0·60	0·60	0·60	0·60	0·60	0·60	0·60	0·60
Concentrate/Mt				0·186	0·372	0·372	0·372	0·372	0·372	0·372	0·372	0·372	0·372	0·372	0·372	0·372	0·372	0·372	0·372
Payable Cu/Mt				0·051	0·103	0·103	0·103	0·103	0·103	0·103	0·103	0·103	0·103	0·103	0·103	0·103	0·103	0·103	0·103
Cash flow statement/M$																			
Revenues				282·7	565·5	565·5	565·5	565·5	565·5	565·5	565·5	565·5	565·5	565·5	565·5	565·5	565·5	565·5	565·5
Ore mining				−20·0	−40·0	−40·0	−40·0	−40·0	−40·0	−40·0	−40·0	−40·0	−40·0	−40·0	−40·0	−40·0	−40·0	−40·0	−40·0
Waste mining				−40·0	−80·0	−80·0	−80·0	−80·0	−80·0	−80·0	−80·0	−80·0	−80·0	−80·0	−80·0	−80·0	−80·0	−80·0	−80·0
Milling				−50·0	−100·0	−100·0	−100·0	−100·0	−100·0	−100·0	−100·0	−100·0	−100·0	−100·0	−100·0	−100·0	−100·0	−100·0	−100·0
G+A				−40·0	−80·0	−80·0	−80·0	−80·0	−80·0	−80·0	−80·0	−80·0	−80·0	−80·0	−80·0	−80·0	−80·0	−80·0	−80·0
Freight				−9·3	−18·6	−18·6	−18·6	−18·6	−18·6	−18·6	−18·6	−18·6	−18·6	−18·6	−18·6	−18·6	−18·6	−18·6	−18·6
Smelting				−14·9	−29·8	−29·8	−29·8	−29·8	−29·8	−29·8	−29·8	−29·8	−29·8	−29·8	−29·8	−29·8	−29·8	−29·8	−29·8
Refining				−9·0	−18·1	−18·1	−18·1	−18·1	−18·1	−18·1	−18·1	−18·1	−18·1	−18·1	−18·1	−18·1	−18·1	−18·1	−18·1
Capex	−200·0	−200·0	−200·0	−100·0	−30·0	−30·0	−30·0	−30·0	−30·0	−30·0	−30·0	−30·0	−30·0	−30·0	−30·0	−30·0	−30·0	−30·0	0·0
Pretax cash flow	−200·0	−200·0	−200·0	−0·5	169·0	169·0	169·0	169·0	169·0	169·0	169·0	169·0	169·0	169·0	169·0	169·0	169·0	169·0	199·0
Cumulative cash flow	−200·0	−400·0	−600·0	−600·5	−431·5	−262·6	−93·6	75·4	244·4	413·3	582·3	751·3	920·3	1089·2	1258·2	1427·2	1596·2	1765·1	1964·1
Net present value (NPV) at 8%	554·3																		

Table 2 Sensitivity of cash flow, net present value and payback to changes in grade (0·60%Cu is base case)

%Cu	%Change	Cash flow year 2/M$	%Change	NPV at 8%	%Change	Payback years
0·66	10	218·9	30	886·5	60	4
0·63	5	193·9	15	720·4	30	4
0·60	**0**	**169·0**	**0**	**554·3**	**0**	**5**
0·57	−5	144·0	−15	388·1	−30	6
0·54	−10	119·1	−30	222·0	−60	7

The inaccuracy in estimation of mineral resources and/or ore reserves is determined by comparing depletions from the long range model to the short range model.

Inefficiency in the mining process is determined by comparing received at mill tonnages, grade and metal content based on mill tonnage measurements and head assays versus delivered to mill tonnages, grade and metal content based on the short range (ore control) model.

There are other sources of error listed below that can be important but will not be considered:

(i) failure to mine within planned areas
(ii) failure to mine at the planned cut-off
(iii) plant performance.

There may be very good mitigating circumstances, such as change in markets or physical impediments (slides, stope/drawpoint collapse), but in the long run mines that fail to follow the plan tend to find themselves with a shortfall in stripping or remnants in ore reserves that have to be written off.

Reconciliation within treatment plants has been discussed at length by Morrison (2008).

Reconciliation between long range model and short range model

The long range model is constructed using geological interpretations based on exploration and/or delineation drill holes. Assays from these holes are composited and used to estimate grades in blocks. The result is a resource model. In massive deposits with broad trends to grade, a pit is developed using mining software and refined and scheduled. The cut-off grade may be changed over the course of the schedule using the theory of Lane (1988). In such deposits, the ore reserve is often the tonnage of measured and indicated mineral resources within a pit.

In deposits with sharp decreases in grade at ore/waste contacts, the resource model may be adjusted to allow for dilution and less frequently ore loss at the contact. These adjustments may take the form of aggregating small blocks into large blocks or adjusting the grades of ore blocks by a given amount at their faces with waste blocks. In underground mines it is common to design stopes and to apply dilution and recovery factors. Where block caving will be used, complex software such as PC-BC is often used to make adjustments for various amounts of mixing in the cave profile.

The short range model is commonly constructed using samples from closely spaced drill holes and/or mapping information. This information is used to delineate ore and waste in mineable shapes. In an open pit mine the shapes are staked in the field, sometimes with adjustment for blast heave (Yennamani *et al.*, 2011; La Rosa and Thornton, 2011). In an underground mine, cross-cuts, channel/chip samples and short drill holes (commonly drilled on 15 m spacings) are used to make the final stope layouts. Where caving methods are used, drawpoint samples and observations of lithology are used to decide when to shut off drawpoints.

The ratios between tonnage, grade and metal in the long range and short range models are used to develop reconciliation factors, known as F1*, as F1 (tonnes), F1 (grade) and F1 (metal)

$$\mathrm{F1} = \frac{\text{short range model depletions}}{\text{long range model depletions}}$$

Where a deposit has more than one grade variable (such as Cu and Au), then there will be multiple F1 factors for grade [F1 (%Cu), F1 (Au g/t)] and for metal [F1 (contained Cu), F1 (contained Au)]. In some deposits F1 factors are developed for deleterious elements. Sometimes a deposit will produce several products. In these cases, F1 factors are constructed for each product.

A high value for an F1 factor indicates conservatism in the long range model; conversely a low value for an F1 factor indicates optimism in the long range model. Table 3 describes sources of error and likely resultant F1 factors.

As can be seen in Table 3, the sources of error can be ambiguous. For example, a low F1 (tonnes) could be just as likely to be related to an over smoothed long range model as to conservative dig lines creating remnants. In general, common sense should prevail; where the drill hole spacing is too wide to identify discrete ore zones, the long range model will be inaccurate. More drilling is needed to support the long range model. This may not occur everywhere in the deposit, but only in specific areas (e.g. lithological contacts, structures). In addition, consideration should be given to whether the mine is undergoing stress, as for instance recovering from a geotechnical event such as a slide after which there are fewer than normal working areas available.

Reconciliation between delivered to mill and received at mill

Most mines have transit stockpiles where ore is stored temporarily. Therefore, material leaving the pit and being depleted from the short range model can be delivered to a stockpile or to the mill (at its short range model grade). Generally, stockpiles are assigned the average grade of all the increments added to them. If the stockpiles are reclaimed and rebuilt fairly regularly (such as to allow for a bad weather season with low productivity from the pit), then this should not significantly affect reconciliation much, particularly if the quantities represented in the numerators and denominators of the factors have been aggregated over quarters or years. Reconciliation in the cases where old/large stockpiles are

*The F1, F2 and F3 factors and the terms 'delivered to mill' and 'received at mill' come from the Nchanga Open Pits survey department, Chingola, Zambia.

reclaimed or where a large percentage of production is related to stockpile reclaim can be problematic.

At larger mines there is often either no mill head sampler or a recirculation/mixing of materials within the mill that prevents a head sample from being taken. This means that the head grade is calculated from the tailings and concentrate tonnages and grades. Concentrate tonnages and grades are usually accurately measured. The tonnage and grade of the tailings stream are sometimes not reliable, which can affect the calculated head grade. The weighing devices used to measure the received at mill tonnage can get out of calibration; in addition moisture is normally backed out of the tonnage.

