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MILP Production Scheduling Models for Evaluating Continuous Improvement Projects

Steven Hoerger
Peak View Mine Planning, Englewood, CO

SUMMARY

Mixed integer linear programming (MILP) production scheduling models can be used to evaluate if tonnage improvement projects will deliver value over all time periods and whether new bottlenecks will be created. These MILP models can evaluate the interconnected effects of mine and process capacities, mining locations, cutoff grades, stockpiles, and orebody grade/tonnage distributions.

A case study compares potential value creation from a mine capacity increase, a mill capacity increase, or a simultaneous increase in all capacities. For perspective, project valuations for capacity increases are compared against valuations for cost reduction projects, mine planning improvements such as phase design, dynamic cutoff grades and stockpiling, and the impact of geostatistics improvements in the areas of block model accuracy and ore control selectivity.

INTRODUCTION

Mine planners seek to create production schedules to maximize the value generated from an orebody subject to mining and processing constraints and defined cost structures. Continuous improvement (CI) leaders often focus on increasing value by expanding constraints and improving cost structures. Accurate prediction of the value to be created by constraint expansions requires understanding how production schedules will change as mine and process capacity changes interact with orebody grade-tonnage distributions. Accurate estimates of value creation allow CI investments to be prioritized against each other and against available capital budgets and organizational change bandwidth.

An open pit production schedule defines which locations are mined when—the mining sequence—and which mined material is sent to which destination at which time—the processing sequence (Clark and Dagdelen, 2023). The destinations could be a waste dump, stockpile or one or more processing facilities that generate a saleable product. At an operation with a single processing facility, the processing sequence is often expressed as a cutoff grade schedule: material above a time period's operating cutoff grade is sent to the processing facility; material below the operating cutoff is sent to the waste dump. A breakeven cutoff ensures that material above cutoff generates sufficient revenue to cover the costs of processing the material vs the alternative of sending the material to the waste dump. An operating cutoff should cover these costs plus the opportunity cost of delaying the processing of future material (Rendu, 2014). At many mines, a stockpile cutoff is also used to define material to be saved for processing at the end of the mine life or during periods of insufficient ore availability. The optimum mining and processing sequences often result in "balanced" operating cutoff grades where yearly tons mined equals the mining capacity constraint and yearly tons mined above the operating cutoff grade matches the processing constraint (Lane, 1988). In this situation, where both mining and processing are bottlenecks, translating constraint expansions into expected metal production increases becomes challenging because the capacity changes lead to cutoff changes in each time period.

This paper uses a case study to demonstrate how a MILP production scheduling model can be used to estimate the benefits of different types of improvement projects. Other models could also be used to estimate value; at a minimum, a value estimation model should account for finite resources and allow for cutoffs to be varied over time to balance processing and stockpiling with the ore grades available at any given time. The model used, nicknamed Peakfinder, also accounts for flexibility to reallocate mining between laybacks and incorporates vertical advance constraints. For perspective, the Peakfinder model can also be used to compare the value of capacity constraint expansions against the value of cost reductions, mine planning changes, geostatistical block estimation changes and mine selectivity changes.

PEAKFINDER PRODUCTION SCHEDULING MODEL

The Peakfinder MILP production schedule optimization model is designed to compute mining and processing sequences that maximize the net present value (NPV) of cash flow generated from a set of mines and destinations (processing facilities or waste dumps). The model will be described using open pit terminology, but it could also be used for operations that include underground mines.

The Peakfinder model works with one or more mines subdivided into laybacks and panels with the tonnage in each panel split into many increments to provide metallurgical and grade resolution. Increments can be defined as histogram bins for single metal deposits or as multivariate bins or k-means clusters for multi-product mines (Goodfellow and Dimitrakopoulos, 2014). Each panel typically consists of one or more benches. The optimization problem is defined in terms of maximizing NPV by choosing tonnage flows for each time period (Figure 1): the tonnages to be mined from each mine, layback, panel, **W**(m,l,p,t); the increment tonnages to be sent from each mine to each destination, **X**(m,i,d,t) and/or to each stockpile, **Y**(m,i,s,t); and the increment tonnages to be sent from each stockpile to each destination, **Z**(i,s,d,t).

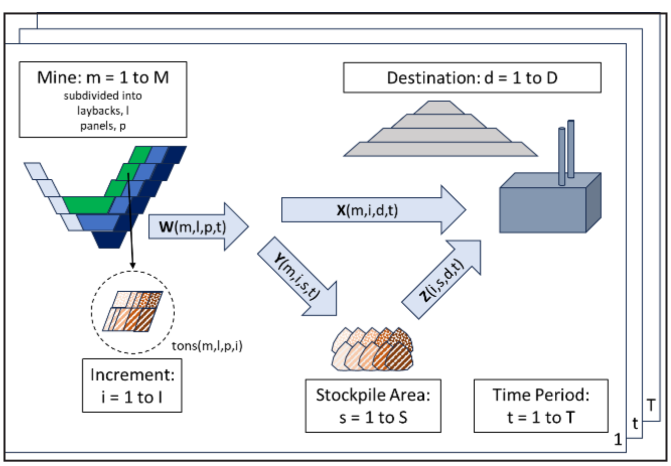


Figure 1. Peakfinder MILP model overview

An overview of the MILP mathematical formulation is presented here. The objective function is expressed as:

Maximize Discounted Net Surplus:

$$\begin{aligned}&\textstyle\sum_m \sum_l \sum_p \sum_t pv(t)^*MineVal(m,l,p,t)^*\mathbf{W}(m,l,p,t)\\&+ \textstyle\sum_m \sum_i \sum_d \sum_t pv(t)^*DestVal(m,i,d,t)^*\mathbf{X}(m,i,d,t)\\&+ \textstyle\sum_m \sum_i \sum_s \sum_t pv(t)^*StockVal(m,i,s,t)^*\mathbf{Y}(m,i,s,t)\\&+ \textstyle\sum_i \sum_s \sum_d \sum_t pv(t)^*ReclaimVal(i,s,d,t)^*\mathbf{Z}(i,s,d,t)\end{aligned}$$

With present value factor, $pv(t) = 1/(1+\text{discount rate})^{t-1}$

Value coefficients for each flow variable are computed using multiple lookup tables for costs and for prices and recoveries for e=1 to E elements:

$$MineVal(m,l,p,t) = -cM(m,l,p)$$

$$\begin{aligned}DestVal(m,i,d,t) = &-cX(m,d) + -cDD(i,d)\\&+ \textstyle\sum_e (pr(e,t)-cK(e,i,d))\\&^*(1-rD(e,i,d))^*yD(e,i,d)^*g(e,i)\end{aligned}$$

$$StockVal(m,i,s,t) = -cS(m,s)$$

$$\begin{aligned}ReclaimVal(i,s,d,t) = &-cR(s,d) + -cDS(i,d)\\&+ \textstyle\sum_e (pr(e,t) - cK(e,i,d))\\&^*(1 - rD(e,i,d))^*yS(e,i,d)^*g(e,i)\end{aligned}$$

where:

pr(e,t) = Metal price ($/unit of metal) for element e
cM(m,l,p) = Mining cost (to mine exit) ($/ton)
cX(m,d) = Ex-mine hauling cost (from mine exit to destination) ($/ton)
cDD(i,d) = Destination processing cost ($/ton) for direct feed
cDS(i,d) = Destination processing cost ($/ton) for stockpiled feed
cS(m,s) = Stockpiling cost (including haulage from mine exit to stockpile) ($/ton)
cR(s,d) = Reclaiming cost (including haulage cost from stockpile to destination) ($/ton)
yD(e,i,d) = Yield (recovery) from direct feed
yS(e,i,d) = Yield (recovery) from stockpile reclaim feed
rD(e,i,d) = Royalty (NSR proportion)
cK(e,i,d) = Marketing cost, including smelting, refining, freight ($/unit of metal)

g(e,i) = grade of element (metal), e, for increment, i

In addition to the cost/value coefficients, the primary data inputs to the model are the grade tonnage information for each increment in each panel, tons(m,l,p,i) and g(e,i), and the total tons in each panel, tons(m,l,p) = $\sum_i$ tons(m,l,p,i).

