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--Manuscript Draft--

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Thermal Fragmentation as a Viable, Alternative Mining Method in Narrow Vein Mining

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Abstract

In currently used mining methods, blasting techniques are often lacking in selectivity, which causes dilution of valuable ore and results in costly processing requirements. In the context of narrow vein mining of thin and highly concentrated orebodies there is a need of a mining method that can reduce dilution in order to remain economically viable.

This research project explored the viability of a new mining technology, thermal fragmentation, in narrow vein mining. Thermal fragmentation technology uses a flame jet to produce extreme heat that spalls the surrounding rock to a strategically located drill hole, as an alternative to traditional blasting. By creating a Net Present Value (NPV) model of a mining method using thermal fragmentation, as well as for an existing method used for narrow vein mining, comparisons and evaluations were made regarding the feasibility of the new technology. It was found that while overall costs for thermal fragmentation were relatively high, reductions in wages, haulage and processing costs, as well as increased productivity and ore recovery, meant that the new method would improve the financial performance of any operation.

These results identify that there is the opportunity to introduce thermal fragmentation technology to narrow vein mines within Australia, in order to lower costs and increase profit.

1. Introduction

1.1 Narrow Vein Mining

Narrow vein deposits are defined by Stewart (2005) as mineralisation zones that are no more than two to three metres wide and with a dip of more than 50 to 55 degrees. Of the narrow vein deposits across the globe, ones possessing high gold content are particularly important due to the high unit value of gold in such low volume orebodies (Dominy et al. 1998).

Improvements in technology and practices over the last 70 years has resulted in various mining methods, however, the mining of narrow vein deposits is still not without its economic challenges. All the methods employed are in general associated with high operating costs and low productivity, and often perform poorly in terms of safety (McCarthy, 1993). Several difficulties that are faced by mining operations wishing to successfully mine narrow vein orebodies are listed by Dominy et al. (1998b), including:

- Difficulty in assessing reserves due to complex geology and distributed mineral grade;
- High degrees of planned and unplanned dilution from surrounding host rock;
- Low ratio between the tonnage of ore produced and the metres developed;
- Low operating efficiencies due to relatively small ore production; and

- Minimal mechanization of operations due to the nature of the suitable mining methods.

1.2 Conventional Mining Methods

The extraction of narrow veins is as old as the mining industry itself. Hall (1987) makes reference to over 10 separate methods used in steeply dipping narrow vein mines encountered across the world. Each of these procedures would contain characteristics that make the method ideal for the particular mine. In order to compare methods, Vergne (2014) categorises all hard rock mining methods as either a bulking or selective method. In relation to narrow vein mining, longhole stoping would be considered a bulk mining method, with resue and shrinkage described as selective methods (Vergne, 2014).

1.3 Stope Dilution

In the context of narrow vein mining, dilution can be defined as the contamination of ore with below cutoff grade material and can further be separated into internal and external dilution (Tatman, 2001). Internal dilution is described as cutoff-grade material that is present inside the stope boundaries (Stewart, 2005). Conversely, external dilution is described as unplanned inclusion of waste rock that can be produced by blast overbreak, sloughage, or backfill (Stewart, 2005). An illustration of planned and unplanned dilution is shown below in Figure 1.

Figure 1 - Planned and unplanned dilution (Stewart, 2005).

For some operations, dilution can mean the difference between being profitable and non-economically viable. There are many ways in which dilution can affect profit. Greater quantities produced from the stope means more bogging, more haulage cycles, and in turn, greater consumption of fuel and wear on equipment. The processing plant is also significantly affected by the increased tonnage associated with dilution. Dilution is one of the most frequent reasons for mine closure (Miller et al. 1992).

The ability to measure and predict dilution is important for finding the right balance for each operation. Equation 1 and 2 are two of the most commonly used equations for quantifying dilution (Stewart, 2005).

$$Dilution = \frac{Waste\ Tonnes}{Ore\ Tonnes}$$

Equation 1

$$Dilution = \frac{Waste\ Tonnes}{Ore+Waste}$$

Equation 2

Pakalnis, Poulin, and Hadjigeorgiou (1995) advise that Equation 1 should be taken as the standard measure of dilution, due to Equation 2 being insensitive to wall slough.

The method chosen for mining can greatly impact the extent of dilution. Applied to a narrow vein orebody, bulk mining methods such as long hole stoping, have high degrees of dilution. Dilution for bulk mining methods can be as high as 20 to 25 percent (Vergne, 2014). In comparison, selective methods such as resue firing and shrinkage stoping are associated with a decrease in dilution.

1.4 Thermal Fragmentation

A current innovation in the mining industry, and in particular underground narrow vein mining, is the fragmentation of rock by use of extreme heat. While the use of thermal fragmentation in its current state is a new development, thermal spalling of rock has been in practice for over a century. In the mid-1800s, rudimentary forms of thermal fragmentation were used throughout Scandinavia in underground mining operations, but declined with the introduction of explosives before being abandoned in the 1880s (Marovelli at el. 1966). More recently, thermal fragmentation has been used as a primary drilling method for over 30 years in large-scale open pits in Russia (Poirier at el. 2002).