The ratios between tonnage, grade and metal content received at mill to delivered to mill are used to develop reconciliation factors, known as F2, as F2 (tonnes), F2 (grade) and F2 (metal)

$$F2 = \frac{\text{received at mill}}{\text{delivered to mill}}$$

Delivered to mill is the combination of short term model depletions expit direct to mill and stockpile reclaim at

Table 3 Sources of error and likely F1 factors

Source of error	F1 (tonnes)	F1 (grade)	F1 (metal)	Comments/ remedial action
Mainly related to long range model				
Conservative grade shells, ore bearing lithologies	>1	<1	>1	Usually the tonnage effect is greater than the grade effect, giving a positive F1 (metal). At early stages (wide spaced drilling), conservatism may be justified, particularly for inferred resources. Projection of grade shells and favourable lithologies on two sets of orthogonal sections and then to plan may help; Also, recognition of contacts, structural, lithological or alteration controls may help in projecting further in sparsely drilled areas. The use of grade shells may cause distortions in the grade–tonnage curve. These can be mitigated by sample sharing at boundaries. In the end, more drilling is likely needed to upgrade orebody knowledge. Will increase ore tonnage in the long range model.
Over smoothed long range model	<1	>1	About same	This can be a very serious error when processing costs will be high. The long range model incorporates too much dilution, which ore control can segregate, with very favourable financial consequences. A stop gap solution is to interpolate grade using fewer samples or a high power inverse distance method. This can have a bad knock-on effect of local conditionally biased estimates. It is better to use deterministic or probabilistic domaining or a Uniform Conditioning/MIK method. However, the best solution is to tighten the drill spacing
Under-smoothed long range model	>1	<1	About same	Domains and grade shells are too selective. Consider some mixing of samples across domain boundaries with outlier restriction or topcutting to control over projection of grades.
Grade effects	About same	>1	>1	Where grade distributions are very skewed, wide spaced drilling will undersample the high end of the distribution. In diamond deposits, large stones may be broken. If core recovery is poor, may underestimate grade, examples are chalcocite, molybdenite and gold. Plucking of soft materials may also result in underestimation of grade.
Reverse circulation drilling	About same	>1, <1	>1, <1	In dry RC, fines may not be recovered; this may affect grade. For wet RC, fines may be washed out; gold may be concentrated (placering). Weak high grade zones may contaminate samples from above. Drilling practices must be improved with mud conditioning, flocculants in buckets, better splitting, or switching to drilling of core (deep holes).
Mainly related to short range model				
Sloppy dig line layout	>1	<1, >1	About same	Effective SMU is larger than plan. Very common for low grade dump leach or stockpile categories with a few erratic high grade values. Better ore control sampling (such as every blast hole versus one in six) can help. Adding short delineation holes may help. Improve quality of face or blasthole sampling (increase size of sample taken). Decrease block size
Biased sampling	About same	>1	>1	Over sampling of fines in blast hole or grab samples can cause a bias
Leaving remnants	<1	>1	<1	Purposely conservative final stope design in times of economic stress

average grade based on short term model increments placed on the pile.

Where a deposit has more than one grade variable (such as Cu, Au), then there will be multiple F2 factors for grade [F2 (%Cu), F2 (Au g/t)] and for metal content [F2 (contained Cu), F2 (contained Au)]. In some deposits F2 factors are developed for deleterious elements.

Sometimes a deposit will produce several products. In these cases, F2 factors are constructed for each product.

A high value for F2 will indicate conservatism in the short term model. A low value for F2 will indicate optimism in the short term model.

In the case of underground mines, stockpiles are not often used, but remuck bays provide temporary storage and an opportunity for mixing of ore and waste.

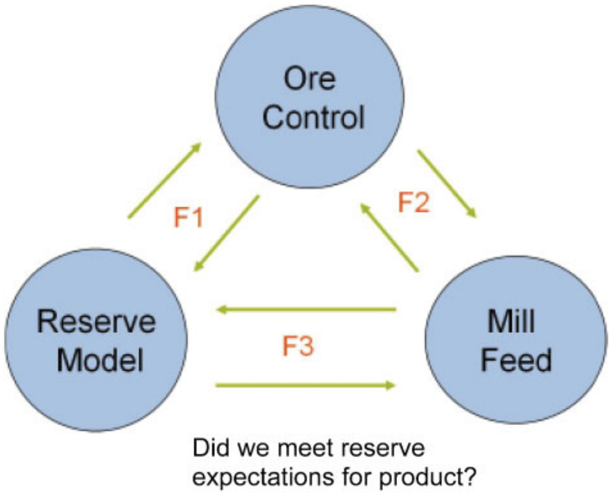
Table 4 describes sources of error and likely F2 factors.

With the advent of GPS control and automated dispatch systems, F2 factors are now generally close to unity in most open pit mines. There are still problems with mixing of ore and waste in underground mines, particularly where development is behind schedule. It is just too easy to stash waste development tonnage into the ore delivery system to the mill.

Sometimes it is preferable to overbreak contacts to ensure against ore loss; however, this should be planned and thus would give a high F1 (tonnes) factor and low F1 (grade) factor. Properly done, there would be no effect of this practice on F2 factors.

Reconciliation between mill and long range model

The long range model (sometimes after adjustment for planned ore loss and dilution) is the basis for the life of mine plan and through it cash flow forecasts. An F3 factor can be created which allows for the effects of stockpiles



1 Summary relationships between factors

$$F3 = \frac{\text{short range model depletions}}{\text{long range model depletions}} \times \frac{\text{received at mill}}{\text{delivered to mill}} = F1 \times F2$$

Over a period of time (annual) short range model depletions will cancel with delivered to mill.

One of the advantages of the F3 metric is that it removes the effect of ore control sampling (grade) bias, which often afflicts ore control.

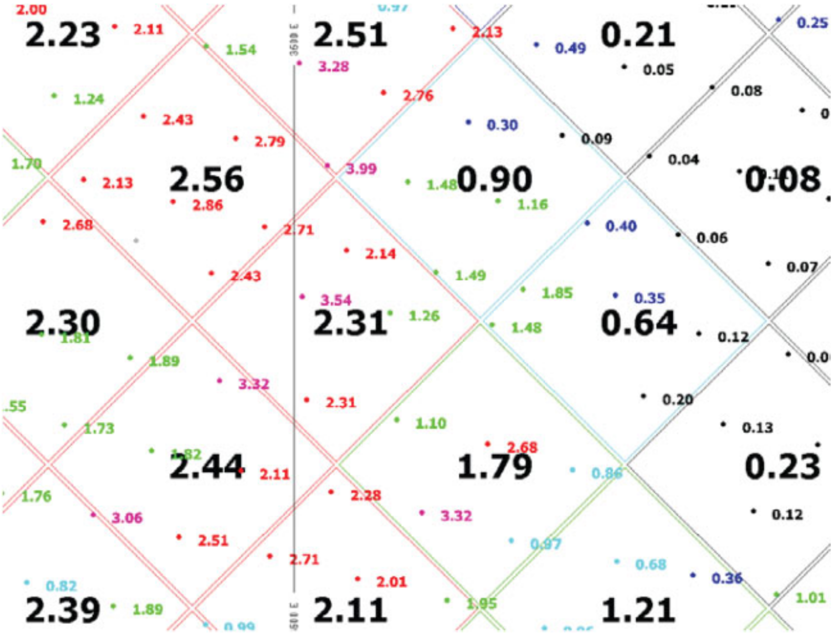
Figure 1 shows a summary depiction of the factors and their relationships.

Remarks

It is becoming more frequent to use the F1, F2 and F3 frameworks at the feasibility stage to test or adjust the long range model to reflect the intended degree of

Table 4 Sources of error and likely F2 factors

Source of error	F2 (tonnes)	F2 (grade)	F2 (metal)	Comments/remedial action
Mainly related to short range model				
Biased sampling	About same	<1	<1	Over sampling of fines in blast hole or grab samples can cause a bias
Truck allocation	Either high/low	>1, <1	>1, <1	Miscount on trucks delivering to mill, stockpiles
Mainly related to poor mining practice				
Overbreak	>1	<1	About same	Mining beyond dig lines; following colour changes in muck unrelated to ore control boundaries; bonus scheme should be based on metal, not tonnes broken
Blast heave, unplanned mixing of ore and waste	About same	<1	<1	Blast ore and waste in opposite directions; separate ore and waste passes, remuck bays; keep waste out of crusher; use modern dispatch system
Pulling too fast in block cave mines	<1	<1	<1	Piping at drawpoints; drawpoints close early due to low grade factors
Mainly related to received at mill				
Bad tonnage	Either high/low	No effect	No effect	Belt scales not calibrated; moisture is guestimated. Calibrate scales every two weeks; obtain weekly moisture samples; determine seasonal corrections
Bad grade	No effect	<1, >1	<1, >1	Usually F2 (grade) <1; need better tailings tonnage and grade measurement and samplers
Multiple products				
	<1, >1	<1, >1	<1, >1	Mill campaigns may not coincide with ore types as segregated in pit; better communication can help



2 Blast blocks (black) and blast hole (coloured) grades (%Cu)

selectivity. Conditional simulation was used by Edward Isaaks as early as 1991 to adjust an over smoothed kriged model for the Lihir gold deposit, located in Papua New Guinea (Parker, 1992a). Table 5 shows an example of adjustment factors applied for various selective mining unit (SMU) sizes. The corrections for smoothing in the long range model are more severe as the SMU size is decreased.

This was repeated in 1994 for the Fort Knox Gold deposit in Alaska. For both Lihir and Fort Knox, a very fine grid of grades was simulated at ~3 m spacing, and these were taken as ground truth. A subset was taken to represent blast holes; ore and waste diglines were drawn manually, and the ore control grade was estimated using kriging. The resultant ground truth grades within diglines were used to formulate received at mill grades. At Quebrada Blanca (Iquique, Chile), conditional simulation was used to determine the impact of over-breaking the leached cap contact with supergene enriched ore to minimise ore loss.