Capacity constraint equations are defined for each time period in terms of maximums or minimums for:

- Tons mined at each layback, mine or group of mines
- Sink rate (panels/year) for each layback
- Tons, grade, metal, recovered metal processed at each destination or group of destinations
- Stockpile inventory

Example capacity constraint equation for total tons mined from all mines in time period, t:

$\sum_m \sum_l \sum_p \mathbf{W}(m,l,p,t) \leq MaxMineTons(t)$

Example recovered metal constraint equation for a specific metal, destination, time period (e, d, t):

$\sum_m \sum_i g(e,i)*yD(e,i,d)*\mathbf{X}(m,i,d,t)$
$+ \sum_s \sum_i g(e,i)*yS(e,i,d)*\mathbf{Z}(i,s,d,t)$
$\leq MaxDestRecMetal(e,d,t)$

Additional constraints are created for each time period to define a feasible mining sequence and ensure material flows are not adding or subtracting tons:

- Uniform mining of increments within a panel: if a panel is only partially mined during a period, then all increments in the panel are mined in the same proportion.
- Panel prerequisites: a panel cannot start mining until its overlying panel is completely mined out or any other defined prerequisites are mined out. Example: panel Lay3P5 constrained by overlying panel Lay3P4 and by inner layback's panel Lay2P5.
- Finite resources: mining limited by panel tonnage input data; no negative flows.
- Mine continuity: mining flows equal stockpile flows plus process flows for each increment, time.
- Stockpile continuity: inventory = prior inventory + inflows – outflows for each increment, time; inventories cannot be negative

Example mine continuity constraint equation for each mine, increment, time period (m, i, t):

$\sum_l \sum_p \mathbf{W}(m,l,p,t)*[tons(m,l,p,i)/tons(m,l,p)]$
$= \sum_d \mathbf{X}(m,i,d,t) + \sum_s \mathbf{Y}(m,i,s,t)$

The Peakfinder MILP model is similar to other MILP production scheduling formulations. Many examples, with varying assumptions, are presented in Dimitrakopoulos, 2018. The Peakfinder model is most similar to a formulation used at Newmont's Nevada Operations (Hoerger et al, 1999), but with expanded mine resolution, sink rate constraints, multi-metal revenue computations and changes to the variable formulation to significantly reduce the number of flow variables.

The Peakfinder model has several features that may show differences to other specific MILP models: increment destinations are computed during the optimization process; all constraints are explicitly defined and enforced without use of penalties; the model uses pre-defined laybacks as inputs; increment proportions are mined uniformly if partial panels are mined during a period; increment grades are defined constants for all flows; to maintain linearity, the model segregates increments in stockpiles rather than forcing blending during reclaim. Most importantly, note that mine sequence, cutoff grades and process/stockpile sequences are all outputs of the optimization process, not inputs.

As with other mine engineering analyses, mining and processing unit costs are key inputs. Unit costs should include all spending, regardless of accounting classification, to be incurred if the mine life, process life or site footprint are extended. This typically includes operating costs, sustaining capital, site overhead, site administration, community development costs, reclamation costs and almost any other spending that is not part of the initial site development capital or part of a profit-based tax. The model formulation allows for fixed yearly costs, but for operations constrained by mill capacity, the same schedules are obtained when fixed yearly costs are included in the milling unit cost.

The workflow used for optimizing a production schedule consists of 1) layback designs and grade-tonnage computations for each panel are designed in a mine design software package; 2) a C program, Peakfinder, formulates the MILP optimization problem; 3) CPLEX, a commercial MILP solver, computes the optimum solution to within 0.01% of optimality; 4) Peakfinder reads the CPLEX solution for reporting and analysis; 5) Excel is used for graphing and comparison of alternative plans. The process described could also be implemented with commercially available production scheduling software.

PRODUCTION SCHEDULING CASE STUDY BASE CASE

The workflow was applied to evaluating value creation opportunities for an operation consisting of a copper mine and mill. Details of the example, including assuming all blocks have sulfide geometallurgy, have been modified for confidentiality and simplification of presentation. With the cost and price assumptions and multipliers used (Table 1), the breakeven cutoff for mill vs waste is 0.4% Cu. The base plan has been solved using base case mill capacity of 20Mt/year and mine capacity of 40Mt/year constrained to a sink rate of six 15 m benches/year vertical advance in any given layback.

Table 1. Case study cost and price inputs

Parameter	Value	Units
Discount Rate, pv	10%	/yr
Price, pr	6500	$/t Cu
Marketing, cK	1100	$/t Cu
Royalty, rD	0%	
Mining, CM	2.50	$/ton mined
Increase w/depth	0.03	$/bench/ton minec
Ex-mine mill, cX	0.50	$/ton hauled
Ex-mine waste, cX	0.10	$/ton hauled
Mill-direct, cDD	17.00	$/ton milled
Mill-stkpile, cDS	17.00	$/ton milled
Stockpiling, cS	0.15	$/ton stockpiled
Reclaiming, cR	0.60	$/ton reclaimed
Recovery-direct, yD	85%	
Recovery-stkpile, yS	80%	

Recoverable resources were provided by conditional simulation calibrated to expected ore control selectivity. For this study, the results of nine conditional simulations were accumulated from all simulations of every block within each panel of each layback. This grade-tonnage reporting method ensures that the planning process reflects the same mining selectivity as the conditional simulations. Tonnages are tracked using 20 increments to provide both tonnage and grade resolution (Figure 2). The geologic model reflects strong copper mineralization surrounded by barren wall rock.

A nested pit approach was used to define 10 laybacks (Figure 3). Panels were defined as the sum of two consecutive benches within a layback. More discussion of layback definition is presented later in the paper.

