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Simulation of Materials Handling Systems in the Mines: Two Case Studies

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In the February 1998 issue of SIMULATION, Alexander Lebedev discussed the applicability of continuous and discrete simulation modelling techniques on bulk conveying systems. In this article, he and Phillip Staples continue on the subject by presenting two case studies simulating real *mining operations. The first deals with development and verification of a process control algorithm for underground bunkers on a simulation model. The objective of the other is sizing of surge and storage facilities.*

Keywords: Mining, conveyor, bunker, stockyard, capacity

1. Introduction

In referenced papers that describe case studies of application of simulation to modelling bulk conveying systems—primarily in the mining industry—the authors used discrete simulation software tools to quantify the performance of a mining operation at a given set of resources [1, 2]. Nowhere in the published literature [3, 4,5] were examples found of the use of continuous simulation modelling approaches for the purpose of control as well as for sizing of materials handling equipment.

2. Case Study: Controlling Underground Bunkers

2.1. Introduction to and Objective of the Simulation Study

In this study, an underground mine produces material from a number of mining sections, each provided with a feeder-breaker. In these mining sections, the material is loaded onto shuttle cars that transport and tip the material into feeder-breakers. From the feeder breakers, the material is sent to the conveyors that feed the bunkers. There are 12 feeding conveyors in total; ten feed the bunkers and the other two feed the trunk conveyors directly (A14, A18).

The capacity of feeder-breakers is 600 tons per hour (t/h). The capacity of the shuttle cars varies, and it can be either 14.5 or 9.0 tons. The smaller shuttle cars operate in mining sections that feed conveyor A18