It is sometimes useful to break the factors down further. For example at Antamina Mine in Peru, a factor F1' was developed that compared the distribution of blast blocks with the long range model blocks. Blast blocks had the same dimensions as the long range model blocks (20×20×15 m). The ore control model (5×5×15 m blocks) was aggregated to blocks with dimensions of 20×20×15 m (see Fig. 2).

This enabled determination as to whether the selectivity implicit in the long range model (20×20×15 m SMU)

Table 5 Adjustment factors for Lihir (Parker, 1992a)

SMU		F3		
Size/m	Volume/m^3	Tonnes	Grade	Metal
6×6×2	72	0·78	1·18	0·92
6×6×4	144	0·82	1·12	0·91
12×5×3	180	0·84	1·09	0·91
9×9×6	486	0·87	1·05	0·91

was reflected in the short term model as measured by F1', and whether the 20×20×15 m SMU demonstrated the same degree of selectivity shown by ore control polygons (diglines) as measured by F1″. Table 6 shows the results as follows:

(i) for copper only ores the F1' factors show that the long range model is slightly underestimating tonnage and grade of blast blocks
(ii) for copper–zinc ores the F1' factors show that the long range model is underestimating tonnage and over estimating grade. The long range model is too selective
(iii) for copper only ores the F1″ factors show that ore control polygons have similar tonnage and grade to the blast blocks. The ore control polygons have a level of selectivity implicit in a 20×20×15 m SMU
(iv) for copper–zinc ores the F1″ factors show that the ore control polygons have much higher tonnage at the same copper, but much higher zinc grades. This was later found to be related to over projection of high grades in the short range model.

At BHP Billiton, the F3 factor is defined slightly differently. The contained metal/coal in saleable product (beneficiated iron ore, marketable coal, or metal contained in concentrates) for a time period is the numerator. The denominator is the forecast metal/coal recovered from the life of mine plan adjusted for stockpile additions and depletions.

Therefore, BHP Billiton is taking into account treatment plant performance as well. BHP Billiton also reports F1 and F2 factors as defined herein.

Sometimes the F1 factors are referred to as block factors which are related to the pregeostatistical era when factors were applied to polygonal and other simple ore reserve estimates. As an example at Jerritt Canyon Nevada in the 1980s, a nearest neighbour estimate was made of gold grades. For production scheduling, grades above cut-off were multiplied by 0·92, and tonnages were multiplied by 1·05 (Parker, 1992b).

Table 6 Detailed comparison of long range to short range models at Antamina Mine

Copper only ores

	Mtonnes	%Cu	%Zn	kt cont. Cu	kt cont. Zn
2001–2004					
Long term model*	66·8	1·49	0·25	993	165
Blast blocks†	71·1	1·54	0·23	1096	164
F1' factors	1·06	1·04	0·93	1·10	0·99
2003–2004‡					
Blast blocks	32·5	1·59	0·25	517	81
Ore control polygons	32·3	1·67	0·22	538	72
F1″ factor	0·99	1·05	0·89	1·04	0·89

Copper–Zinc ores

	Mtonnes	%Cu	%Zn	kt cont. Cu	kt cont. Zn
2001–2004					
Long term model*	46·7	0·98	2·94	457	1375
Blast blocks	55·4	0·96	2·66	532	1474
F1' factors	1·19	0·98	0·90	1·16	1·07
2003–2004‡					
Blast blocks	22·7	1·01	2·83	229	642
Ore control polygons	28·8	1·02	3·17	294	911
F1″ factor	1·27	1·01	1·12	1·28	1·42

*Long range model built in 2005 after infill drilling programme.
†Blast hole model reblocked to $20\times20\times15$ m blocks.
‡Comparison of blast blocks to ore control polygons for 2001–2002 is not shown because oxidised material was sent to waste based on pit mapping; this information was not preserved in blast blocks.

The F2 factors are also sometimes referred to as mine call factors. These have variously covered a myriad of issues such as mining recovery, unplanned dilution and ore loss.

Case study for Antamina Mine

Antamina Mine is located in the Peruvian Andes and extracts copper, zinc, molybdenum, silver and bismuth from endoskarns and exoskarns developed in limestones surrounding a porphyry intrusive. Copper-only ores are concentrated in endoskarns inboard from copper–zinc ores that are concentrated in exoskarns. Copper and zinc concentrates are produced and shipped by slurry pipeline to a port at Huarmey on the Pacific Ocean. Molybdenum and minor lead concentrates are produced as byproducts. Nominal ore production is ~30 Mtpa.

Reconciliation in 2003

Table 7 shows reconciliation between long range and short range models for the first to third quarters (Q1–3) for 2003. In this case the long range model was produced using feasibility-stage exploration drilling.

The F1 factors are far from unity. For the copper only ores occurring in endoskarns, the 75–100 m spaced exploration drilling undersampled narrow (5–20 m wide) zones of high grade breccias (see Fig. 3). This led to F1 (tonnes) of 1·31 and F1 (%Cu) of 1·22. For the copper–zinc ores, the resource model based on 50–75 m spaced exploration drill holes was smoothed, resulting in F1 (tonnes) of 0·87 and F1 (%Zn) of 1·20 (see Fig. 4). In these ores F1 (%Cu) was lower (0·86), and reflects lower copper grades in the higher grade zinc areas.

The reconciliation over the shorter time periods used to plan mill campaigns and concentrate shipments was unacceptable to mine management. An investigation showed that more closely spaced drilling would be required to delineate the high grade zones which were found to be discrete bodies within the deposit. The high grade breccias were delineated as a separate unit for copper grade estimation. The high grade zinc zones were found to occur near the marble/green exoskarn contact, but their variable width (20–40 m) was only able to be domained through grade shells interpreted from detailed drilling.

The mine staff and mine owners were in agreement that the drill hole spacing had to be reduced, but by

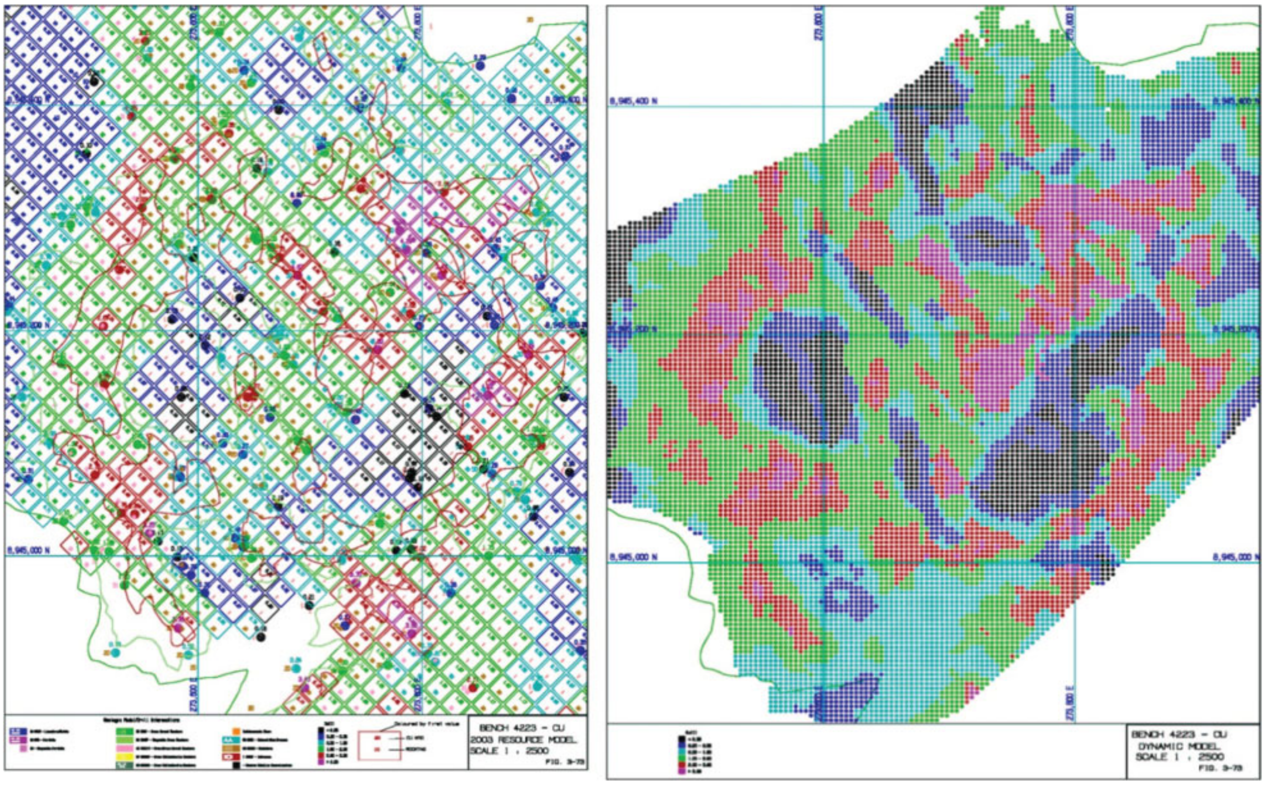
Table 7 Antamina Mine reconciliation between long range and short range models (Q1–3, 2003)

