

Geostatistical Modelling of an Orebody as an Aid to Mine Planning

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ABSTRACT

The increasing complexity of modern mining technology makes it ever more difficult to decide on the 'best' way to solve problems in mine planning. Yet with escalating costs it becomes increasingly more important for the mining engineer to be able to evaluate quantitatively the implications of the 'what happens if' postulation on the efficiency and economics of exploitation. The use of a model of the orebody can yield an estimate, not just of overall expected results, but also of the sensitivity of these results to possible variations in the orebody.

Estimation techniques, such as Kriging, will produce the expected value for the grade of ore at any given point, or over any given volume within a deposit. However, the actual grades mined during production will - of necessity - differ from the predicted values. These possible variations, and the sensitivity of the management decisions to them, can be investigated in detail by the use of simulation.

An orebody can be simulated which follows the known characteristics of the deposit at any *given stage in* development. These properties might be the histogram of grade values, the semivariogram from the samples, the actual borehole assays, geological constraints and/or interpretation, topography, lease boundaries etc. This would be one possible 'realisation' of the deposit as dictated by the currently known data. If several of these simulations are carried out, each conditioned to the current knowledge of the orebody, then it is possible to gain an idea of the range of possible outcomes of any mining procedure or decision taken - all of which are equally likely. Not only can the best decision then be taken, but its sensitivity to the inherent variation within the orebody, and to the ever present estimation errors may be thoroughly studied.

INTRODUCTION

The orebody being considered in this paper is a complex base metal sulphide (Pb/Zn) deposit of extensive strike length. The orebody is tabular in form with a fairly uniform true thickness of six metres, and a consistent average dip of 25 degrees, outcropping in fairly flat topography. The country rock, which represents overburden, is a succession of strong limestones interbedded with shales and for the purposes of this exercise is assumed to be competent and with little groundwater.

It is proposed to mine the deposit at a rate of about one million tons of ore per annum, initially by open-pit methods, and then to continue underground at the same production rate. The open-pit method proposed is a simple strip-mining technique proceeding down dip, and it is intended to develop the underground pillar and stall operation directly from the bottom of the open-pit using the same (or similar) mining equipment. The initial decision to be made is when to change from open-pit to underground operations. Ultimately, the further (or possibly alternate) decision could also be made between this interim underground operation and the use of a wholly underground system using shaft and/or decline access and purpose built underground equipment, with the aid of further simulation down dip.

In the early stages of mine planning it is often necessary to base decisions on inadequate or incomplete information regarding the nature of the orebody. Very frequently, an overall average grade for the deposit is used alone without reference to the actual disposition of the ore grades. Subsequently, 'geological' plans and sections are sometimes produced, based on borehole intersections and such orebody exposures as are available, for more detailed planning and feasibility studies. Such plans and sections must be only one of the many possible - and equally probable - interpretations of the data available. Their preparation is costly in terms of time, effort and money, and they may still be unrealistic unless the characteristics of the orebody are clearly defined. It is desirable to be able to test the 'model' produced to determine the influence of any inaccuracies in interpretation on the overall result.

Since each set of plans/sections is based on a given prediction of the orebody variations, this may prove to be impractical.

The use of orebody simulations such as described in this paper attempts to predict with greater reliability the ore distribution and its inherent variations. Repeated simulations enable the testing of the study for sensitivity to these variations and to error in interpretation. Thus we have a tool that can supplement, or even eliminate the necessity for the preparation of detailed working plans at an early stage in determining the probability of, and margins of error in, feasibility studies.

This paper demonstrates the use of the method with a realistic unconditioned simulation prepared from borehole data. The approach is intentionally simplistic throughout in order to illustrate the use of the method rather than to attempt a definitive project study.

Subsequently, the model would be conditioned to the data used in its preparation and such other information as may be available, thereby making the variability in the simulations conform to the actual spatial variations in the localised data. Additional 'realistic' complications could be incorporated at this stage.