With the objective of improving and adapting the Russian technology for North American conditions, Canadian organization, Rocmec International Inc., was founded in 1995 (Poirier et al., 2002). The most significant element of the technology, the burner, creates a jet of flame using diesel fuel and compressed air. The heat created in a confined space by the jet, reacts with the surrounding rock to create a spalling effect (Brisebois & Brisebois, 2010). In practice, a pilot hole with an approximate diameter of 158 mm is first drilled. The burner is then inserted the full length into the hole and pulled back in stages, in order to expand the width of the hole to 30-80 cm (Brisebois & Brisebois, 2010). The duration in which the burner stays in place for each stage, as well as the length of each retreat, is termed as the recipe.

The potential for such a technology in the context of narrow vein mining, is a solution to the unresolved problem of stope dilution. For minimum dilution, the stope width can be greatly reduced when compared to currently employed extraction techniques. Unplanned dilution control is also reduced as the resulting hole face is smooth and well defined. Additionally, the versatility of thermal fragmentation allows the operator to create bulges or bottle necks in individual sections of a hole to better adapt to the individual vein characteristics.

2. METHODOLOGY

A net present value (NPV) model was created for the thermal fragmentation method as well as for another narrow vein mining method. The aim was to develop a practical and optimised model for each method. The modelling can be further broken down into individual elements. Once the parameters of the design had been outlined, physicals and resources of the mine could be quantified and used to develop a schedule. Along with cost estimations, this schedule was used to develop the NPV model.

2.1 Design Parameters and Mine Physicals

The site chosen to be the focus of the study was the One Tree Hill Mine, Victoria, Australia. The One Tree Hill site was first mined in the 19th century and has in recent times been re-opened for small-scale production. Old style technologies such as pneumatically powered equipment is currently being used in the operation. The mine operator now plans to increase the mine in scale and productivity.

Currently, the One Tree Hill mine employs the shrinkage stopeing method of narrow vein mining. Hence this mining method was used to compare with the thermal fragmentation alternative in the NPV modelling. Both methods were modelled using a design for future mine workings that was provided by the mine operator. The design aimed to access four separate stopes; named the buck, the butch, the mystery, and the mystery central. The development and production areas were separated into physical sections. These physicals were evaluated for size and direction characteristics, as well as grade for the production physicals. For the production physicals the capabilities of the methods were taken into consideration to evaluate the dilution and effective size characteristics of the individual stope sections. From this, calculations could be made of the waste, tonnage and gold ounces of each section of development and production respectively. Dilution was calculated using Equation 1.

2.2 Development Duration

In order to develop a mining schedule, durations for the mining of each physical were calculated. Each section of development was divided into separate cuts and the cycle time for each cut to be extracted was summed. For a standard cut, the cycle time comprises of drilling, charging, blasting and ventilation, bogging, and support installation. For charging, blasting and ventilation, and support installation, standard durations were assumed as 0.5, 1, and 1.25 hours respectively.

The drilling time per cycle (Equation 3) was calculated by dividing the number of holes per cut, by the number of holes that could reasonably be drilled per hour. For the development sections associated with the mystery and butch stopes, the assumption was made that the drill rate would be less than other sections due to more difficult ground conditions. Bogging times were deemed as being largely dependent on the haulage distance. Due to narrow profiles and relatively small haulage distances, a bogger would be used for haulage to the dumping location outside the mine. Other factors contributing to bogging time included cut volume, swell factor, bucket capacity, transit speed, and loading/dumping time. The Equation 4 used for determination of bogging time per cut and the duration of each development section could then be calculated using Equation 5.

$$\text{Drilling time per cut (h)} = \frac{n_h}{D} \quad \text{Equation 3}$$

where:

n_h - Holes per cut (holes)

D - Drilling rate (holes/h).

$$\text{Bogging time per cut (h)} = \frac{V \times K}{c} \left(t_L + t_d + \frac{x}{s} \right) \quad \text{Equation 4}$$

where:

V - Cut volume (m³)

K - Swell factor

C - Bucket capacity (m^3)

S - Transit speed (km/h)

t_L - Loading time (h)

t_d - Dumping time (h)

x - Haulage distance (km).

$$\text{Total section time (h)} = n_S \times (t_D + t_C + t_V + t_B + t_S)$$

Equation 5

where:

n_S - Cuts per section

t_D - Drilling time per cut (h)

t_C - Charging time per cut (h)

t_V - Ventilation time per cut (h)

t_B - Bogging time per cut (h)

t_S - Support installation time per cut (h).