A series of dashboard graphs shows the results of the production schedule optimization with base case mining

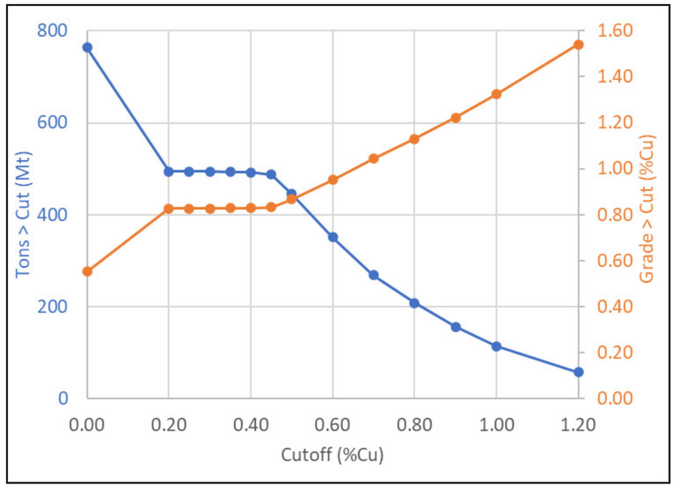


Figure 2. Case study grade-tonnage distribution

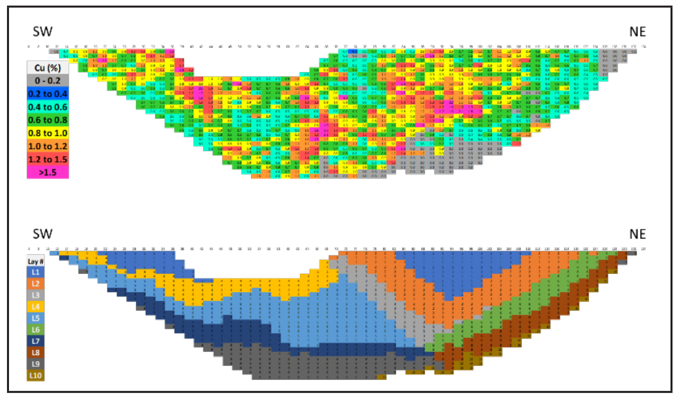
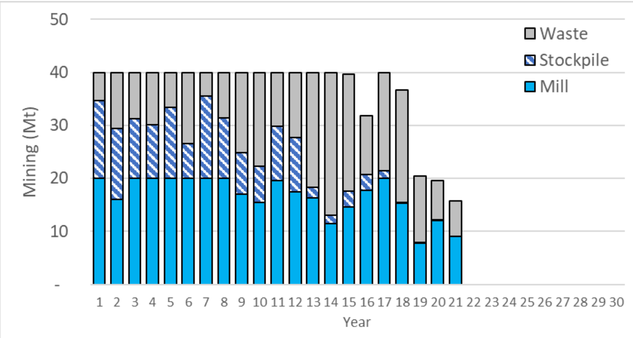


Figure 3. Case study long section (NE-SW): Cu grades (simulation #6) (top) and layback definitions (bottom)

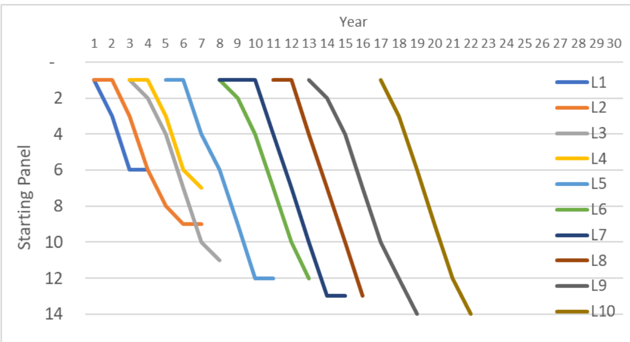
and milling capacity constraints (40Mt/yr and 20Mt/yr). The base mining schedule shows full utilization of mining capacity through year 15 with lower grade tonnages stockpiled in the first twelve years (Figure 4a). Alternate display formats emphasize vertical advance (Figure 4b), or tons per layback (Figure 4c) and how many laybacks are active in a given year (Figure 4d). Starting in year 16, the mining rate is scaled back below the maximum possible rate because the remaining material in laybacks 9 and 10 has a low profit margin which provides little incentive for mining faster and elevating the cutoff grade to accelerate metal production at the expense of higher current mining and stockpiling costs and eventual lower recovery for stockpiled ore.

The results of the mining schedule can also be mapped back to the block model for 3D viewing or cross-section viewing (Figure 5).

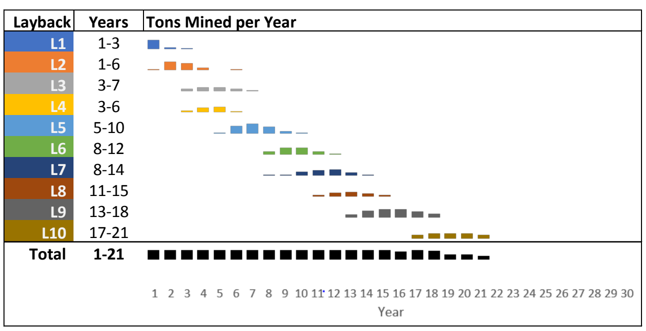
The process schedule shows 24.5 years of full mill capacity utilization, with only limited stockpile reclaim through year 17 (Figure 6a; direct feed from pit color coded by grade



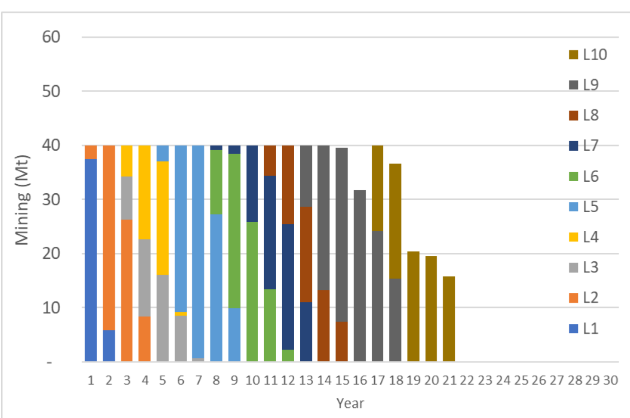
4a) Yearly tons mined



4b) Layback vertical advance



4c) Layback tons per year



4d) Yearly tons mined by layback

Figure 4. Base case mine schedule dashboard graphs

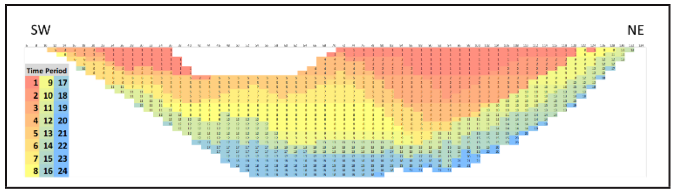
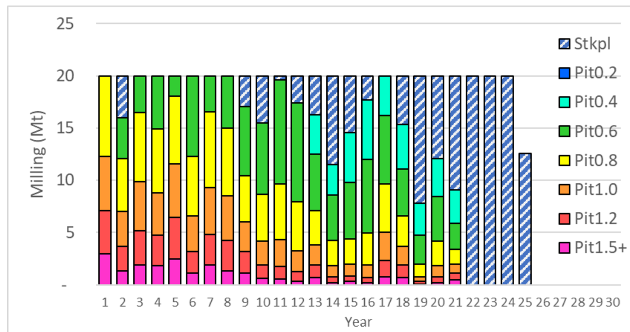
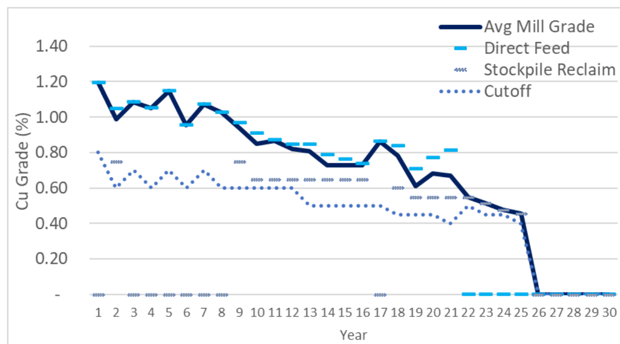