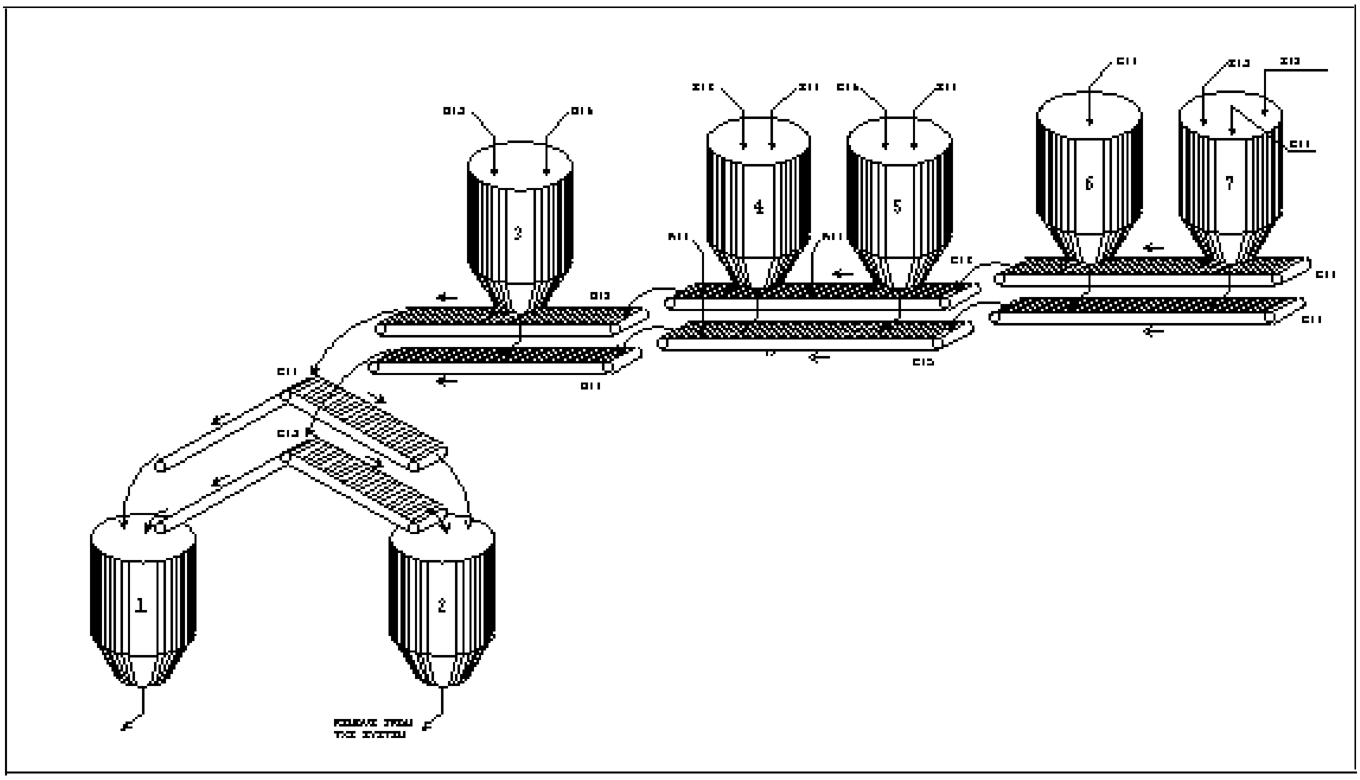

Figure 1. Schematic flowchart of the conveyor system

only. All the rest of the mining sections make use of 14.5-ton cars. The mine logged the number of tips into each feeder-breaker, and the numbers were then converted into average feed rates of the relevant feeding conveyors. However, for conveyors A14 and A18, instead of flow rates, discrete feed of parcels equivalent to the load of the shuttle cars was applied with the actual inter-arrival time.

There are in total seven bunkers in the system, each with a 1,000-ton capacity. Five of them—numbered 3 to 7 and later referred to as "surge bunkers'' - receive the material from the feeding conveyors. The other two bunkers—numbers 1 and 2, which will be called "main bunkers'' - are fed by the trunk conveyors (see Figure 1).

Each of the bunkers is provided with four vibrating feeders, two for each trunk conveyor. The vibrating feeders each have a 600 t/h capacity. The rate of discharge can be controlled in steps of 200 t/h; i.e., the discharge rate from a vibrating feeder onto a trunk conveyor belt can be 0, 200, 400 or 600 t/h. Each of the four vibrating feeders can be controlled individually.

The level of material in the bunkers is monitored.

There are two identical trunk conveyor lines in use, each consisting of four conveyors rated at 2,500 t/h. Material is transferred from one conveyor belt to the other to make up a continuous line, as shown in Figure 1. Conveyors C13 and C14 can feed main bunkers on an alternative basis; however, conveyor C13 feeds primarily bunker 1, and conveyor C14 feeds bunker 2.

The current flow rate of conveyor belts C08/07, C06/05 and D02/01 is monitored by belt weighers.

The problems at this mine were caused by a lack of control over mining sections operation and over discharge rates from the surge bunkers on the trunk belts. This resulted in overflow of the bunkers and the trunk belts. Every time a trunk belt was over-loaded, a stoppage of about six hours was required for cleaning. If a surge bunker overflowed, the corresponding mining sections had to be shut down. The loss of production due to breakdowns of the equipment and overflow of conveyor belts was 40%.

The objective of the simulation project was to develop and verify a process control algorithm with a simulation model to:

(1) Avoid overflow of the surge bunkers, and

(2) Eliminate excessive feed of the trunk conveyor belts to minimise the probability of blockages.

2.2. Simulation Approach and Assumptions

In a discussion with the mine, it was decided that in view of the inertia of the vibrating feeders, control actions to reduce or increase the discharge rate