Similarly to the development sections, the separate ore bodies were broken up into individual stopes. Due to differing capabilities, the mining method chosen also influenced the strike length of each stope. The stope sections were then evaluated for size characteristics and grade. The time it takes to mine each stope, needs to be calculated separately for each of the two chosen mining methods due to a difference in equipment and processes.

2.3 Shrinkage Stoping Duration

For shrinkage stoping, blasting and bogging can be calculated as two distinct cycles, as the majority of bogging would be carried out after the stope has been depleted. Stopping duration was calculated using the individual stope tonnage and the average tonnage blasted each day (Equation 6). For two workers the daily production is assumed, through consultation with the One Tree Hill mine, as 70 tonnes.

The shrinkage stoping method requires daily drawing off (20% of daily production) to occur after a week of production. With the daily bogging tonnage removed, the duration for bogging of the remaining material was evaluated using stope and bogging characteristics. Equation 7 was developed to incorporate these variables into one calculation.

$$\text{Stopping duration (days)} = \frac{T}{P}$$

Equation 6

where:

T - Stope tonnage (t)

P - Production (t/day)

$$\text{Bogging Duration (days)} = D_S + \frac{(4T+7P)}{50 \times C} \left(t_L + t_d + \frac{x}{S} \right) - 7$$

Equation 7

where:

D_S - Stopping duration (days)

T - Stope tonnage (t)
 P - Production (t/day)
 C - Bogger capacity (t)
 t_B - Loading time (h)
 t_S - Dumping time (h)
 x - Haulage distance (km).

2.4 Thermal Stopping Duration

For the given case study the mining method chosen utilised both thermal fragmentation and blasting in order to maximise productivity and recovery. Primary holes would be drilled and thermally fragmented, and the pillars created in between would be blasted. As the hole width requirements for thermal and blasting will be different, therefore thermal production and blasted production need to be treated as separate items with two separate drill rigs in operation at the same time.

The chosen way to undertake this was to have the primary hole drill start one week ahead of the blasthole drill, with thermal fragmentation starting one week after the blasthole drill. The three machines could then work simultaneously with enough spacing in between so as not to interfere with each other as they progress along the drive. As a vacuum is required for effective thermal fragmentation, a 1.5m thick sill pillar above the drive and 1m thick at the top of the stope would be left. After the thermal fragmentation has been completed in a drive; blasting of the stope pillars would be undertaken.

To calculate the time spent drilling blast holes or thermal pilot holes for a given stope section, Equation 8 was used. For thermal fragmentation time, Equation 9 is similar to drilling time with an allowance for the 2.5 m of pillars and the conversion of thermal recipes to rates. Blasting time can be estimated using blasting rate, number of holes, stope tonnage, bogger cycle time and bogger capacity (Equation 10).

$$\text{Drilling time per cut (h)} = \frac{n_h \times L}{D} \quad \text{Equation 8}$$

where:

n_h - Holes per cut (holes)
 L - Hole length (m)
 D - Drilling rate (holes/h).

$$\text{Thermal time (h)} = \frac{R \times n_h \times (L - 2.5)}{60} \quad \text{Equation 9}$$

where:

n_h - Holes per stope section (holes)
 L - Hole length (m)
 R - Thermal recipe (minutes/m).

$$\text{Blasting and time (h)} = \frac{n_h}{B} + \frac{T \times (t_L + t_d + \frac{x}{S})}{C} \quad \text{Equation 10}$$

where:

n_h - Holes per stope section (holes)
 B - Blasting rate (holes/h)

T - Blasted stope tonnage (t)
 C - Bogger capacity (t)
 S - Transit speed (km/h)
 t_L - Loading time (h)
 t_d - Dumping time (h)
 x - Haulage distance (km).

2.5 Scheduling and Resource allocation

Once the durations of the tasks involved in mining each of the physicals had been calculated, an operation sequence was applied to develop the schedule for thermal stoping and shrinkage stoping. Using durations and prerequisites for each task, earliest start dates were calculated, and a Gantt chart was created to help visualise the schedule logic. Due to the difference in mining methods, separate Gantt charts were generated for thermal and shrinkage stoping. Due to the small size of the mine, it was assumed that only one stope would be developed at a time.

While there are many different types of costs associated with the mining methods, all costs could be categorised as either a fixed cost or a variable cost. Fixed cost are the items that cost the same per day regardless of activities carried out. Most of the equipment used in the mine would be hired at a weekly or monthly rate and would therefore have a fixed cost for its duration on site, even when it is not being used. Variable costs are items that are proportionally related to the amount that a resource is used in any given day and is best calculated based on the quantity of physicals mined. Variable costs include the costs of fuel to run equipment, servicing of equipment, and consumable items such as drill bits and tyres.