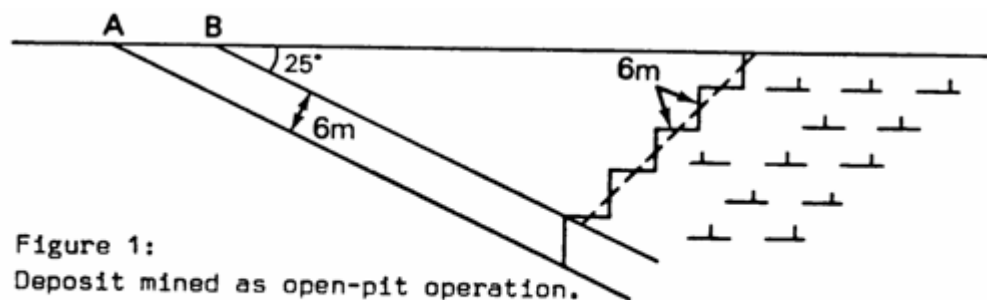
The actual simulation used is of a deposit in which the grades follow a log-normal distribution, with average grade 7.5% combined metal and a standard deviation of 7.1%. This deposit was found to have a semi-variogram of the spherical type, with a range of influence

of 15 metres. Six metre cube blocks have been selected, each being equivalent to approximately 580 tonnes of overburden (specific gravity 2.7) or 650 tonnes of ore (average s.g. 3.0). The study was limited purposely by considerations of computer real and operating time to the simulation of 60 blocks (360m) down dip on a representative strip 10 blocks (60m) wide. This pattern could be extended by further simulations across the orebody with some increase in computer costs and care requirements. The distribution, which is a realistic approximation to natural conditions, yields an average grade in situ of 9.5% combined metal at a working cutoff grade of 3%.

THE MODEL

Mining Criteria - Open-pit

The method for the open-pit mining will be strip mining using 6m high benches proceeding down dip (Figure 1). The operation is considered to be by conventional drilling and blasting, using benching holes with mobile loading equipment and truck haulage. 100% extraction is assumed with ore and internal waste (below cutoff) being mined selectively. A safe high-wall slope of 45° has been assumed. In the interests of simplicity the operation has been considered as a planar slice in the centre of the deposit, ignoring 'end-effects'. In addition the determination of the overburden removed has been by calculation of the volume of the extending wedge.



THE MODEL

Underground

Similar loading and hauling equipment is to be used underground, but tunnelling rounds will be drilled in 6m square cross-section drives using jumbo drilling rigs. The method proposed is a simple regular pillar and stall operation with 6m square pillars yielding a nominal extraction ratio of 75%. Waste rounds of low grade ore will be cleaned and transported to dump but not processed. For simplicity it is considered that the extraction profile will be and down dip. In practice rooms advancing along strike this would probably be complicated by the provision of inclined access roadways or by orienting the workings at a more shallow diagonal inclination across the line of true dip. Such modifications could easily be incorporated at any stage. Four alternative extraction patterns with different pillar locations have been used for each simulation (see Figure 2). The first two - 2(a) and 2(b) - are considered as having the first 'block' in each strip mined by open-pit methods, and then pillars are started, whilst the others assume that both of the first two rows from surface are mined by the open-pit techniques.

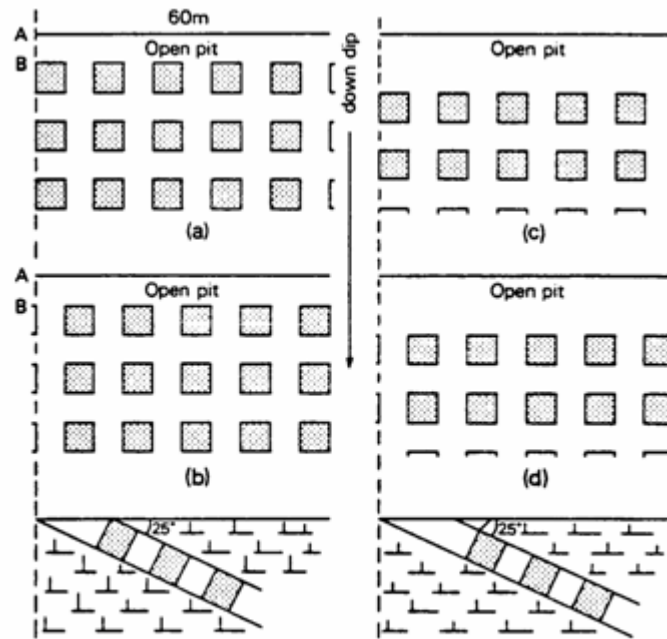


Figure 2: Deposit mined underground showing alternative pillar locations.