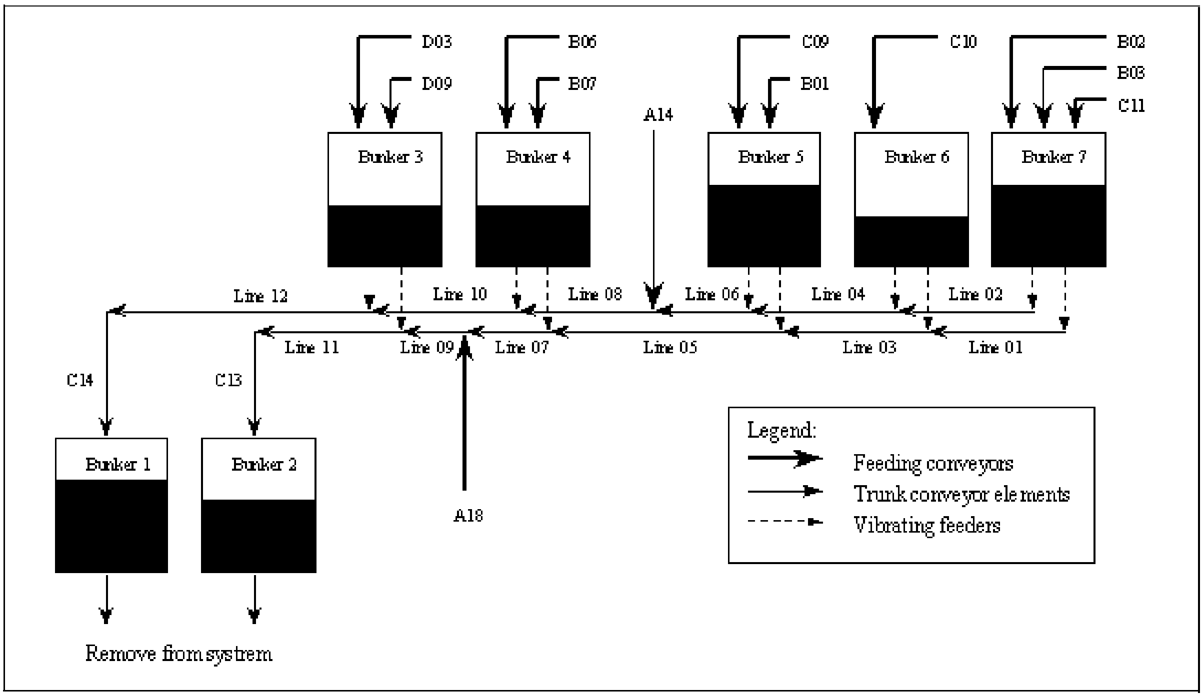

Figure 2. Block diagram of the simulation model

would begin within a period of 10 minutes. The current discharge rate of a vibrating feeder was sampled from a random distribution, and no harmonic profile could be assigned to it, However, the control step excitation had a longer period than the transport delay of a trunk belt; therefore, a continuous simulation approach was applied in accordance with the conclusion of the paper, "Simulation Modelling of Bulk Conveying Systems" [6].

Due to the fact that in a simulation model a conveyor may only be loaded at the rear, the actual conveyor belts had to be divided into segments between the feed points, consecutively linked together (see Figure 2).