For variable costs associated with maintenance, the time that each resource is required for a certain physical is multiplied by a maintenance cost and then divided by the tonnage of the physical (Equation 11). Maintenance costs are the approximate summation of mechanical upkeep and consumables required for daily use. Variable costs related to fuel use are dependent on the fuel consumption, fuel price and required hours of operation. The exception to this equation is for single use consumable resources such as explosives or supports. Costs associated with these are calculated by multiplying the quantity of units required to mine a physical, multiplied by the cost per unit.

$$\text{Variable Costs} = \frac{m \times t_T}{T} + \frac{F_P \times F_C \times t_T}{T} \quad \text{Equation 11}$$

where:

m - Maintenance costs (\$/h)
 t_T - Total time required (h)
 T - Tonnage
 F_P - Fuel price (\$/L)
 F_C - Fuel consumption (L/h).

Fixed costs apply to all resources except for consumables, and resources that had been purchased as capital expenditure. For the resources that have fixed costs, two categories apply: rental costs and wages. Rental costs are multiplied by the length of time they are on site. Wages are given to the workers on site at an hourly rate. Different rates would apply

to the different levels of work being conducted. Calculation of wages was undertaken by multiplying the number of workers in a given wage level by their pay rate, summed for all wage levels.

2.6 Net Present Value

The last stage in modelling the two mining methods before comparisons could be undertaken, was to sum the income and costs to determine a yearly cash flow. The first step undertaken in the calculation of cash flow was to divide development and production tonnages into the year they were mined, according to the schedule. Similar to production tonnage, the ounces of gold produced were summed given the approximate grade of each stope. By multiplying the ounces produced by the current gold price, the forecast yearly income for both mining methods could be calculated.

The estimated yearly costs for both mining methods result from five sources: wages, fixed resource costs, variable resource costs, processing, and haulage. While wages were previously included as a fixed resource cost, they were separated in the cash flow calculations for the benefit of future analysis. Processing costs were added under the assumption that the capital investment and operational space requirements would be too great given the size of the operation. Instead, all processing would be done off-site. Due to this, the cost of transporting ore to a nearby processing plant would need to be included.

Once the yearly cash flow had been calculated by subtracting the yearly expenses from the yearly income, the present value could be calculated (Equation 12). The NPV was then calculated by summing each year's present value (Equation 13). Following the estimation of NPV, the data calculated was used in the comparison and analysis of both mining methods.

$$Present\ Value = \frac{CF}{(1+i)^n} \quad \text{Equation 12}$$

where:

C_F - Cash Flow (\$)

L - Hole length (m)

D - Drilling rate (holes/h).

$$NPV = \sum PV \quad \text{Equation 13}$$

where:

PV – Present value (\$).

3. RESULTS

3.1 Mine Design and Characteristics

The design for future mine workings (Fig. 2) was provided by the One Tree Hill mine and each resource and development section were divided into physicals (Fig. 3).

The size characteristics for all development physicals were established. For calculation of tonnage, the density of waste rock mined in development stages was assumed as 2.7g/cm³. The physical characteristics of the Mystery, Buck and Butch stopes determined, taking on account that the proportion of the physicals that would actually be mined using the two methods are different to the true physical characteristics. The strike lengths of the methods

are also different, as shrinkage stoping will require a 1 m thick pillar to be left between each stope section. The Mystery, Buck and Butch mining method physicals are shown in Tables 1 – 3.

Figure 2 - One Tree Hill mine design. Note: names and terms that are referred to differently in the rest of this paper; including: the Mystery level 3 is referred to as the Mystery Central stope; Declines that sit under stopes are called drifts; the Butch/Swede is referred to as the Butch stope.

Table 1 - Mystery mining method physicals.

Production Physical	Airleg Stoping				Thermal Stoping			
	Mining Width (m)	Tonnes	Block Grade (g/t)	Gold Ounces	Mining Width (m)	Tonnes	Block Grade (g/t)	Gold Ounces
Central Stope	0.9	1574.6	13.3	675.0	0.5	850.5	20	547
Mystery 1	0.9	758.2	8.3	203.1	0.5	421.9	15	203
Mystery 2	0.9	933.1	8.3	250.0	0.5	523.1	15	252
Mystery 3	0.9	933.1	8.3	250.0	0.5	523.1	15	252
Mystery 4	0.9	1458.0	8.3	390.6	0.5	826.9	15	399
Mystery 5	0.9	1691.3	8.3	453.1	0.5	961.9	15	464
Mystery 6	0.9	1982.9	8.3	531.3	0.5	1130.6	15	545
Mystery 7	0.9	1749.6	8.3	468.8	0.5	995.6	15	480
Mystery 8	0.9	1749.6	8.3	468.8	0.5	955.8	15	461
Total		11255.8		3015.7		6338.9		3057

Figure 3 - Physicals of each resource and development section: a – Mystery stope and drift physicals; b - Butch rise and drift physicals; c – Main decline, Butch stope and drift physicals; d – Mystery stockpile, Mystery Central stope and Buck rise physicals.

Table 2 - Buck mining method physicals.