The simulation technique used was the now standard 'turning bands' method of Journel (1). An unconditional simulation has been used for this paper, so that the resulting deposit follows the 'global' characteristics of the real deposit, but not as yet the particular local characteristics. The deposit has been sampled by means of drillholes, and it was found that the grades of the area in question follow a log-normal distribution with a mean of 7.5% and a standard deviation of 7.1%. The spatial nature of the area was also investigated, yielding an experimental semi-variogram of the spherical type with a range of influence of 15 metres.

The method of simulation requires some approximations in its application, which will be discussed only briefly here. Since the turning bands method has been admirably explained in Journel and elsewhere, the authors refer those unfamiliar with the technique to that paper. The main advantage of the method over classical variants is its speed and simplicity. It is only necessary to produce continuous simulations in one dimension, and 'revolve' these to produce three dimensional results. Three approximations are inherent in the method:

- (i) The one dimensional simulations must be produced from a finite number of points, not a continuum as in the ideals
- (ii) The three dimensional simulation is produced from a finite number of one dimensional ones, not a solid sphere as in the ideal cases
- (iii) A 'block' of ore must be approximated by a network of points instead of a solid continuous volume.

Some extensive study was made on the first of these three points. It was found that if the discrete independent variables were simulated at intervals of one-hundredth of the range of influence (i.e. 100 independently simulated variables are combined to produce each sample point), a very satisfactory semi-variogram could be produced from the samples. Using more than 100 points yielded no significant improvement in the fit, whilst using less worsened the fit considerably.

The second of the approximations is discussed in Journel, who concludes that 15 independent directions are sufficient to convert the one dimensional to three dimensional simulations. The third part has been a topic of study elsewhere, and Marechal (2) states that 64 points are sufficient to characterise the estimation of a block to within 1% accuracy.

Economic Considerations

In both underground and open-pit operations 10% dilution by waste, due to the mining of waste rock at irregular ore boundaries, has been assumed, effectively reducing the grade mined, but increasing the tonnage. In addition 10% loss of ore at irregular boundaries has been assumed, reducing the tonnage of ore extracted. Using these criteria, assuming normal commercial recoveries, concentrate grades, and smelter and shipping charges, at an average price for lead of 1200 per tonne and for zinc of £350 per tonne, one tonne of 10% ore in situ would have a value of about £3.50.

Costs

An operation based on a 250 day year using two 8-hour shifts per day has been assumed, requiring approximately 2000 tonnes of ore to be mined each shift. Basic excavation costs have been assessed at £0.35 per tonne for the open-pit with no provision for restoration, and £1.50 per tonne for the underground mine. Ore, low grade mineral (

The costs of transportation and pumping will increase as the work progresses down dip. However, we shall ignore the differences here between the open-pit and underground operations in these cost elements by assuming flat inclined access in the underground operation and comparable water influx. Thus we may work with differential costs only. Ventilation and support costs underground are also assumed to be constant. In a more stringent engineering study allowances could easily be made for calculable variations.

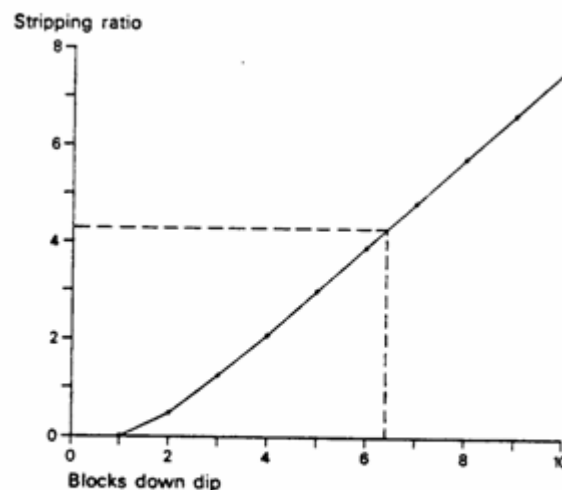


Figure 3: The simplest decision rule, Stripping ratio versus depth.

The Decision Point

In the simplest possible orebody model, i.e. assuming a constant average grade of 9.5%, we can ignore the question of revenue from the mineral actually mined. and, by calculation of the stripping ratio, determine the point at which the increasing open-pit mining cost (due to increasing depth of overburden) equals the underground mining cost. This changeover point occurs at a volumetric stripping ratio of 4.29:1 and reference to Figure 3 indicates that this will be at 6.4 blocks (or 38.4m) down dip from the outcrop.