Copper only ores

	Mtonnes	%Cu	%Zn	kt cont. Cu	kt cont. Zn
Long range model	6·39	1·41	0·16	90·1	10·2
Short range model	8·41	1·73	0·16	145·5	13·5
F1 factors	1·32	1·23	1·00	1·61	1·32

Copper–zinc ores

	Mtonnes	%Cu	%Zn	kt cont. Cu	kt cont. Zn
Long range model	15·6	1·11	2·56	173·2	399·4
Short range model	13·5	0·95	3·06	128·3	413·1
F1 factors	0·87	0·86	1·20	0·74	1·03

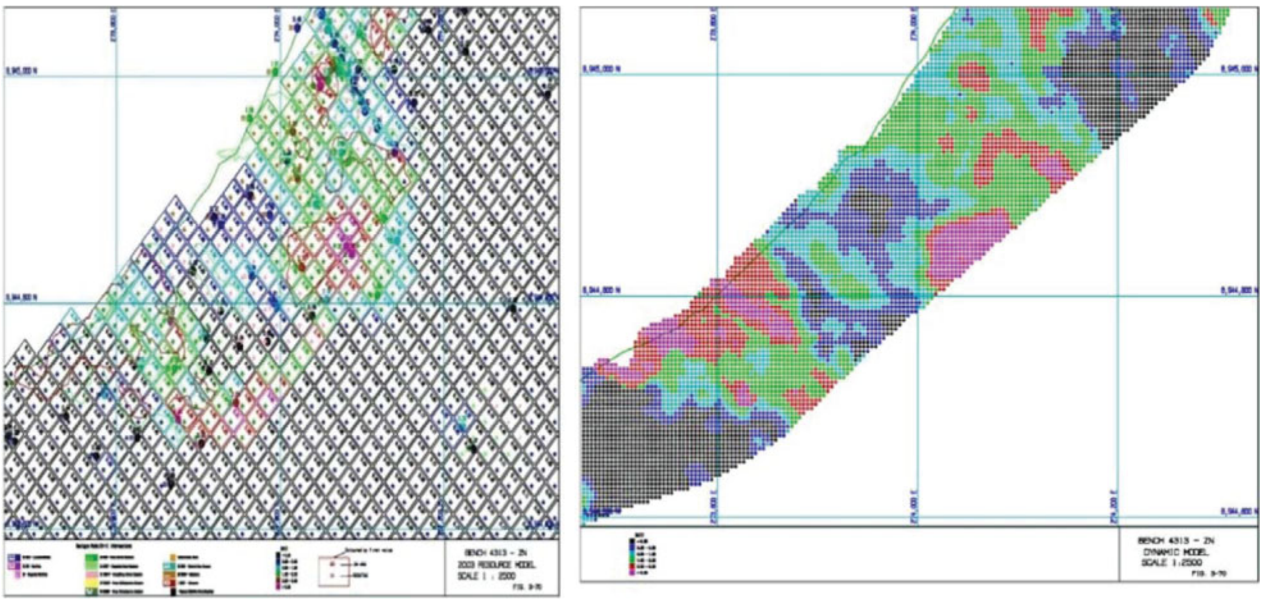


3 Antamina Mine 2003 long range Cu model (left) and short range model (right): red colour indicates >2%Cu, localised in breccia zones; blocks (left) are 20×20 m and 5×5 m (right)

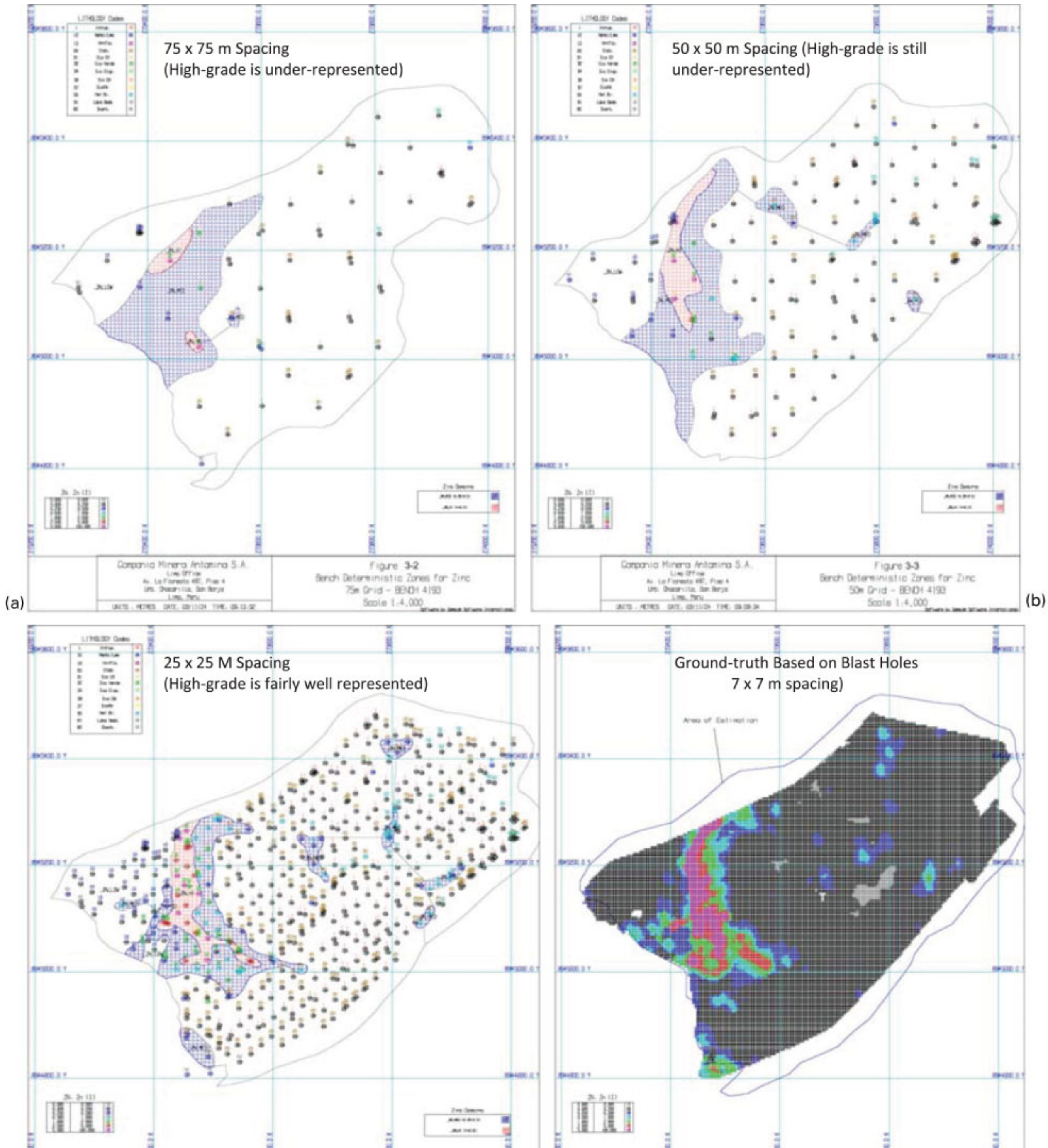
how much? Conditional simulation could have been used for this purpose, but was rejected as being too time consuming; both multiple grade variables (Cu and Zn) that were required to be estimated, as well as lithological contacts. Instead, subsets of blast holes representing about 60 Mt were used to create pseudo exploration grids, and these were in turn used to estimate experimental long range models. The experimental long range models were then compared with the short range model estimated using all the blast holes (and taken as ground truth). Since blast hole assays were used as the basis for both the experimental long range models and the short range model (ground truth), there was no noise introduced by using different sample types (such as drill cores versus auger samples of blast hole cuttings).

Figure 5 shows the results of three grids for zinc and the ground truth for comparison. The high grade zone does not become well delineated until a 25×25 m spacing is achieved.

Table 8 shows the change in bias as the drill spacing is reduced. The 1% cut-offs for copper and zinc represent typical cut-offs used in 2003 by ore control for mill ore. The 2·5% cut-offs were designed to evaluate the high



4 Antamina Mine 2003 long range Zn model (left) and short range model (right): red and magenta colour indicates >2·5%Zn, localised in structural zones; blocks (left) are 20×20 m and 5×5 m (right)



5 Zinc grade zones for three grids interpreted from experimental grids of blast holes: high grade zinc over 2·5% is red; medium grade zinc (0·25–2·5%) is blue: *a* 75×75 m spacing (high grade is under represented); *b* 50×50 m spacing (high grade is still under represented); *c* 25×25 m spacing (high grade is fairly well represented); *d* ground truth based on blast holes 7×7 m spacing

grade zones that contain about half the metal above cut-off. An F1 factor of <1·1 was deemed desirable to declare indicated mineral resources at the 1% cut-off. To declare measured mineral resources, it should be a goal to ensure that there are no biases at the high (2·5%) cut-off.