Production Physical	Airleg Stopping			Gold Ounces	Thermal Stopping			Gold Ounces
	Mining Width (m)	Tonnes	Block Grade (g/t)		Mining Width (m)	Tonnes	Block Grade (g/t)	
Buck Stope 1	0.9	1244.2	11.7	466.7	0.8	1156.7	13.13	488
Buck Stope 2	0.9	1244.2	11.7	466.7	0.8	1156.7	13.13	488
Buck Stope 3	0.9	1205.3	11.7	452.1	0.8	1120.0	13.13	473
Buck Stope 4	0.9	1205.3	11.7	452.1	0.8	1120.0	13.13	473
Buck Stope 5	0.9	1166.4	11.7	437.5	0.8	1083.2	13.13	457
Buck Stope 6	0.9	1127.5	11.7	422.9	0.8	1046.5	13.13	442
Buck Stope 7	0.9	1127.5	11.7	422.9	0.8	1046.5	13.13	442
Buck Stope 8	0.9	1127.5	11.7	422.9	0.8	1046.5	13.13	442
Buck Stope 9	0.9	1127.5	11.7	422.9	0.8	1046.5	13.13	442
Buck Stope 10	0.9	1156.7	11.7	433.9	0.8	1009.8	13.13	426
	Total	11732.0		4400.6		10832.4		4571

Table 3 - Butch mining method physicals.

Production Physical	Airleg Stopping			Gold Ounces	Thermal Stopping			Gold Ounces
	Mining Width (m)	Tonnes	Block Grade (g/t)		Mining Width (m)	Tonnes	Block Grade (g/t)	
Butch Stope 1	0.9	1458.0	5.6	260.4	0.5	824.9	10	265
Butch Stope 2	0.9	1701.0	5.6	303.8	0.5	965.3	10	310
Butch Stope 3	0.9	1944.0	5.6	347.2	0.5	1105.7	10	355
Butch Stope 4	0.9	2247.8	5.6	401.5	0.5	1281.2	10	412
Butch Stope 5	0.9	2369.3	5.6	423.2	0.5	1351.4	10	434
Butch Stope 6	0.9	2084.9	5.6	372.4	0.5	1140.8	10	367
Total		11804.9		2108.5		6669.0		2144

Table 4 - Planned Dilution.

Stope Name	Airleg Stopping	Thermal Stopping
Mystery Central	0.50	0.00
Mystery	0.80	0.00
Buck	0.29	0.14
Butch	0.80	0.00

Table 4 shows the planned dilution of each stope for the two selected mining methods. It can be seen that the shrinkage stopping method contains large quantities of planned dilution. The Mystery and Butch stopes both have a planned dilution of 0.8, which equates to approximately 44% of the total tonnage being made up of waste rock. In comparison, only the Buck stope contains planned dilution for the thermal stopping method. The dilution is also much lower in comparison with 0.29 for shrinkage and 0.14 for thermal (22% and 13% waste content respectively). Figure 4 displays a comparison of tonnage for the shrinkage stopping method, the thermal stopping method and the actual stope tonnage. Figure 5 illustrates the ounces produced from both methods against the ounces available from the stope.

Figure 4 - Stope tonnage comparison.

Figure 5 - Stope gold ounces comparison.

It is evident that shrinkage stoping consistently produces a greater tonnage, due to the greater mining width that must be used for the method. For thermal stoping, the tonnage produced is mostly less than the actual stope because of the pillars required above and below the stope. The exception to this is the Buck stope, which required a small amount of over blasting in order to recover the full stope width. The tonnes mined in the production stages can greatly increase expenditure due to the greater quantity of ore needing to be transported out of the mine and to processing facilities. In the Mystery, Mystery Central and Butch stopes the shrinkage stoping method produces over 80% more rock when compared to the thermal stoping method.

For the Mystery, Buck and Butch stopes, thermal stoping produces more ounces of gold than shrinkage stoping; both of which contain less than the full quantity available (Fig. 5). While both methods require pillars above and below the stope, the shrinkage stoping method requires a 1 m thick pillar between each stope section; therefore, reducing the potential quantity of ore produced. This is not the case for the Mystery Central which does not require a pillar as there is only one stope section. Figure 5 reflects this, as the ounces produced for the Mystery Central is greater using shrinkage stoping compared to thermal stoping.

3.1 Development Duration

For the calculation of development cycle time were used. Table 5 shows the time spent drilling holes for each cycle. Due to varying geology, drill rates for the Buck and the Decline were higher than the drill rates for the Butch and the Mystery. Bogging time was dependant on the factors shown in Table 6, but ultimately varied for each physical depending on the haulage distance. It should be noted that a swell factor of 1.3 would be assumed for all blasted rock.

The total time taken for charging, ventilation, support and drilling per section were calculated, as well as the bogging time per section and the total time taken for the development of each section.

Table 5 - Drilling time per cycle.