To obtain a more realistic estimation by conventional methods would entail the preparation of detailed plans and sections based on subjective interpolation of sample information and would entail a great deal of tedious trial and error estimation.

The preparation of a simulated orebody model provides the opportunity to try out a multitude of possible interpretations of the information available even when conditioned to the existing data. We therefore have the situation of having a 'known' variation of block value and must consider not just the varying mining cost but also the revenue from the mineral extracted.

The net revenue from each strip of ten blocks progressing down dip has been computed for both open-pit and underground operations using the following relationship:

Revenue from block - block value-mining cost-overheads-treatment cost

= (metal content x in situ value) – ore mining cost for block – overheads cost for block – treatment cost for block

No contribution in value is made by low grade ore mined as waste, but costs of mining and overheads are still accounted.

DISCUSSION

Fourteen separate simulations of an area ten blocks wide and thirty blocks down dip were carried out on a CDC 6400 computer. Each simulation is carried out in two stages - the first being the one-dimensional simulations which are then stored on magnetic disc, and the second is the actual mine simulation. The size of the study area was selected to investigate requirements of core space, run time and overall cost for this type of simulation.

The range of variation between these simulations is illustrated in Figure 4, which shows the net revenue accrued in open-pit mining a strip of ten blocks at a certain distance down dip. The very large difference in revenue is due to the simulations being completely independent of one another – that is they have not yet been conditioned to localised sample data, but only to the overall characteristics of the deposit. However, the downward trend in revenue is immediately apparent from the graph. A similar graph for underground mining presents a solid line of bars across the page as should be expected. Figure 5, therefore, has been chosen to illustrate the net revenue obtained from one of the underground simulations as each 'strip' is mined. The variations experienced in this graph would not greatly be reduced by conditioning to local data, since the simulation contains information on the continuity of the ore, but would conform more closely to the 'true' variations.

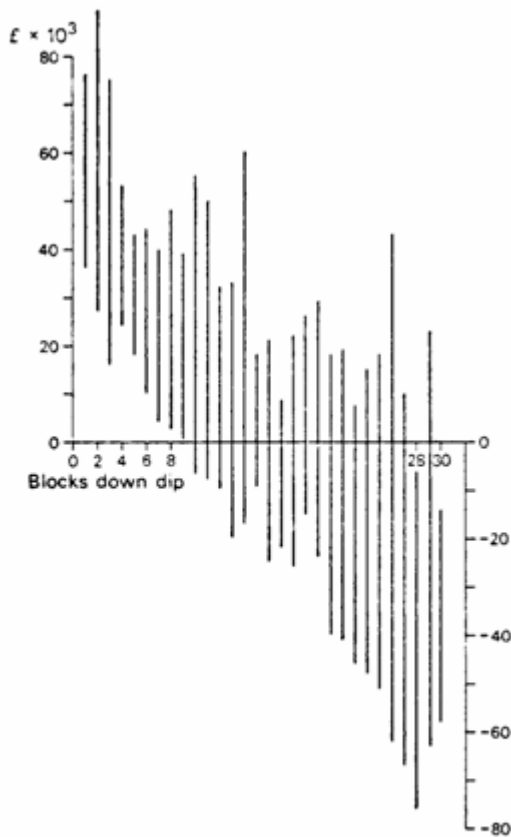


Figure 4:
Net revenue from open-pit mining
per strip of ten blocks -- all
simulations.

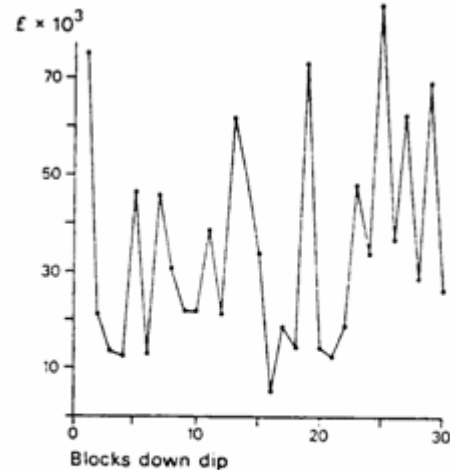


Figure 5:
Net revenue from underground
mining per strip of ten blocks
-- one simulation only.

Taking a trend line through Figures 4 and 5 produces an intersection in the vicinity of 5 to 6 blocks down dip, supporting the simple solution. However, this simple solution does not take into account the range of variation within the deposit, which indicates that the decision point could be reached anywhere between surface and twelve blocks down dip.