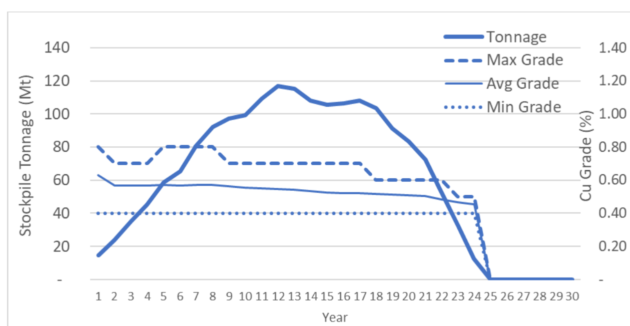
Figure 5. Base case mine schedule mapped to blocks—cross section showing time period mined



6a) Yearly tons milled—direct feed from pit (color coded by grade category) and stockpile reclaim (hatched)



6b) Yearly mill grades



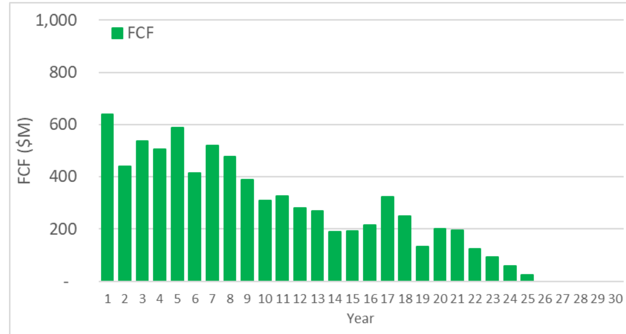
6c) Stockpile grades and inventories

Figure 6. Base case process schedule dashboard graphs

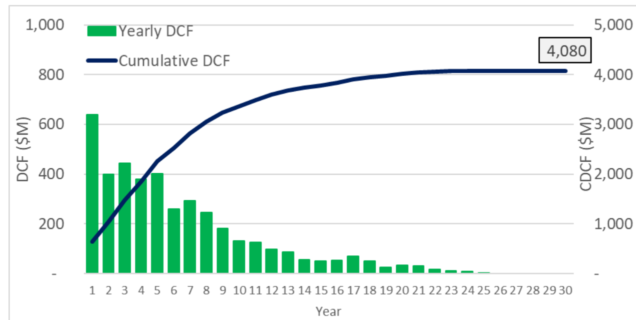
increments with stockpile reclaim shown as a hatched bar). With both mining and milling as bottlenecks, the operating cutoff becomes a function of the grade tonnage distribution mined during each year. The operating cutoff starts at 0.8% Cu in year 1 and bounces between 0.6% and 0.7% through year 12 (Figure 6b). Through year 8, the operating cutoff balances mine and mill capacity without any need for stockpile reclaim, except when the 0.7% to 0.8% increment stockpiled in year 1 is milled in year 2. Some intermediate grade increments are also reclaimed from stockpile in years 9, 10 and 11 before the operating cutoff grade drops to 0.5% Cu in year 13. Throughout the 21 years of mining, any material between the operating cutoff and the 0.4% stockpile cutoff is stockpiled for processing at the end of the mine life. Stockpile inventory peaks just below 120Mt in year 12 as layback 7 nears completion (Figure 6c).

Dashboard graphs are best viewed simultaneously on a large screen monitor which also allows easy comparison and highlighting of differences between plans. Note that the operating cutoff terminology is useful for explaining the schedule results even though the MILP model does not explicitly solve for a cutoff, but instead chooses an optimum destination for each increment in each time period.

The cashflow schedule (Figure 7a) shows significant drop-offs in yearly cash flow after years 1, 5 and 9 corresponding to depletion of laybacks 1, 4 and 5. With a 10% per year discount rate, discounted cash flow drop-offs are further accentuated (Figure 7b).



7a) Undiscounted cash flow by year



7b) Discounted cash flow by year and cumulative

Figure 7. Base case cash flow dashboard graphs

EVALUATING CAPACITY AND COST/ REVENUE CONTINUOUS IMPROVEMENT OPPORTUNITIES

Starting from the base case schedule, alternate production schedules were optimized, and corresponding incremental value creation numbers were evaluated for various combinations of 10% increases in mining capacity, sink rate and milling capacity (Table 2). For perspective, incremental value creation was also computed for 5% reductions in mining and milling costs per ton, a 1% increase in Cu price and a 10% reduction in Cu metal lost to tailings (equivalent to increasing direct mill feed recovery from 85% to 86.5% and stockpile mill feed recovery from 80% to 82%).

Table 2. Value Creation from Continuous Improvement

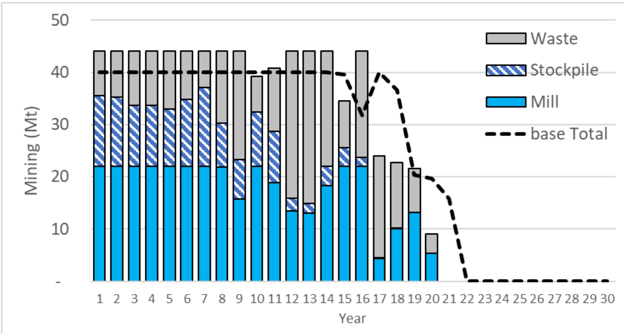
Base Case	NPV	Diff vs Base Case	
40Mt/yr mining; 20Mt/yr milling	4,080	-	0.0%
Continuous Improvement Levers	NPV	Diff vs Base Case	
CI - Physicals			
Mine t/yr +10%	4,122	42	1.0%
Panels/yr +10%	4,083	4	0.1%
Mill t/yr + 10%	4,235	155	3.8%
Mill t/y, Mine t/yr & Panels/yr +10%	4,295	215	5.3%
Tails grade -10%	4,283	204	5.0%
CI - Costs/Revenue			
Mine cost/t -5%	4,131	51	1.3%
Mill cost/t -5%	4,249	169	4.1%
Copper price/t +1%	4,183	104	2.5%

Capacity Improvements

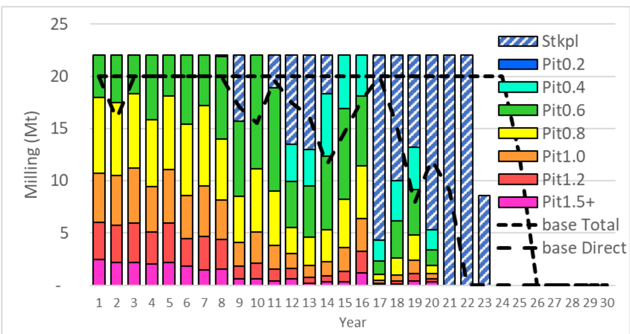
A 10% increase in mining capacity alone improved value by 1.0% by allowing higher grade ore to be mined and milled sooner at the expense of higher yearly ore and waste mining costs and more stockpiling of ore at the operating cutoff margin. A 10% increase in sink rate only yielded a 0.1% increase in value as sink rate was only a bottleneck for a small number of layback/time period combinations. A 10% increase in milling rate improved value by 3.8% by allowing ore with grade just below the operating cutoff to be milled in the same period mined instead of being stockpiled; this benefit is only partially offset at the end of the mine life when the mill shuts down earlier.