Hole type	Number of Holes	Drill rate Buck & Decline (holes/h)	Drill rate Butch & Mystery (holes/h)	Drill Time Buck & Decline (h)	Drill Time Butch & Mystery (h)
Blast	34	25	15	1.36	2.26
Relief	1	12.5	7.5	0.08	0.13
			Total	1.44	2.4

Table 6 - Bogging time per cycle.

Cut Depth (m)	Volume of Cut (m ³)	Blasted Volume	Bucket size	Number of rounds	Haulage Speed (m/h)	Loading time (h)
2.2	11.88	15.444	2	7.722	10,000	0.2574

3.2 Mining Method Schedules

For the shrinkage stoping mining method, equations 8, 9 and 10 were used to produce the durations for airleg drilling, blasting and the bogging durations for each stope section.

For the thermal fragmentation mining method, there are 4 stages of production that durations were calculated for: the durations for drilling of the thermal holes and blast holes for each section, the durations for thermal fragmentation production, the blasting and bogging duration for each stope section.

As with the development design characteristics, the development schedule has little effect on the comparison of the two methods as both shrinkage and thermal stoping share the same development schedule. However, while the haulage distance greatly effects the bogging time, it has no great effect on the overall cycle time. This is because the majority of the cycle time consists of drilling, support installation and ventilation. The bogging time is expected to play a larger role in the production stages. Figure 6 illustrates the durations of each task within the development section. As bogging times vary, the average bogging time for each development section was used.

For the production schedules, the difference between methods makes duration comparison difficult. Using the shrinkage stoping method, the total time spent in the drill and blast phase (which also includes 14 tonnes of daily bogging) for all stopes is 520 days. The thermal stoping method takes 223 days for thermal hole drilling, 223 days for blast hole drilling, 143 days for thermal fragmentation and 10 days for blasting. In total, this equates to 600 days for the drill and blast phase for thermal stoping as compared to the 520 days for shrinkage stoping. Nevertheless, the thermal method allows for two drill rigs and the thermal fragmentation to operate within the same drift concurrently.

Bogging for the shrinkage stoping method has a total duration for all stopes of 440 days. This includes the daily bogging that occurs during the drill and blast phase, so a more representative number would be the bogging that occurs after drilling and blasting is complete; a duration of 96 days. For the thermal stoping method, the total time for bogging is 288 days. The bogging time for the thermal stoping method is shared with time for installation of services. Due to the complexity and differences between the two methods, it is easier to compare them by looking at each schedule as a whole.

Figure 6 - Development cycle time.

The simplified Gantt charts are presented together in the same figure to allow for easier comparison (Fig. 7). From this graph, it can be seen that both methods are relatively similar in their schedule approaches and that the development stages have the same durations. It is clear that the main difference between the two methods is the production duration. With the exception of the Mystery Central, the mining of stopes using the shrinkage stope method appears to be substantially longer than that of the thermal stope method. This results in a finishing date difference of 232 days.

Figure 7- Simplified schedule comparison.

3.3 Resource Selection

Selection of equipment and estimates around the equipment rental was through recommendation by the One Tree Hill mine. Where possible, equipment that was already owned by the mine would be utilised in order to reduce fixed costs.

For development sections, it was decided that a Long Tom Drill Jumbo would be used. Long Toms have two booms with which to carry airleg drills, pneumatically powered and small enough to accommodate the width and height of the mine passageways. Benefits of the Long Tom include a reduction in labour fatigue and increased productivity (Parts HeadQuarters Inc., 2013). The approximate rental cost of a Long Tom was taken as \$1,000

per month.

A key piece of equipment for the development and the production stages of both mining methods is the Bogger. The chosen Bogger to be used in the mine is a Sandvik LH204, which is specifically designed for applications such as narrow vein mining, with an average fuel consumption of 14.5 l/h (Sandvik, n.d.). The other important gear employed for the shrinkage stoping method is the airleg. The airleg is also used in development sections for installing supports and as the primary drilling method for raises.

To undertake the drilling in the production stages of the thermal stoping method, two different drill rigs were selected. For the blast holes needing to be 64 mm in diameter, a long hole drill rig was planned for use. As the thermal holes are required to be 152 mm in diameter, an In-The-Hole (ITH) drill rig would be needed for increased productivity.

The most important machinery for the thermal stoping method is the thermal fragmentation device, the Nippon Dragon. An available smaller version of the Dragon is suited to the narrow workspace of the mine, the Mini Dragon. It is required that the Mini Dragon has a supply of compressed air, a tank of diesel fuel and a vacuum tube connected to the surface to remove fumes. It is expected that the fuel consumption of the Mini Dragon would be approximately 43.5 litres of fuel per hour of operation.

Other equipment needed for daily operation of the mine include the vacuum tube, electricity generator and air compressor. The One Tree Hill mine is naturally ventilated but requires some form of ventilation for a reliable working environment. For this reason, the vacuum tube would be used in development and production stages for both mining methods. The generator and compressor are utilized for maintaining a constant supply of electricity and compressed air respectively to other equipment.