Examination of Table 8 shows:

(i) using a 1% copper cut-off, the F1 criterion for tonnage, grade and metal is achieved at a 50×50 m spacing

(ii) using a 2·5% copper cut-off, the F1 criterion for tonnage, grade and metal is achieved at a 50×25 m spacing

(iii) using a 1% zinc cut-off, the F1 criterion for tonnage and metal is not achieved for any spacing. The F1 criterion is met at a 50×25 m spacing for grade (%Zn)

(iv) using a 2·5% zinc cut-off, the F1 criterion is met for metal at the 25×25 m spacing. Tonnage is

Table 8 Antamina Mine F1 factor reconciliation between test long range and short range models (60 Mt)

Drill spacing/m	F1 (tonnes)	F1 (%Cu)	F1 (cont. Cu)
1% copper cut-off grade applied			
75×75	0·97	1·15	1·11
50×50	0·93	1·01	0·93
50×25	0·88	1·02	0·90
25×25	0·92	1·01	0·93
2·5% copper cut-off grade applied			
75×75	2·04	1·14	2·33
50×50	1·11	0·96	1·06
50×25	1·05	0·98	1·03
25×25	1·03	1·02	1·04

Drill spacing/m	F1 (tonnes)	F1 (%Zn)	F1 (cont. Zn)
1% zinc cut-off grade applied			
75×75	1·52	1·35	2·04
50×50	1·35	1·20	1·64
50×25	1·43	1·06	1·52
25×25	1·35	0·92	1·25
2·5% zinc cut-off grade applied			
75×75	3·70	0·69	2·56
50×50	2·22	0·79	1·75
50×25	1·37	1·00	1·37
25×25	1·18	0·88	1·04

under estimated, and grade is over estimated. This is an artifact of the grade zoning process, and is now ameliorated by sharing samples across the high grade zone boundary.

In 2004 the drill spacing was reduced from 100 to 50 m to support indicated mineral resource declaration and from 75 to 25–35 m to support measured mineral resource declaration. As copper is the primary metal of economic interest, it was decided that the greater uncertainty for zinc could be accommodated within these drill spacings.

Reconciliation after infill drilling

Approximately 100 000 m of infill drilling was performed in 2004. This brought nearly all the ore mined to the end of 2004 to measured mineral resource status. Table 9 shows the reconciliation that was achieved during 2003 and 2004.

For this comparison a new long range model was used; this model was constructed in 2005 after completion of infill drilling. For copper only ores the reconciliation is good. F1 (tonnes) and F1 (%Cu) are still higher than 1·0, but only marginally. There has been great improvement compared to the results shown in Table 8 for previous estimates. The F2 factors are <1·0. This likely indicates that some copper only ores were routed to copper–zinc campaigns by the mill, thus explaining F2 (tonnes) at 0·93. The mill head grade is lower than the short term model depletions [F2 (%Cu)=0·91]. This is likely evidence of high bias in blast hole sampling (meaning the auger method used for ore control samples oversamples fines). The F3 factors for tonnage, %Cu and contained copper are near parity (0·99–1·01).

For copper–zinc ores, the F1 (tonnes) and F1 (%Zn) are both greater than 1·0. The tonnage increase in short term model depletions compared to the long range model is either related to overbreaking the marble/exoskarn contact where high grade zinc occurs, or a conservative long range model (probable from Table 8). The former explanation is more likely. The received at mill grade is less than the short range model depletions (2·61 versus 3·21%Zn). Subsequent investigations revealed the short range model was over projecting high grade blast holes; this finding accounts for some of the observed bias.

Comments

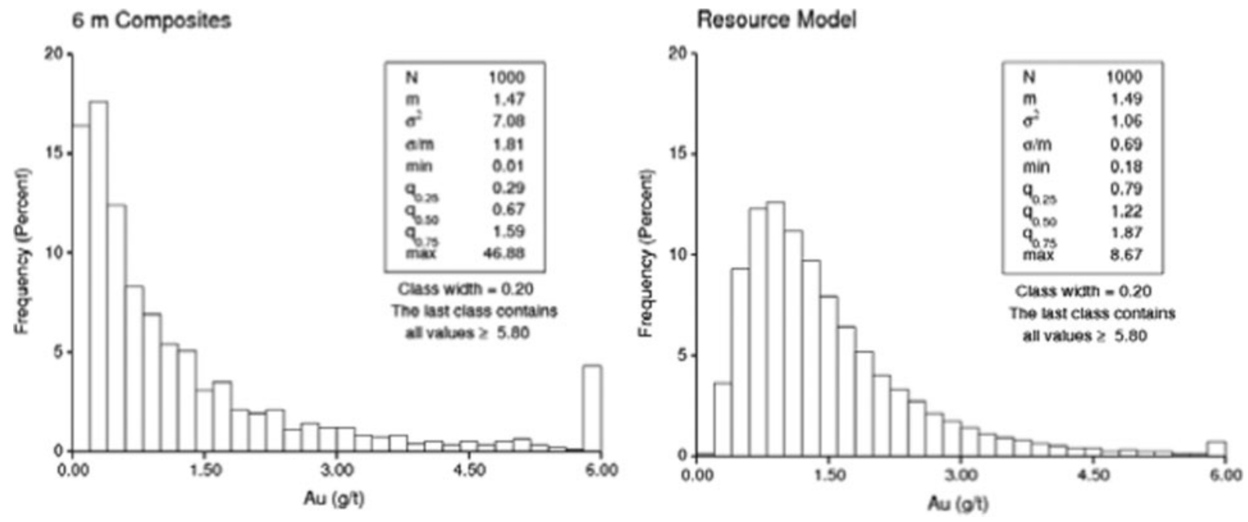
The reconciliation data were extremely useful:

(i) the F1 factors measured in 2003 identified a problem with the long range model

(ii) the blast hole data were used to demonstrate the improvement to be gained in the accuracy of the long range model by infill drilling

Table 9 Antamina Mine reconciliation for 2003–2004

	Mt	%Cu	%Zn	kt cont. Cu	kt cont. Zn
Copper only ores					
Long range model depletions	30·5	1·51	0·25	460·6	76·3
Short range model depletions	32·3	1·67	0·22	539·4	71·1
Received at mill	30·1	1·52	0·25	457·5	75·3
F1 factors	1·06	1·11	0·88	1·17	0·93
F2 factors	0·93	0·91	1·14	0·85	1·06
F3 factors	0·99	1·01	1·00	0·99	0·99
Copper–zinc ores					
Long range model depletions	23·9	0·96	3·04	229·4	726·6
Short range model depletions	27·1	1·07	3·21	290·0	869·9
Received at mill	27·6	1·01	2·61	278·8	720·4
F1 Factors	1·13	1·11	1·06	1·26	1·20
F2 Factors	1·02	0·94	0·81	0·96	0·83
F3 Factors	1·15	1·05	0·86	1·21	0·99



6 Histograms for composites and resource model

(iii) the F2 factors for 2003–2004 identified sampling biases in blast holes and over projection of high grade blast holes for zinc
(iv) the F3 factors measured for 2003–2004 show that the tonnage, grade and metal for copper only ores forecast by the long range model are achievable
(v) the F3 factors measured for 2003–2004 show that the zinc metal forecast by the long range model is achievable. However, this comes at a higher tonnage and lower grade. Future long range models should consider allowing for overbreak of the marble/exoskarn contact.

Case study for hypothetical gold deposit

Schofield (2001) has raised the issue that reconciliation of the resource model and production does not go far enough. Inappropriate resource models and/or ore control practices can cause value to be lost. The case study presented below is typical of Carlin-type gold deposits in the USA. It shows how using too smoothed a resource model can destroy value. Figure 6 shows a histogram for composites and a resource model. It is possible to make a change of support transformation to the composite distribution to obtain distributions of SMU grades (Journel and Huijbregts, 1978). The discrete Gaussian method has been used for this case. Figure 7 shows the transformed distributions for four possible SMUs:

SMU1 = $5 \times 10 \times 6$ m (typical of a 3000 tpd operation)

SMU2 = $10 \times 10 \times 6$ m (typical of a 10 000 tpd operation)

SMU3 = $15 \times 15 \times 10$ m (typical of a 40 000 tpd operation)

SMU4 = $20 \times 20 \times 15$ m (typical of a 80 000 tpd operation).

It is important to note that the coefficients of variation (σ/m in Figs. 6 and 7) for the SMUs lie between the coefficients of variation for the composites and the resource model. Therefore, the resource model has a lower coefficient of variation than the coefficients of variation for all the candidate SMUs.

Figure 8 shows grade–tonnage curves. The tonnage for SMUs is usually less than the resource model tonnage; the average grade of the SMUs is always higher than the average grade of the resource model.