Figure 8 shows how the plan changes with simultaneous 10% increases to mining, sinking and milling capacity. The mining capacity increase is fully utilized through year 9 (Figure 8a). With higher mining, higher mill capacity can be fully filled from direct pit feed through year 8 (Figure 8b).

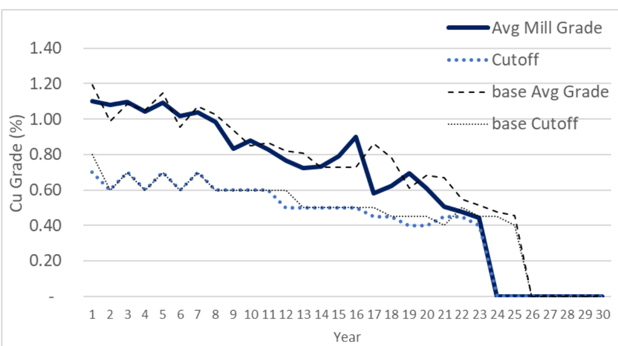
With simultaneous increases to mining, sinking and milling constraints, some synergy is seen with the total



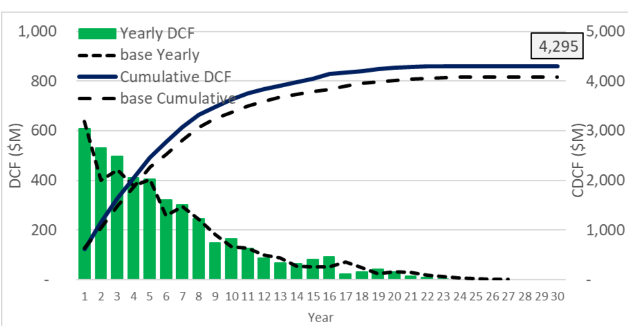
8a) Yearly tons mined comparison



8b) Yearly tons milled comparison



8c) Yearly mill grade comparison



8d) Discounted cash flow profile comparison

Figure 8. Continuous Improvement comparison highlights: solid lines=+10% mining tons, milling tons and panels per year; dashed lines=base case plan

value creation (5.3%) being greater than the sum of the three individual parts (4.9%). With the higher mine tonnage, the higher mill tonnage can be provided without lowering the operating cutoff grade significantly in any year and generally maintaining (years 1 to 7) or increasing mill grades (years 8 to 14) (Figure 8c); with a net result of higher net present value (Figure 8d).

Cost and Revenue Improvements

The cost and revenue improvements have higher leverage to value creation than the capacity increases (Table 2, bottom section). A 5% reduction in mining cost per ton increases value by 1.3%; a 5% reduction in milling cost per ton increases value by 4.1% and a 1% increase in copper price increases value by 2.5%. The cost and revenue improvements were small enough that they created little impact to the optimal production schedule; however, larger improvements could be enough to raise operating cutoffs and reduce stockpiling breakeven cutoffs and even make new laybacks profitable.

In this example, lowering mining cost/ton created significantly more value than expanding mining capacity at a constant cost/ton. The author is looking forward to testing a high strip ratio gold deposit case study where the relative impacts may be different. Depending on the improvement project, increasing tons per year may be an effective way to reduce cost per ton.

EVALUATING MINE PLANNING IMPROVEMENT OPPORTUNITIES

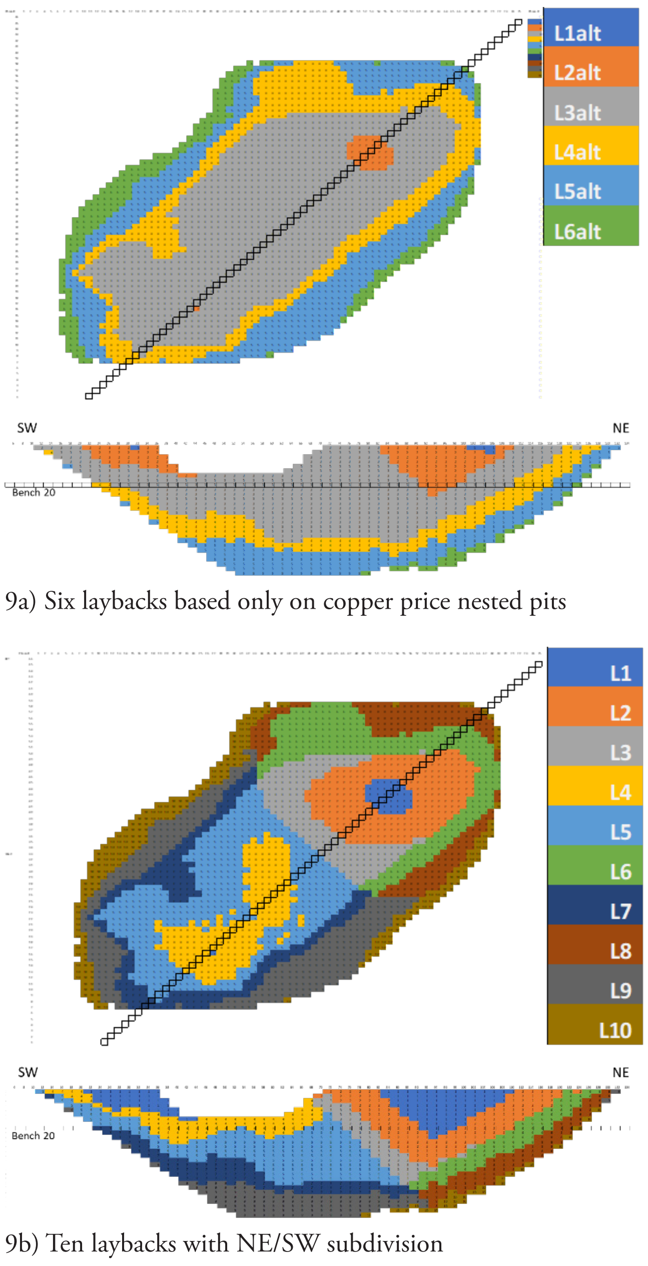
To provide additional perspective, the Peakfinder MILP production scheduling workflow was also used to demonstrate the importance of several mine planning levers. By necessity, the analyses demonstrate the value lost if not using several mine planning techniques incorporated into the base plan: subdivision of large laybacks, stockpiling, and dynamic cutoff grades. Results are presented in Table 3. Note that, although the stockpiling lever does not show a significant increase in Net Present Value, stockpiling does preserve significant resource tonnage and can provide significant option value.

Table 3. Value Creation vs Mine Planning

Mine Planning Levers	NPV	Diff vs Base Case	
Layback Design			
Don't subdivide large laybacks	3,966	(114)	-2.8%
Stockpiling			
No stockpile	3,978	(101)	-2.5%
Max stockpile 40Mt	4,043	(37)	-0.9%
Cutoff Grade Optimization			
Breakeven fixed cutoff (0.4%)	3,648	(431)	-10.6%
Best fixed cutoff (0.6%)	3,840	(240)	-5.9%

Subdividing Large Laybacks

An initial set of six laybacks was defined based on nested pits corresponding to several increasing copper prices. This resulted in six laybacks with 4, 39, 274, 154, 200 and 93 million tons respectively. For the base plan, laybacks 1 and 2 were combined, layback 3 was split into four pieces, laybacks 4 and 5 were split into two pieces, resulting in ten laybacks with 43, 71, 47, 45, 110, 82, 72, 58, 142 and 93 million tons respectively (Figure 9).