The same simulation is again illustrated in Figure 6, which shows the total revenue for the first thirty blocks down dip plotted against the point at which the decision to go underground was made. The graph can be seen to be slightly erratic. This is due to the fact that the first 'strip' mined as underground workings must contain the line of pillars, so that the alternate figures are based on a different configuration of pillar and stall. The figure indicates that the decision point could be chosen anywhere between four and nine blocks down dip, with a suggestion that six to eight blocks would be preferred due to local high grades. Conditioning of the model to the local data should provide a more distinct (and accurate) decision point.

One concluding comment may be made here. Although all fourteen simulations were completely independent, the final conclusion on the decision point was virtually the same in all cases, differences being due entirely to the local variations in grades. This suggests strongly that conditioning of the model would produce a very valuable tool for this type of situation.

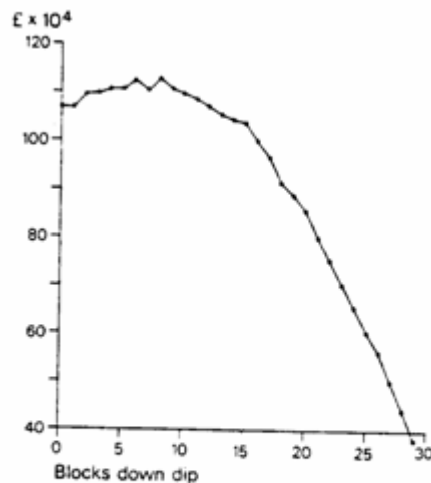


Figure 6:
Total revenue from deposit ver-
sus the decision point for
changeover.

Extensions of the Model

The deposit constructed for this paper was essentially simplistic. However, it was found that the major constraints on the simulated model were those of actual physical volume of ground covered. Using the interactive system at Imperial College, only 25K core storage is allowed at run time, although virtually unlimited time is available. It was found that within this core space, six hundred blocks (i.e. about 40,000 individual points) could be produced in a total core time of 840 seconds. Of this about 120 secs is used in stage one - the one dimensional simulations, and the majority of the rest in sorting out the fifteen sets of co-ordinates for the conversion to three dimensions. It is hoped that further study will cut down the time required for the second stage considerably. In user time this represented about one to one and a quarter hours per run. The problems involved in expanding to cover a larger area are mostly data and file manipulation problems - not theoretical or programming problems.

Other considerations can easily be incorporated into the model to produce a more 'realistic' engineering model. These could include: topography - including environmental and climatological constraints; geology - folding, faulting and bedding structures. the presence of superficial deposits, variations in dip and thickness ore distribution and composition - zoning of mineralisation and/or values, secondary enrichment, presence of byproducts or penalty elements, mixtures of ore types; specific gravity - variations with ore grade and rock type; mining methods and conditions - more stringent determination of extraction profile, sequence of operations, orientation of pillars, variation of parameters with rock type etc.; costs influence of depth, nature of material, time-dependence etc.; mineral value - marketing considerations such as time, grade, escalation, also extraction efficiency.

Conditioning of the model to the available local data is no great problem. as this simply entails the production of kriging estimates for the blocks from both real and simulated sample data and forming a linear combination of these with the simulated block values.

Potential Uses

Such orebody models, once conditioned to conform to the original individual assay values, can be used as living mathematical representations of the mine. They can be progressively updated as new ore is developed and proved, and as stopes are exhausted such that a progressive ore reserve estimation can be readily to hand which takes into account the latest available information.

Another application could be in the field of production planning and scheduling where the tonnage and grade of ore from producing units, and the likely variability, can be predicted with a greater degree of accuracy. This could lead to the production of a more consistent mill head grade and the more efficient extraction of the orebody.

Long term plans may be projected with more certainty as the model 'fills in' the unknown values between widespread sampling points or exposures. The model could also be used to test proposed development plans such that the most cost-effective and productive are utilised.

In the educational field the simulated orebody represents a most powerful teaching and research tool applicable to a multitude of problems in exploration, planning and production. Through its use the student can fairly painlessly see the effect of his/her decisions in the search for the optimal solution.

REFERENCES

Journel, A., "Geostatistics for conditional simulations of orebodies". Economic Geology, Vol. 69, No. 5. 1974 pp.673-687

Marechal, A., Personal Communication, October 1975.