3.4 Resource Allocation

For each task in the schedule, the required resources had variable and fixed costs associated (Table 7 and 8).

Table 7 - Shrinkage stoping resource costs.

Resource	Variable Costs (\$/t)	Total Variable Costs (\$)	Fixed Costs (\$)
Long Tom		66,177	11,953
Bogger		40,067	261,256
Airleg		53,981	-
Vacuum		139	179,301
Compressor		428,878	52,141
Generator		25,495	239,069
Miners	-	-	1,594,233
Explosives		45,008	-
Support		82,863	-

When comparing the two mining methods in terms of equipment and resources, it is important to look at the quantity of requirements as well as overall costs. From Table 7 and Table 8 it can be seen that the number of resources used in the thermal stoping method is larger than for shrinkage. Where this creates a potential problem is the amount of room for equipment to be stored and operated. As the site is located in a State park, space for storage outside the mine is minimal.

Table 8 - Thermal stoping resource costs.

Resource	Variable Costs (\$/t)	Total Variable Costs	Fixed Costs
Long Tom	6.85	66,177	9,599
Bogger	0.84	7,177	183,424
Airleg	0.31	3,106	-
ITH Drill Rig	33.12	268,164	164,426
Long Hole Drill Rig	13.79	111,640	32,889
Mini Dragon	16.45	133,199	16,388
Vacuum	0.01	186	268,137
Compressor 1	13.12	342,352	33,243
Compressor 2	19.88	160,929	32,885
Generator	1.26	43,286	357,516
Miners	-	-	1,289,604
Explosives	1.23	32,651	-
Support	8.58	82,863	-

The comparison of costs between the two mining methods is shown in Figure 8. Consumables consist of explosives and supports which will be used. These have only variable costs associated with them and make up only a small proportion of the total resource costs. The cost associated with personal and wages make up the greatest proportion of costs for both mining methods. While the thermal stoping method will result in more employees, shrinkage stoping has a greater personnel cost due to the longer duration of operation.

The cost of drilling is greater for thermal stoping than that of shrinkage stoping due to the increased costs associated with the two long-hole drills and the thermal fragmentation device. These costs are deemed necessary to facilitate the higher productivity associated with thermal fragmentation. Services consist of the compressor, generator and vacuum tube needed for daily operation. For thermal stoping this makes up a large proportion of the total costs as rental and operation costs are relatively high. The requirements for services in the shrinkage stoping method are less than that of the thermal stoping method, however they still make up a large portion of the overall costs.

Following the methodology, the NPV for both the shrinkage stoping method and the thermal fragmentation was calculated (Table 9 and 10). As the project has no forecast start date, it is important to select a gold price that can accommodate for future changes. In order to account for both highs and lows within the commodity price cycle, the average gold price over 10 years can be selected (Guarnera & Martin, 2011). An approximate average of \$1,400/oz was selected as Gold Price (2018) records a 10-year high and low of 1,821 and 906 AUD/oz respectively.

Figure 8 - Resource cost comparison Net Present Value.

For haulage costs, there are two viable processing plants that ore could be hauled to. These are located in Ballarat and Bendigo which are approximately the same distance from One Tree Hill mine. An approximate quote of \$32.5/t was supplied by a local haulage company, given that haulage would be constant over a 2-3 year period. For processing costs, the assumption was made that blasted rock would cost \$25/t to be processed by a third-party plant. Due to the much smaller particle size that thermal fragmentation produces, it is expected that the cost of processing would be reduced by 20%.

Table 9 - Shrinkage stoping NPV.

Year	0	1	2	3	Total
Development Tonnage		9902	0	0	9902
Production Tonnage		13380	18905	4082	36367
Income					
Ounces		2784	6323	1094	10200
Income @\$1,400/oz		3,896,975	8,851,503	1,531,276	\$14,279,754
Costs					
Wages		935,697	900,865	123,351	\$1,959,913
Fixed resource Costs		555,406	166,518	21,798	\$743,723
Variable resource Costs		404,286	268,152	50,110	\$722,547
Haulage Costs (\$/t)		756,647	614,426	132,678	\$1,503,751
Processing Costs (\$/t)		334,490	472,635	102,060	\$909,185
Cash Flow	0	910,449	6,428,908	1,101,279	
Discount factor	1	0.909	0.826	0.751	
Present Value	0	827,681	5,313,147	827,407	
Net Cash Flow					\$8,440,636
Net Present Value					\$6,968,235

Table 10 -Thermal stoping NPV.