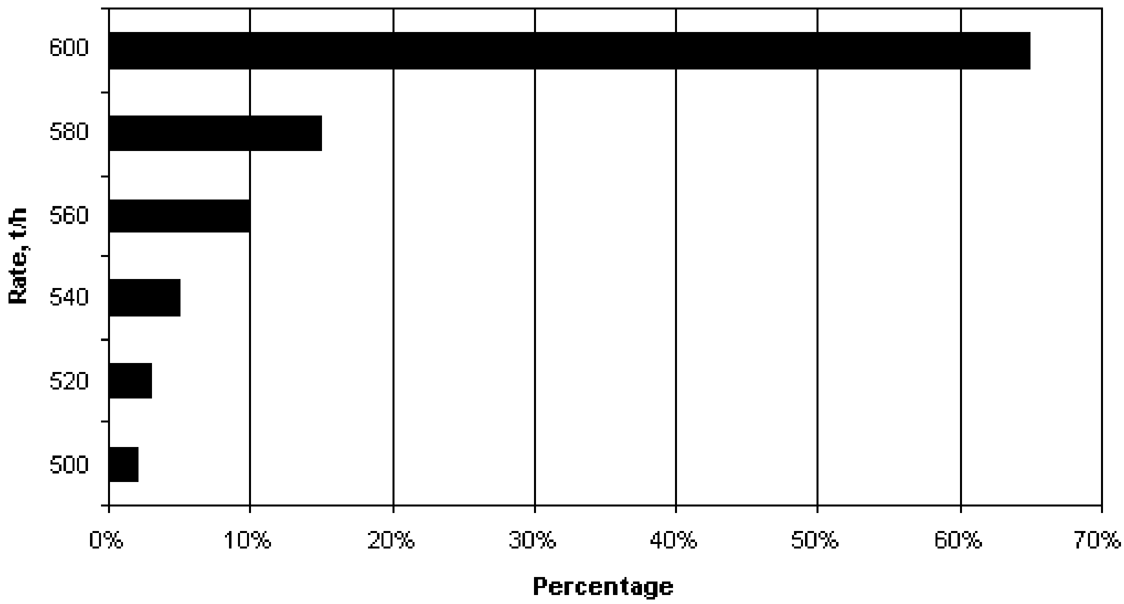

Figure 3. Discharge rate distribution

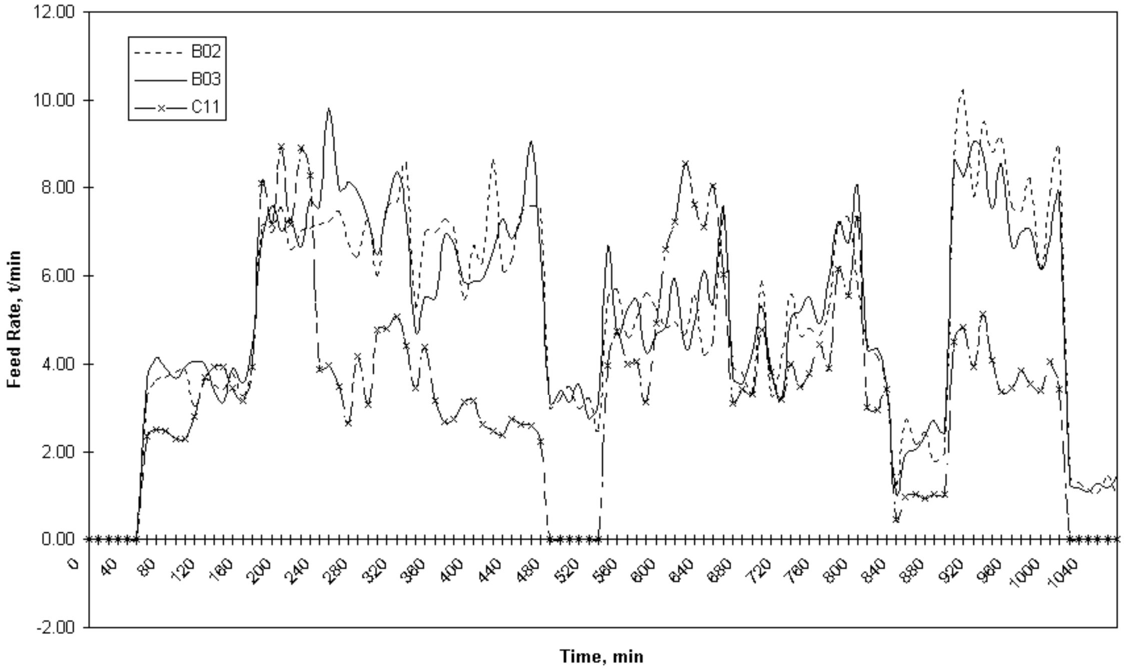

Figure 4. Feed to bunker 7

The actual discharge rate from a vibrating feeder was simulated as a distribution to take into account the effect of dispersion of the particle size. The profile of the distribution, which is shown in Figure 3, was based primarily on the operation experience of the mine personnel.

Four vibrating feeders physically installed beneath each bunker were simulated as two outputs, each responsible for equivalent discharge of material from the bunker onto one of the two trunk conveyor belts.