9a) Six laybacks based only on copper price nested pits

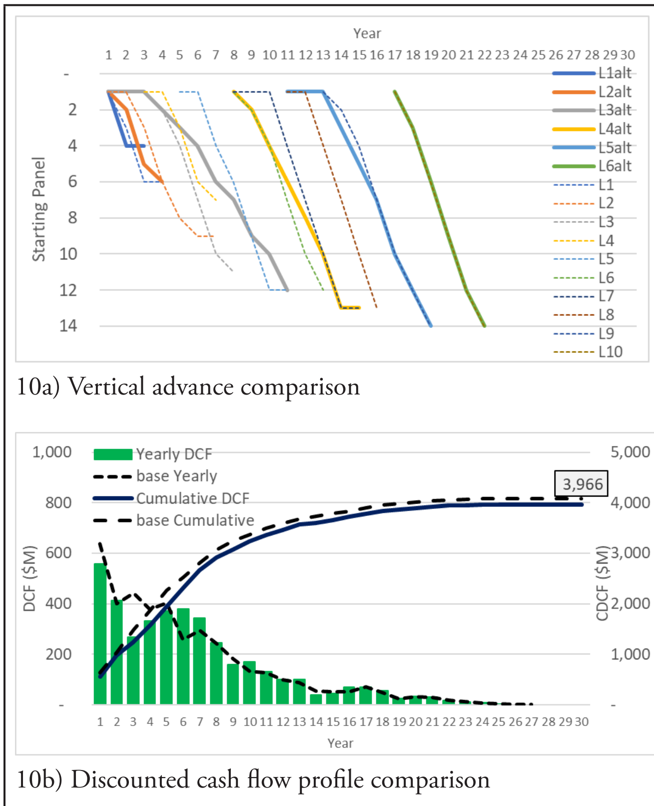
9b) Ten laybacks with NE/SW subdivision

Figure 9. Layback design alternatives: plan views (bench 20 slice) and NE/SW sections

The subdivision was achieved by intersecting the first set of nested pits with additional sets of nested pits which only assigned value to blocks on the northeastern or southwestern halves of the deposit. The 2.8% value loss for the larger laybacks is the result of slower access to higher grade ore at the bottom of layback L3alt (Figure 10a) and corresponding deferral of cash flows from years 1–4 into years 6+ (Figure 10b). For different deposits with a more dramatic distinction between stripping at the top and ore zones at the bottom, subdivision of large laybacks would be expected to have a larger value impact and minimize or eliminate periods of negative cash flow by limiting the duration and peak amount of stripping investment before a layback's ore is exposed.

Stockpiling

The no stockpile case only loses 2.5% of life-of-mine value because the stockpiled ore has a small margin above break-even and because of heavy discounting since most of the stockpiled ore won't be processed until the end of the mine life. In addition to some NPV benefit, stockpiling may be desirable for: resource preservation (12% more ounces produced vs no-stockpile case); providing a backup ore source



10a) Vertical advance comparison

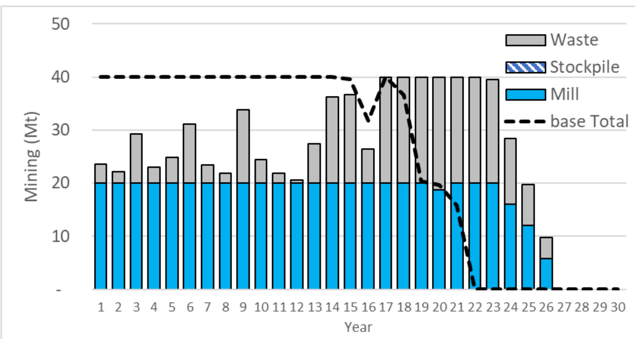
10b) Discounted cash flow profile comparison

Figure 10. Alternate laybacks comparison highlights: solid lines=six laybacks; dashed lines=base case ten laybacks

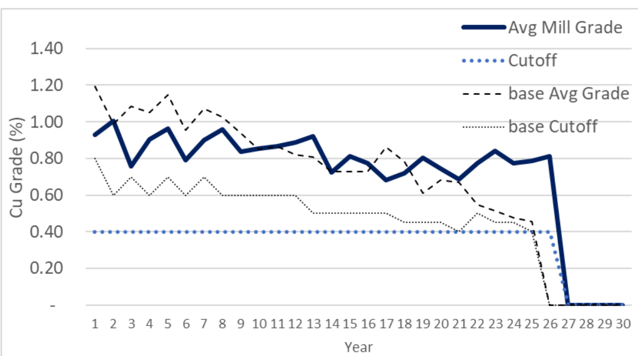
during periods of low model reconciliation or mining interruptions; providing option value at the end of the mine life when the stockpile can be processed if prices are high or left on the ground if prices are low.

Dynamic Cutoff Grades

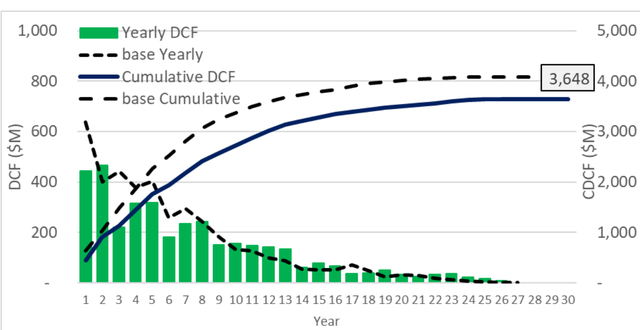
If the operation wants every ton above a breakeven cutoff to be milled in the same period mined, the mining rate must be throttled back dramatically until starting to mine the outer laybacks (Figure 11a). Each ton processed at breakeven cutoff grade provides negligible current year profits while consuming limited mill capacity and deferring profits from future years to reduce NPV by 10.6%. Compared to the base case's dynamic cutoff grades, which change with each time period, the fixed breakeven cutoff case shows significantly lower mill grades (Figure 11b) and corresponding sacrificed cash flow and NPV (Figure 11c) in each of the first nine years. As a side note, in this breakeven cutoff case where mining is not a bottleneck, increasing mining capacity would have little to no continuous improvement value.



11a) Yearly tons mined comparison



11b) Yearly mill grade comparison



11c) Discounted cash flow comparison

Figure 11. Fixed cutoff comparison highlights: solid lines=fixed breakeven cutoff grade; dashed lines=base case with dynamic cutoff grade and stockpiling

EVALUATING GEOSTATISTICS IMPROVEMENT OPPORTUNITIES

Mining selectivity can have a significant impact on the grade and tonnage of ore above cutoff and consequently on the value of an operation. The base plan model reflects a probabilistic estimate of mine selectivity, with each of the nine simulations honoring all drilling information and identified spatial continuity trends to provide multiple equally probable views of the distribution of grades throughout the deposit. For grade tonnage reporting, each block of each simulation is tested against the increment histogram bin limits and the total accumulated tonnage in each bin is divided by the number of simulations; for purposes of this paper, this model is called the IK (for indicator kriging) model. To investigate the impact of lower selectivity, two alternative models were created.