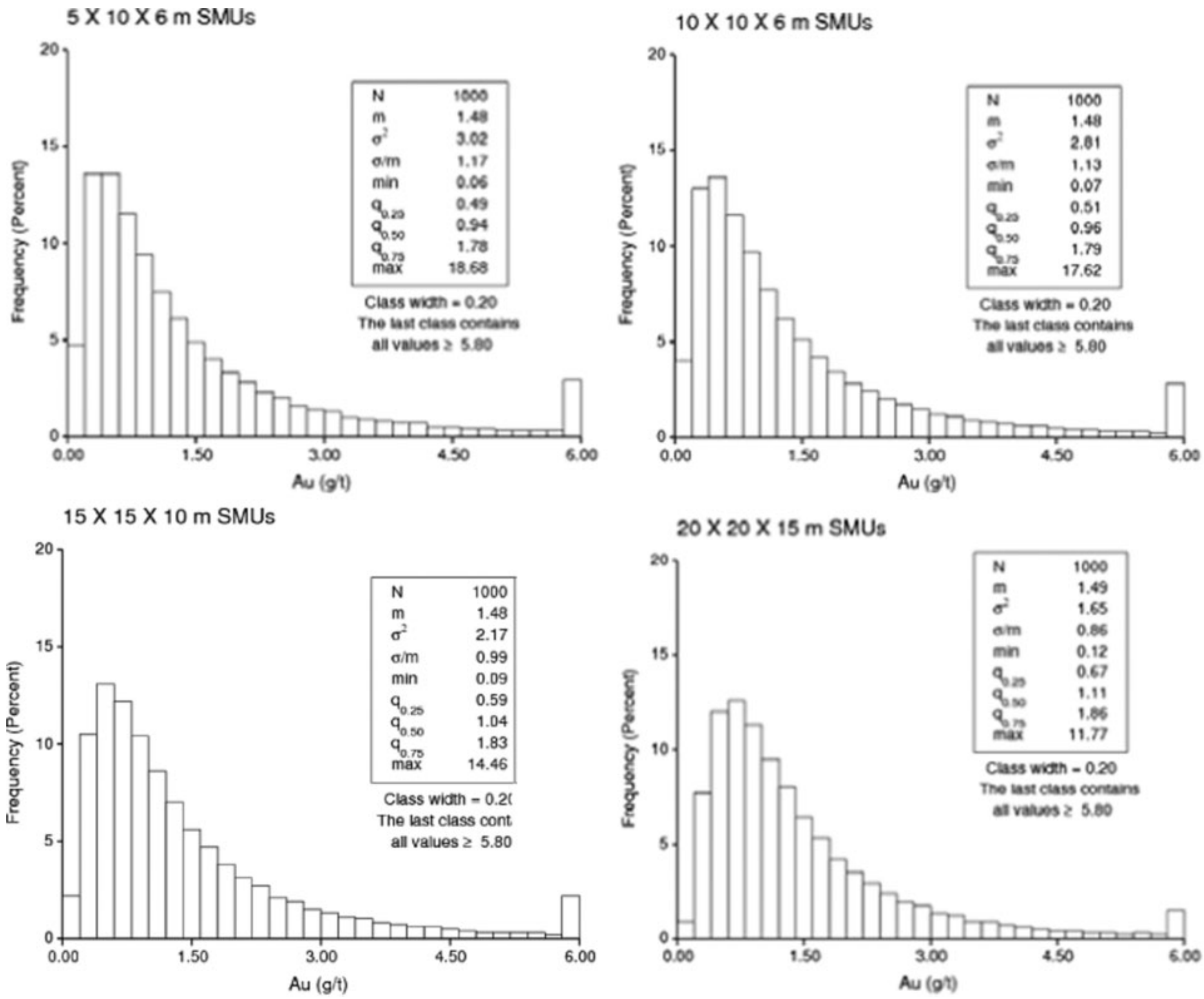
Table 10 shows the tonnages and average grades at a 1·5 g/t cut-off, with associated F1 factors. Table 11 shows a comparison for various mining scenarios. SMU2 would be considered the base case for a 10 Mtpa operation with 3 Mtpa of ore (10 000 tpd). It can be seen that there is very little improvement in cash flow for a more selective scenario (SMU1), and it might be impractical to be more selective in light of the need for more working room for smaller equipment. For SMU3 and SMU4 the effect of dilution and high treatment costs negates economies of scale in mining. The cash flows for a plan based on the resource model are about \$40 million/year less than the base case.

The lesson should be clear. Selectivity matters, particularly where treatment costs are high. An over smoothed model destroys value, as was eloquently shown by Zhang (1998). The solution suggested by Zhang was multiple indicator kriging. Since then the industry in the USA has turned more toward probabilistic domaining of high grade zones.

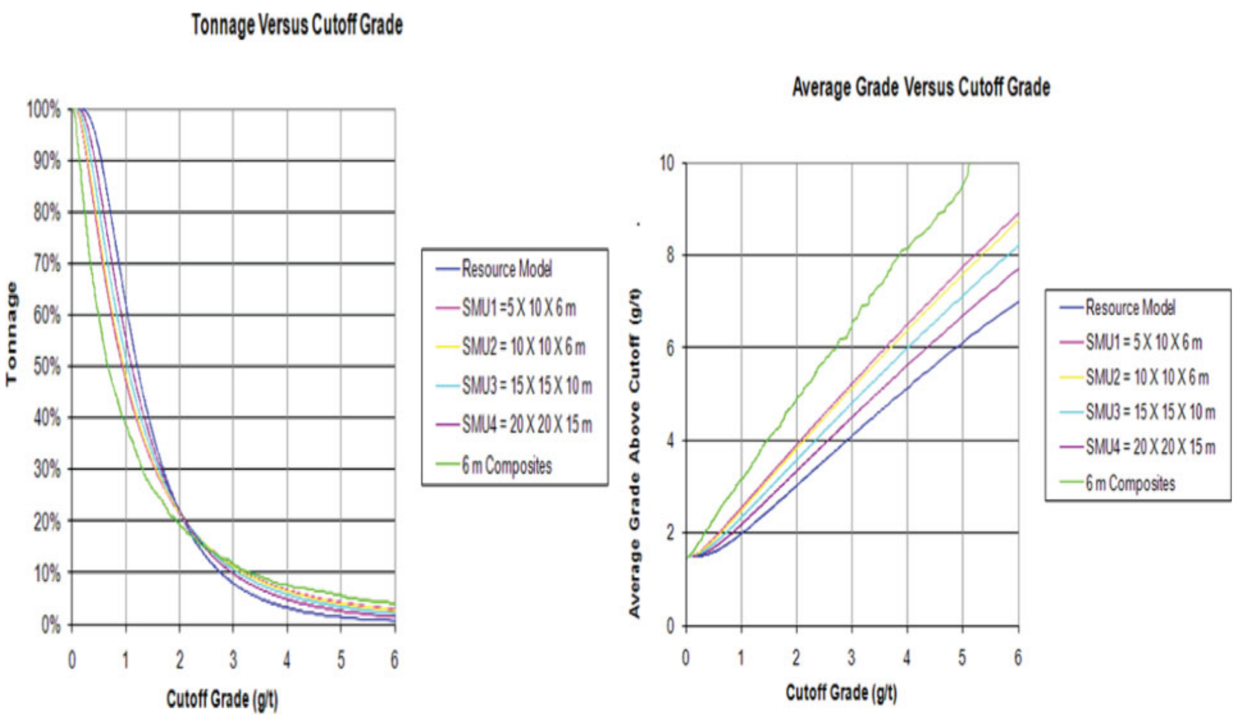
Case study for hypothetical iron ore deposit

This case study was motivated by discussions with Geoff Ballantyne of Rio Tinto. It involves ore loss and contact dilution. A good description is contained in Stone (1986), who spent many years collecting data and developing prediction formulae. Figure 9 shows a sketch of dilution and ore loss at contacts. If mining will tend to follow the dashed lines, both dilution and ore loss will occur. If mining overbreaks the lower contact (follows the dotted line), there will be no ore loss but dilution will occur.

To evaluate contact dilution and ore loss a simulator was developed. This simulator has been successfully used to adjust blocks for the Greens Creek underground mine, located near Juneau, Alaska and the True North open pit mine, located near Fairbanks, Alaska. Figures 10–12 show contact conventions, mixing zones and mining conventions. The mixing zone may be thought of as a zone where the contact shows short



7 Histograms for SMUs



8 Grade–tonnage curves

Table 10 Tonnages, grades and F1 factors

Results at 1·5 g/t cut-off grade (assume 100 Mtonnes ore + waste deposit)

	%tons	Mt	Grade (g/t)	Contained Au (Moz)
Composites	26·2	26·2	4·062	3·421
SMU1 = 5 × 10 × 6 m	31·1	31·1	3·229	3·228
SMU2 = 10 × 10 × 6 m	**31·5**	**31·5**	**3·167**	**3·207**
SMU3 = 15 × 15 × 10 m	33·0	33·0	2·959	3·139
SMU4 = 20 × 20 × 15 m	34·7	34·7	2·754	3·072
Resource model	37·1	37·1	2·487	2·966
F1 factors		Tonnage	Grade (g/t)	Contained Au (Moz)
Composites		0·706	1·633	1·153
SMU1 = 5 × 10 × 6 m		0·838	1·298	1·088
SMU2 = 10 × 10 × 6 m		**0·849**	**1·273**	**1·081**
SMU3 = 15 × 15 × 10 m		0·889	1·190	1·058
SMU4 = 20 × 20 × 15 m		0·935	1·107	1·036
Resource model		1·000	1·000	1·000

Table 11 Annual cash flows for selectivity scenarios (base case shown in bold)