Year	0	1	2	3	Total
Development Tonnage		9902	0	0	9902
Production Tonnage		8277	16414	0	24691
Income					
Ounces		3056	7263	0	10,319
Income @\$1,400/oz		4,278,555	10,168,150	0	\$14,446,706
Costs					
Wages		1,034,571	577,434	0	\$1,612,005
Fixed Resource Costs		744,132	354,379	0	\$1,098,511
Variable Resource Costs		690,707	581,029	0	\$1,271,736
Haulage Costs (\$/t)		590,799	533,464	0	\$1,124,263
Processing Costs (\$/t)		165,531	328,286	0	\$493,817
Cash Flow	0	1,052,816	7,793,558	0	
Discount factor	1	0.909	0.826	0.751	
Present Value	0	967,106	6,440,957	0	
Net Cash Flow					\$8,846,374
Net Present Value					\$7,398,063

3.5 Net Present Value

Typically for Net Present Value calculations, capital expenses and salvageable income would be included. However, as these works are a continuation of an already existing mine, all the equipment needed will be hired, therefore making these negligible. Any capital that would be needed would most likely be equal for both the shrinkage stoping and thermal stoping methods.

The income of each mining method is the result of the ounces produced and the gold price. However due to the assumption that the gold price remains constant with each year of operation, the income then becomes fully dependant on the ounces produced. The total income for the thermal stoping method is greater than that of the shrinkage stoping method, with thermal stoping producing an increase in income of \$166,952 (1.169% of the shrinkage stoping income).

As well as the costs associated with resources, haulage and processing costs contribute a large proportion to the total costs for each method. Due to the greater tonnage of ore needing to be transported and processed for the shrinkage stoping method, thermal stoping is approximately 25% and 45% more cost efficient for haulage and processing respectfully. For greater haulage distances, the cost difference between thermal stoping and shrinkage stoping would increase. Processing is expected to cost less for thermal stoping due to the decreased requirement for comminution. In total, the costs are lower for thermal stoping than those for shrinkage stoping. The difference between the two methods is \$238,787, which is 4.26% of the total thermal method costs.

The differences in income and cost discussed above, give a difference between the Net Cash Flow of each methods, of \$405739. Despite the fact that NPV is less than Net Cash Flow for both mining methods, the positive beneficial difference between the thermal and shrinkage methods only increases. This is because the thermal stoping method extracts the available ore in a much shorter time period than the shrinkage stoping method. Using the NPV model, the later that potential income can be capitalised on, the less the overall

profit. Overall, the thermal fragmentation method is shown to increase NPV by \$429,828, or 6.17%, when compared to the shrinkage stoping method.

Conclusions

The research data presented suggests that thermal fragmentation would be a viable alternative for narrow vein mining. In comparison to shrinkage stoping, thermal fragmentation resulted in lower total costs and an increased income. Resource costs for thermal fragmentation were higher, however these were outweighed by the higher costs associated with shrinkage stoping of wages, haulage and processing. Thermal fragmentation also resulted in a greater recovery of ore, which leads to a higher income. The use of Net Present Value calculations also confirmed the viability of the thermal stoping method when compared to shrinkage stoping. Overall the differences in NPV were not large but were enough to be confident of thermal fragmentations validity.

A reduction in overall mining duration was possible with the thermal stoping method. While both methods shared the same duration for development sections, thermal fragmentation achieved the extraction of production sections significantly quicker. The extra duration associated with the shrinkage stoping method resulted in extra wage and hire costs, as well as application of a lower discount factor as the mining method reached its third year. While not explored in this paper, it is predicted that processing time and costs would be greatly reduced using thermal fragmentation due to the smaller particle sizes compared to traditional blasting. A reduction in size removes much of the need for comminution.

One possible benefit that shrinkage stoping had over thermal fragmentation, is the lesser requirement of equipment. Additional equipment not only increases the resource costs, it has the potential to put stress on the mine sites already limited space. However, this is seen as necessary in order to remove the low productivity and highly labour-intensive characteristics associated with shrinkage stoping.

Recommendations for further work

As thermal fragmentation is a new technology to Australia, there is much research to be conducted before the method could be trusted by mining companies as a viable mining method option. Throughout this study, potential research avenues that were outside the paper's scope had become apparent and would be beneficial to the future of thermal fragmentation.

The comparison used in this research paper was limited in that it only related to one location and one mining method. It is important that the mining method be tested in more case studies in order to build confidence surrounding the technology. A large part of this is the comparison to another mining method other than shrinkage stoping. The most logical method to compare to would be longhole stoping, as this is the most popular method for narrow vein mining in Australia.

Throughout this paper the assumption was made that the thermal fragmentation method would reduce costs and time for ore processing. This is because a much smaller particle size can be achieved with thermal fragmentation when compared to blasting, so reducing the amount of comminution required. Data and research needs to be collected around these assumptions so that the extent of the savings can be quantified. This would be highly important in gaining greater accuracy for future mining method comparisons.

Two aspects of viability for thermal fragmentation that were not explored in this study were safety and environment. In order to gain confidence around thermal fragmentation, it is

important to determine what benefits, if any, are likely to be gained in the areas of safety and environment. It is predicted that the lower requirement for haulage, and subsequently the reduction in trucks would be beneficial for the environment. However, the increase need for other equipment for the thermal fragmentation method, such as compressors, may offset any potential benefit.

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