E-Type Estimate (Ok)

First, each block was assigned the grade of the average of the nine simulations; this e-type (e for expected value) model is called the OK (for ordinary kriging) model. Figure 12 illustrates the averaging process for a sample bench and the resulting smoothing of grades.

Diluted Simulation (IKD)

Second, a diluted model was created for each simulation by computing a weighted average of the block grade and the grades of the four neighboring blocks that share a common edge; this diluted model is called IKD and uses the same grade tonnage reporting process as the IK model. Figure 13 illustrates the process for a sample bench of a sample simulation. The assigned weighting (68% for the central block and 32% split between the four adjacent blocks) was chosen to achieve an arbitrary 5% reduction in the grade distribution's standard deviation; conceptually, the 32% weight would correspond to an additional 2.35 meters of mixing around the 80 m perimeter of a 20 m x 20 m block.

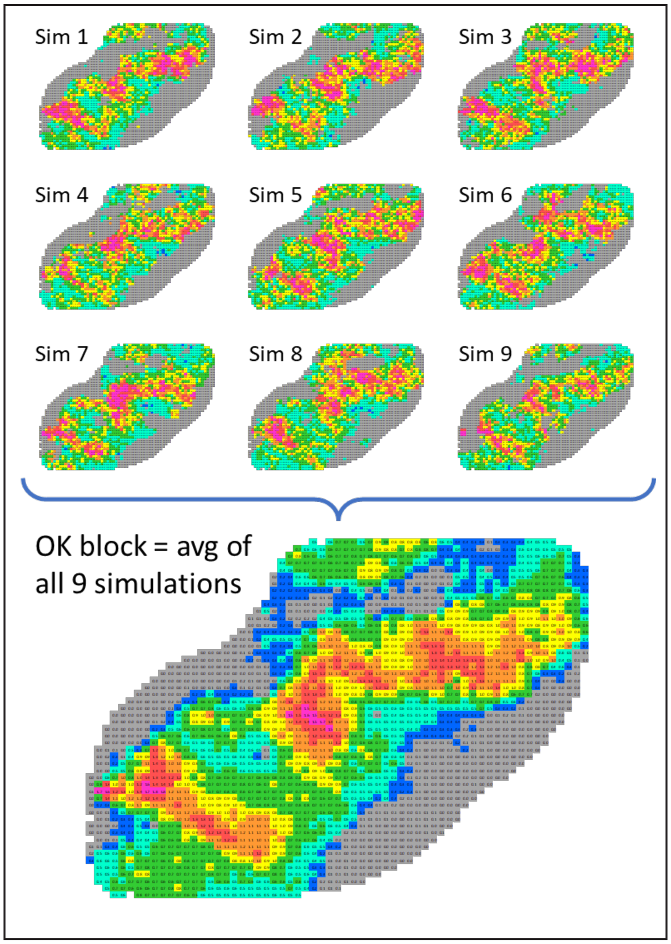


Figure 12. Creation of OK model from simulations—example for bench 20

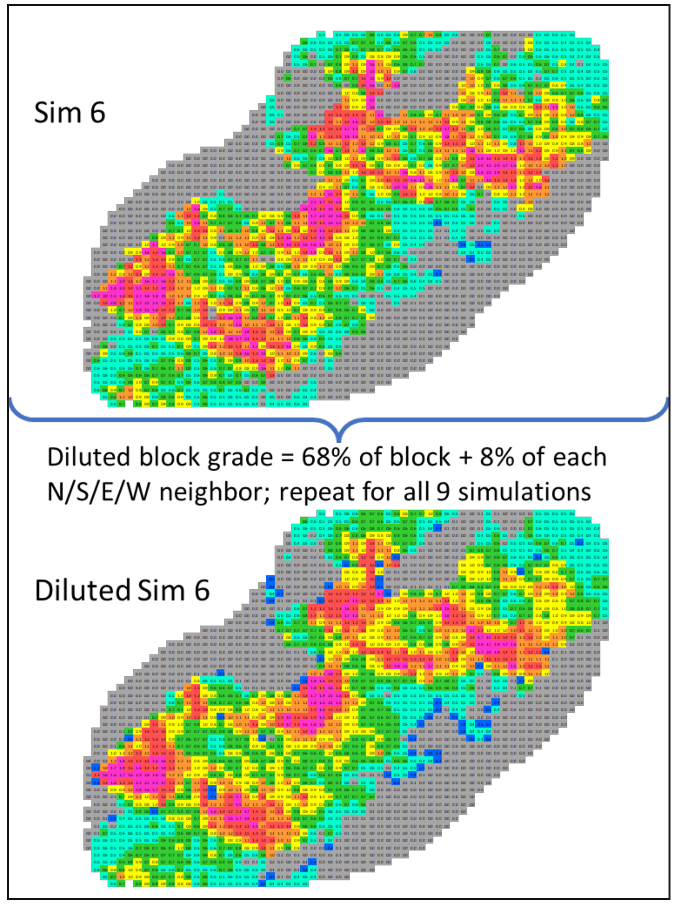


Figure 13. Creation of diluted simulations—example for simulation #6, bench 20

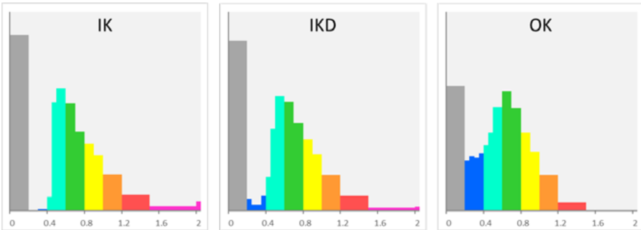
The alternate models show how a small reduction (IKD) or large reduction (OK) in selectivity can impact the grade distribution (Figure 14a). If tons sent to the mill are at 50% of mining capacity, the IKD model shows 3% lower mill grade and the OK model shows 9% lower mill grade (Figure 14b).

Planned NPV sensitivity to block models

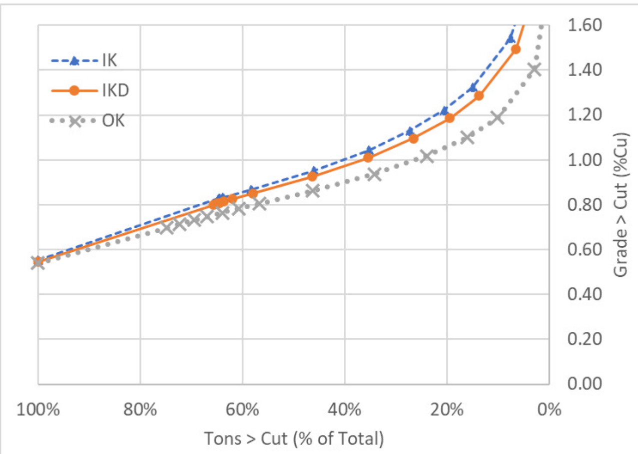
Plans built with the three models show significantly different NPV's, with the OK based plan showing a 16.8% lower NPV than the IK plan (which can also be expressed as IK is 20% higher than OK) and the IKD based plan showing 5.2% lower NPV than the OK plan (Table 4 top section). These planning-stage NPV differences are large and can have a significant impact on project prioritization and whether projects are executed or not.