Input variables

	SMU1	SMU2	SMU3	SMU4	Resource Model
Mining costs/$/t	2·00	**1·80**	1·60	1·40	1·80
Treatment costs/$/t	25·00	**25·00**	25·00	25·00	25·00
G + A costs/$/t	3·50	**3·50**	3·50	3·50	3·50
Sustaining Capex/$/t	0·30	**0·27**	0·24	0·21	0·27
Metallurgical recovery/%	88	**88**	88	88	88
Gold price/$/oz	1250	**1250**	1,250	1,250	1,250

Annual pretax cash flow (millions $); Assumes 10 Mtpa ore + waste

SMU1	SMU2	SMU3	SMU4	Resource model	
Ore mined/Mt	3·11	**3·15**	3·3	3·47	3·71
Ore grade/g/t	3·23	**3·17**	2·96	2·75	2·49
Contained MOz	0·323	**0·321**	0·314	0·307	0·297
Recov. MOz	0·284	**0·282**	0·276	0·270	0·261
Revenues/M $	355·1	**352·8**	345·3	338·0	326·3
Mining/M $	−20·0	**−18·0**	−16·0	−14·0	−18·0
Treatment/M $	−77·8	**−78·8**	−82·5	−86·8	−92·8
G + A/M $	−35·0	**−35·0**	−35·0	−35·0	−35·0
Sustaining Capex/M $	−3·0	**−2·7**	−2·4	−2·1	−2·7
Cash flow/M $	219·4	**218·3**	209·4	200·1	177·8

amplitude deviations from a plane or a zone of mixing during blasting.

Ore loss and dilution are expressed in distance in metres perpendicular to block faces on contacts. For an ore block:

$$\text{Adjusted tonnes} = \text{tonnes } \textit{in situ} - \text{face area} \times \text{Density}_{\text{ore}} \times \text{Ore loss (m)} + \text{face area} \times \text{Density}_{\text{waste}} \times \text{Dilution (m)}$$

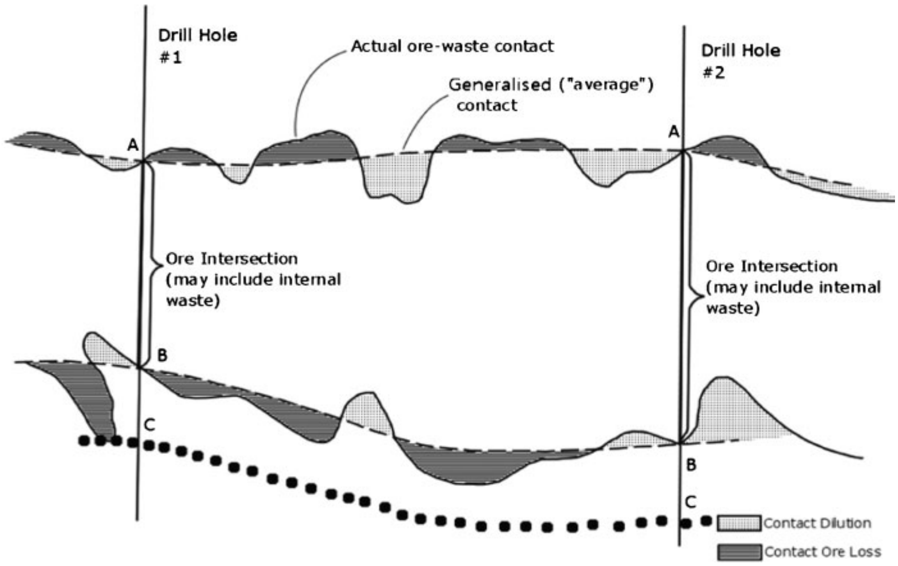
$$\text{Adjusted grade} = [\text{tonnes } \textit{in situ} \times \text{grade } \textit{in situ} - \text{face area} \times \text{Density}_{\text{ore}} \times \text{Ore loss (m)} \times \text{grade } \textit{in situ} + \text{face area} \times \text{Density}_{\text{waste}} \times \text{Dilution (m)} \times \text{dilution grade}]/\text{adj. tonnes}$$

Similar formulae apply to transfer of ore loss into adjacent waste blocks.

In the hypothetical case, the focus is on the sensitivity of dilution and ore loss to the degree of mixing and mining face position. Held constant are the strike and dip of the contact (0° and 45°) and the slope of the mining face (70°). Table 12 shows the simulator output.

Where the face position is more negative (compare runs 3, 8, 13, 18 and 23) there is more dilution and less ore loss. For a given face position (compare runs 11, 12, 13, 14 and 15) there is more ore loss and dilution as the half width (TOLMIX) of the mixing zone is increased from 1 to 5 m.

Table 13 shows production results. For simplicity the short term model is assumed to follow a tabular bed of iron ore in the field. Table 14 shows the simulator cases and the resultant F2 factors. Run 15 (mixing is 5, mining face position is 0) has F2 factors that match the production results (Table 13).



9 Ore loss and contact dilution (after Stone, 1986)

Table 14 shows the calculated NPVs, which take into account not only run of mine grade but an adjusted mine life (10 Mtpa run of mine assumed) related to the net amount of ore loss/dilution gain. These have been recast in Table 15, with the optimal case for each mixing zone shown in italic. If the mixing remains at 5 m, the highest NPV will come by moving the face position west to −5. The contact will be overbroken, but ore loss will be minimised. If mixing can be reduced, the degree of overbreak required can be reduced.

The observation made by Schofield (2001) is again made clear. The reconciliation should be taken as a basis for improvement. In a real life situation, the grade at the contact would need to be considered, with the amount of overbreak being dependant on the ore grade.

Conclusions

In most mines the most critical factors in poor reconciliation relate to lack of orebody knowledge or failure to mine to the defined mine plan. In the first case, the plan is based on an incorrect interpretation of the orebody in terms of orebody position, local tonnage and grade, or degree of selectivity implicit in the long term

Table 13 Production statistics

	Mtonnes	%Fe	Mt cont. Fe
Short term model depletion	10·00	40·00	4·00
Received at mill	9·75	38·00	3·71
F2 factor	0·975	0·950	0·926

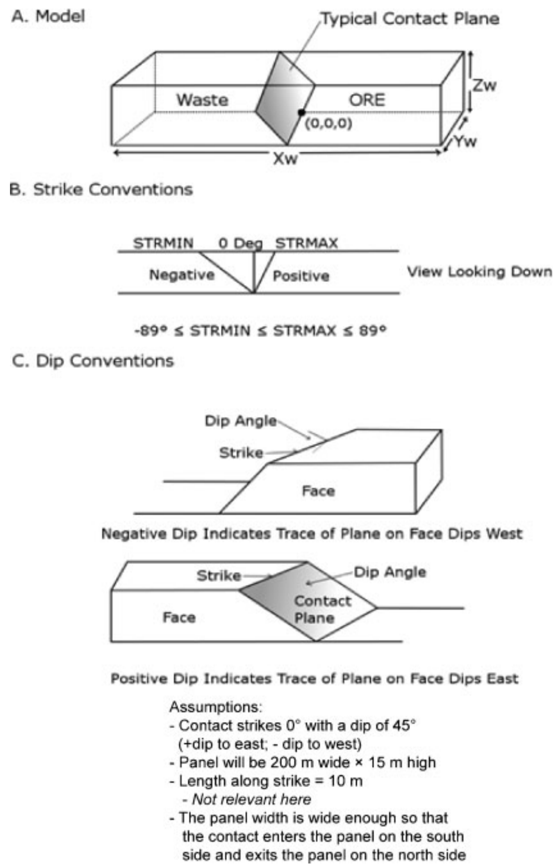
Table 12 Simulator Output

Run	Mixing	Mine face position	Dilution Width (m)	Ore Loss Width (m)
1	1	−5	4·94	0·00
2	2	−5	5·00	0·03
3	3	−5	5·37	0·27
4	4	−5	5·62	0·58
5	5	−5	6·10	1·06
6	1	−2·5	2·58	0·17
7	2	−2·5	2·89	0·46
8	3	−2·5	3·53	0·96
9	4	−2·5	4·13	1·62
10	5	−2·5	4·74	2·24
11	1	0	0·91	0·97
12	2	0	1·30	1·33
13	3	0	2·23	2·13
14	4	0	2·83	2·80
15	5	0	3·52	3·49
16	1	2·5	0·17	2·76
17	2	2·5	0·40	2·96
18	3	2·5	1·01	3·44
19	4	2·5	1·55	4·04
20	5	2·5	2·25	4·76
21	1	5	0·00	5·06
22	2	5	0·03	5·06
23	3	5	0·27	5·17
24	4	5	0·56	5·52
25	5	5	1·11	6·08

Table 14 Calculation of F2 factors for simulation runs*

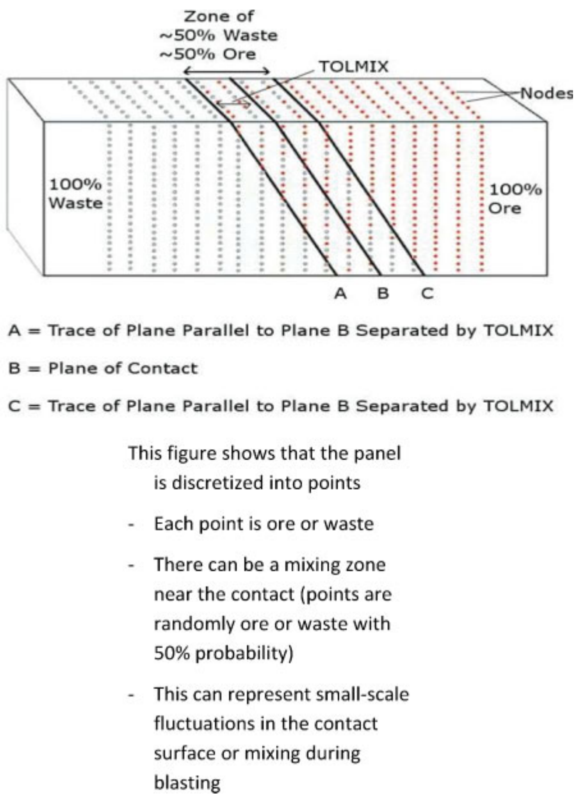
									From simulator		Run of mine				F2 factors			NPV
Run	Mixing	Mine face position	Ore grade/ %Fe	Density of ore	Diluted grade/%Fe	Density of waste	Short term reserve depl. Mt	Horiz. width ore/m	Dilution width	Ore loss width	Width/ m	Grade %Fe	Density/ t/m^3	Mtonnes	Tonnes	Grade	Metal	$M
1	1	−5	40	3·6	0·1	2·6	10	40	4·94	0·00	44·94	37·50	3·490	10·89	1·089	0·939	1·022	1235
2	2	−5	40	3·6	0·1	2·6	10	40	5·00	0·03	44·97	37·51	3·489	10·90	1·090	0·938	1·022	1233
3	3	−5	40	3·6	0·1	2·6	10	40	5·37	0·27	45·10	37·33	3·481	10·90	1·090	0·933	1·017	1224
4	4	−5	40	3·6	0·1	2·6	10	40	5·62	0·58	45·04	37·20	3·475	10·87	1·087	0·930	1·011	1215
5	5	−5	40	3·6	0·1	2·6	10	40	6·10	1·06	45·04	36·95	3·465	10·84	1·084	0·924	1·001	1200
6	1	−2·5	40	3·6	0·1	2·6	10	40	2·58	0·17	42·41	38·66	3·539	10·42	1·042	0·966	1·007	1256
7	2	−2·5	40	3·6	0·1	2·6	10	40	2·89	0·46	42·43	38·50	3·532	10·41	1·041	0·962	1·002	1246
8	3	−2·5	40	3·6	0·1	2·6	10	40	3·53	0·96	42·57	38·16	3·517	10·40	1·040	0·954	0·992	1229
9	4	−2·5	40	3·6	0·1	2·6	10	40	4·13	1·62	42·51	37·84	3·503	10·34	1·034	0·946	0·978	1208
10	5	−2·5	40	3·6	0·1	2·6	10	40	4·74	2·24	42·50	37·51	3·488	10·30	1·030	0·938	0·965	1188
11	1	0	40	3·6	0·1	2·6	10	40	0·91	0·97	39·94	39·50	3·577	9·92	0·992	0·988	0·980	1257
12	2	0	40	3·6	0·1	2·6	10	40	1·30	1·33	39·97	39·29	3·567	9·90	0·990	0·982	0·973	1245
13	3	0	40	3·6	0·1	2·6	10	40	2·23	2·13	40·10	38·78	3·544	9·87	0·987	0·969	0·957	1217
14	4	0	40	3·6	0·1	2·6	10	40	2·83	2·80	40·03	38·44	3·529	9·81	0·981	0·961	0·943	1196
15	**5**	**0**	**40**	**3·6**	**0·1**	**2·6**	**10**	**40**	**3·52**	**3·49**	**40·03**	**38·05**	**3·512**	**9·76**	**0·976**	**0·951**	**0·929**	**1173**
16	1	2·5	40	3·6	0·1	2·6	10	40	0·17	2·76	37·41	39·90	3·595	9·34	0·934	0·998	0·932	1225
17	2	2·5	40	3·6	0·1	2·6	10	40	0·40	2·96	37·44	39·77	3·589	9·33	0·933	0·994	0·928	1218
18	3	2·5	40	3·6	0·1	2·6	10	40	1·01	3·44	37·57	39·41	3·573	9·32	0·932	0·985	0·919	1201
19	4	2·5	40	3·6	0·1	2·6	10	40	1·55	4·04	37·51	39·09	3·559	9·27	0·927	0·977	0·906	1181
20	5	2·5	40	3·6	0·1	2·6	10	40	2·25	4·76	37·49	38·68	3·540	9·22	0·922	0·967	0·891	1157
21	1	5	40	3·6	0·1	2·6	10	40	0·00	5·06	34·94	40·00	3·600	8·74	0·874	1·000	0·874	1174
22	2	5	40	3·6	0·1	2·6	10	40	0·03	5·06	34·97	39·98	3·599	8·74	0·874	1·000	0·874	1174
23	3	5	40	3·6	0·1	2·6	10	40	0·27	5·17	35·10	39·83	3·592	8·76	0·876	0·996	0·872	1169
24	4	5	40	3·6	0·1	2·6	10	40	0·56	5·52	35·04	39·65	3·584	8·72	0·872	0·991	0·865	1157
25	5	5	40	3·6	0·1	2·6	10	40	1·11	6·08	35·03	39·31	3·568	8·68	0·868	0·983	0·853	1138

*Shown in bold is the case that most corresponds to production staitistics (see Table 13).

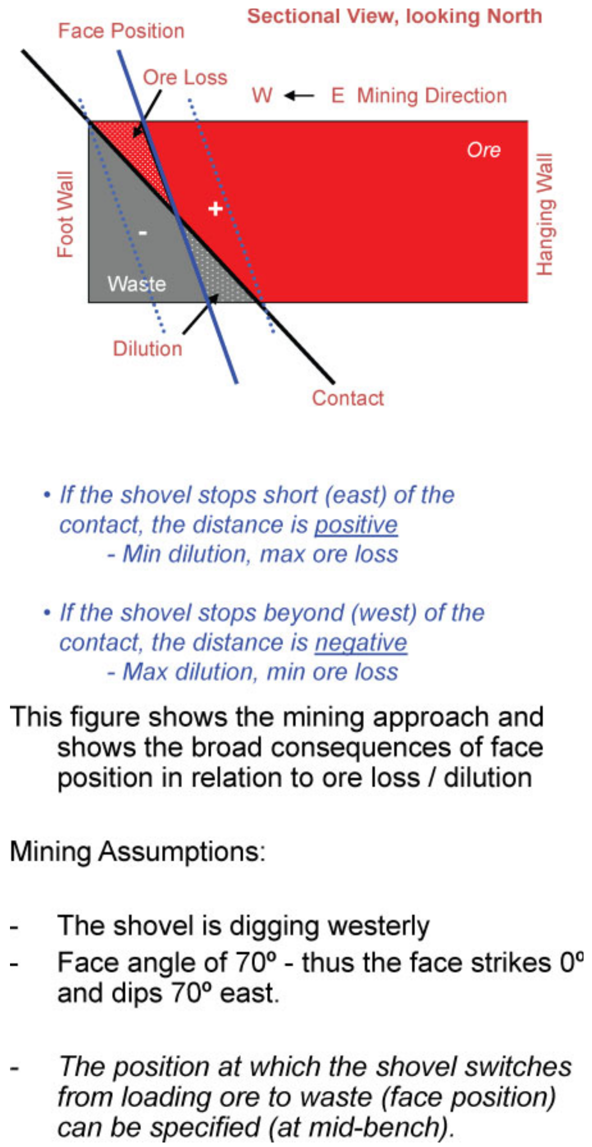


10 Contact conventions

model. In nearly all cases [Jerritt Canyon, Hayden Hill and Goldstrike (USA); Kelian, Indonesia; Escondida, Chile; Antamina, Peru; Diavik, Canada, and Palabora, RSA are examples), the solution is infill drilling, and



11 Mixing zones



12 Mining conventions

wherever possible blocks should be informed so that they can be considered as proved ore reserves well in advance of mining. In the second case, given that the plan has been optimised, failure to mine to it will in the end have the consequence of destroying value.

Reconciliation data can also serve as a basis for improvement of the mining process. Examples are Greens Creek Alaska (improved face sampling),

Table 15 Net present value presented as case matrix*

	Mining face position				
Mixing	−5	−2·5	0	2·5	5
1	1235	1256	*1257*	1225	1174
2	1233	*1246*	1245	1218	1174
3	1224	*1229*	1217	1201	1169
4	*1215*	1208	1196	1181	1157
5	*1200*	1188	**1173**	1157	1138

*Shown in bold is the case that most responds to production statistics, shown in italic is the optimal case for a given mixing zone half width (TOLMIX). The overall optimum case has mixing of 1, mining face position of 0.

Nchanga, Zambia (reduced unplanned dilution in both open pit and underground operations) and Stillwater, Montana (reduced mixing of ore and waste in remuck bays). In many cases these improvements come at very little additional cost.

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