Realized NPV sensitivity to plans built from different block models

If a project is executed, the realized NPV will depend upon the actual ore control selectivity rather than the selectivity that is modeled in the block model. A same-ore-control-selectivity comparison of plans created from alternative blocks models was analyzed by freezing the mining sequence from each plan and then reoptimizing the processing sequence using the IK model (Table 4 middle section). The mining sequence for the IK and IKD plans are very similar, so they have very similar NPV's when evaluated using the same selectivity (IKD based plan 0.1% lower). The OK mining sequence is quite different with the final five phases delayed while stockpiled material is processed; however, when evaluated using the IK model, the NPV reduction is still small (OK based plan 0.8% lower) because the NPV of the last five phases is very similar to the NPV of the material stockpiled while mining the first five phases.

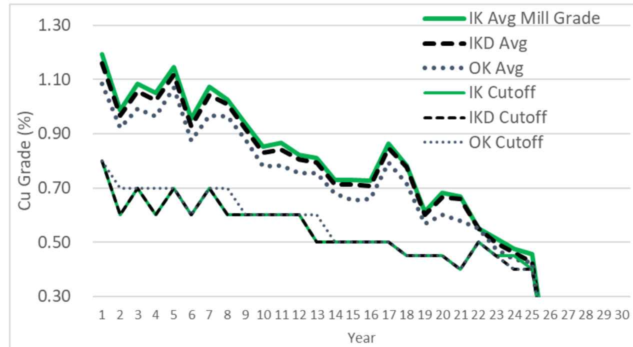


14a) Histograms of alternate copper grade models

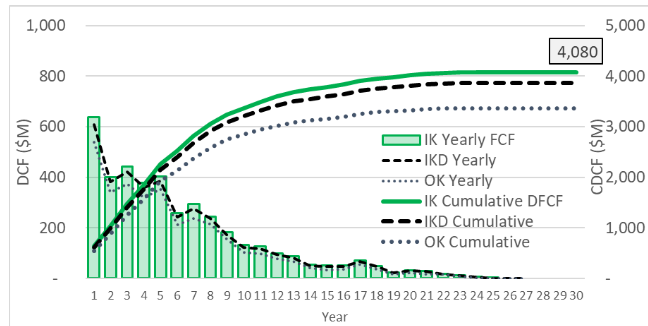


14b) Grade above cutoff for alternate grade models

Figure 14. Selectivity impacts to grade distributions



15a) Yearly mill grade comparison



15b) Discounted cash flow comparison

Figure 15. Selectivity impacts on mill grade and cash flow; mine sequence held constant (plan based on IK model) and process sequence varied by block model

Table 4. Value Creation vs Geostatistics

Geostatistics Levers	NPV	Diff vs Base Case	
Model impact to planned NPV			
OK model plan & selectivity	3,396	(684)	-16.8%
IKD (-5%) model plan & selectivity	3,869	(210)	-5.2%
Model impact to realized NPV (constant selectivity)			
OK model plan w/ IK selectivity	4,047	(32)	-0.8%
IKD (-5%) model plan w/ IK selectivity	4,076	(3)	-0.1%
Selectivity impact to realized NPV (constant mine plan)			
Base Plan (IK) w/ OK selectivity	3,371	(709)	-17.4%
Base Plan (IK) w/ IKD (-5%) selectivity	3,868	(212)	-5.2%

Realized NPV Sensitivity to Ore Control Selectivity

For continuous improvement opportunity testing, a key analysis is to evaluate how ore control selectivity can impact value. To test this question, the mining sequence was fixed by the base plan (built with the IK model) and the processing sequence was optimized using the different block models. The results isolate the dramatic impact that mining selectivity can have on value realized from a mine plan (Table 4 bottom section)—5% lower NPV with the IKD model's selectivity and 17% lower NPV with the OK model's level of selectivity. Figure 15 shows how the lower selectivity model impacts mill grade and discounted cash flows over time.

CONCLUSION

Because an operation only has a finite budget and limited change management bandwidth, continuous improvement projects should be prioritized based on expected value and change complexity. The MILP production scheduling tool can be useful in evaluating the value expected from continuous improvement projects which improve mining and processing constraints. The tool accounts for how improvements will interact over time with the mining sequence, grade-tonnage distributions, cutoff grades and mine/mill constraints. The tool was also used to compute the value resulting from cost per ton improvement projects, mine planning changes, and mine selectivity changes.

For the copper mine example presented, cost per ton improvements showed more benefit than tonnage capacity increases. Mine planning levers and mine selectivity levers showed similar or larger potential value impacts than cost or capacity improvement levers. Mine planning and mine selectivity improvements should also be considered within an operation's portfolio of continuous improvement projects.

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24-059

Mine Ventilation Pathway Simulation on a Hypothetical Shale Gas Well Breach Utilizing the Longwall Instrumented Aerodynamic Model (LIAM)

Robert A. Kimutis
NIOSH, Pittsburgh, PA

Marcia L. Harris
NIOSH, Pittsburgh, PA

Steven J. Schatzel
NIOSH, Pittsburgh, PA

James D. Addis
NIOSH, Pittsburgh, PA

Mark M. Mazzella
NIOSH, Pittsburgh, PA

Mayley Guitard
NIOSH, Pittsburgh, PA

Vasu Gangrade
NIOSH, Pittsburgh, PA

Abstract

Researchers from the National Institute for Occupational Safety and Health (NIOSH) evaluated the impact of potential shale gas well breaches and subsequent gas inflow on selected mine ventilation systems of operating longwall panels using a 1:30 scaled Longwall Instrumented Aerodynamic Model (LIAM) and Sulfur hexafluoride (SF_6) as a tracer gas. A series of tests were performed at 340, 400, and 500 cfm inflow levels utilizing different mine ventilation scenarios. Results suggest that the breached gas can be diluted to meet statutory levels when the longwall panel is adequately ventilated. The results enhance the understanding of the gas inflow and mine ventilation system interaction and provide critical information to the industry and regulatory agencies for improving miners' safety.

INTRODUCTION

In the tristate area of Pennsylvania, West Virginia, and Ohio, unconventional shale gas wells have been drilled through current and future coal reserves. Impacts on the mechanical integrity of these wells becomes a concern when mining occurs in and around these wells. The shale gas wells can potentially penetrate the coal seams and the coal seams are subsequently mined. Therefore, it is imperative that the coal and shale gas industries to know what the impacts of longwall mining may be on the shale gas wells, what the potential deformation may be, and what stresses are imposed on the gas wells.

In 2012, the Pennsylvania Department of Environmental Protection (PADEP) recognized that the 1957 Pennsylvania Gas Well Pillar Regulation (Commonwealth of Pennsylvania, 1957) was created with limited data from modern day longwall mining and called for research to revise the outdated regulation. Therefore, given the posed questions and the need for further guidance given modern mining technologies and practices, the National Institute for Occupational Safety and Health (NIOSH) initiated research to address these issues.

To this end, several research efforts have focused on predicting the potential quantity of methane that could enter an underground coal mine environment if a shale gas well casing should be breached. One such area is gas permeability in ground strata. Measurements of the permeabilities of the ground strata under varying depths of cover are being conducted. Current research shows that shallow cover conditions (<500 ft depth to the mined coal seam) have the highest permeability compared to deep cover sites [1, 2]. Researchers have modeled and predicted the ground permeabilities which are well aligned with the field measurements [3]. Based on the collected field measurements, researchers modeled and predicted quantities of methane