The maximum theoretical feed rate of a trunk conveyor line was assumed to be 50 tons per minute (t/m), or 3,000 t/h, to allow for detection of the overload of a conveyor belt. It was assumed that a conveyor would be blocked any time the current flow rate exceeded some pre-set value defined by the user. The average downtime period to clean the belt and remove spillage was set at six hours.

In addition to overloads of the conveyor trunk belts, mechanical breakdowns were also introduced

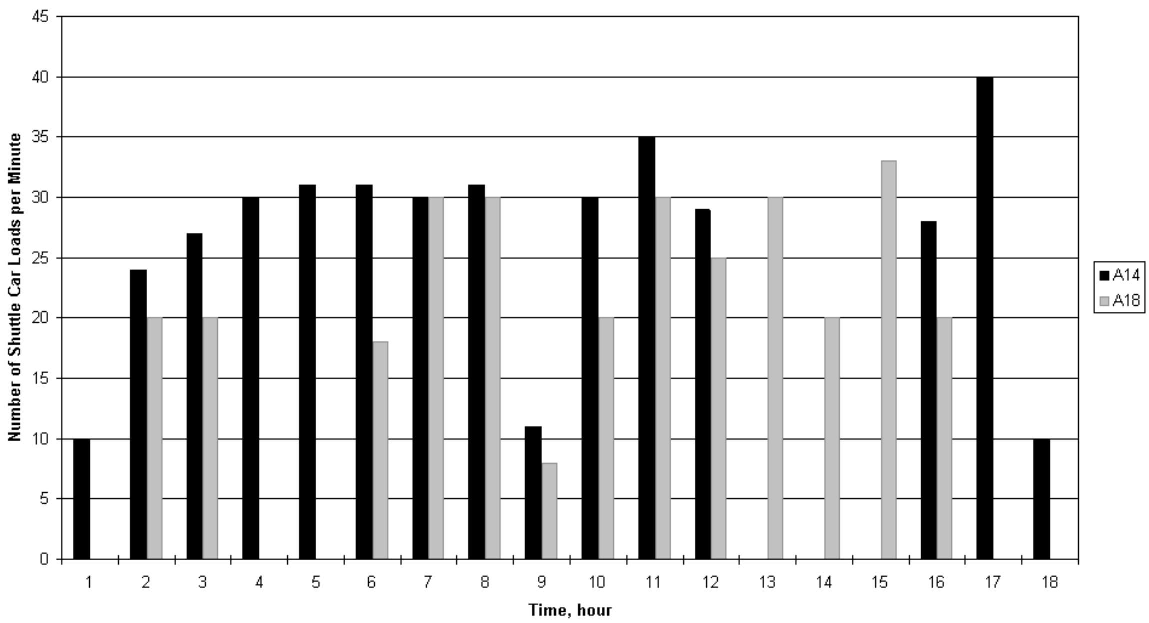

Figure 5. Direct feed to the trunk conveyors

into the model. The mean time between failures (MTBF) of every individual trunk conveyor was sampled from a negative exponential distribution with a mean of 6,220 minutes. The mean time to repair (MTTR) was sampled from a Gamma distribution with a mean of 260 and a variation of 130 minutes,

It was assumed that the feeder-breakers could be controlled by means of red and green "tipping lights." A shuttle car was allowed to tip material into the feeder-breaker only if the green light was on. If the red light was lit, no material could be discharged into the feeder-breaker.

A simple yet robust algorithm was developed to control the discharge of material from the surge bunkers onto the trunk conveyor belts to avoid overload, as well as to control the tipping process of the shuttle cars into the feeder-breakers to prevent the bunkers from overflowing. The objective of the algorithm was to maintain the total discharge rate of the material from the surge bunkers on each of the two trunk belts (not exceeding 1,900 t/h) to accommodate the parcels arriving from conveyors A14 and A18 at a 600-t/h rate (those two conveyors were not managed) and to delay the tipping of shuttle cars in the mining sections if there was a risk of the surge bunkers overflowing. Some minimum residual volume of material in the surge bunkers was also set in the algorithm.

This algorithm was also designed to account for any types of breakdowns of the trunk conveyors; however, it cannot be described here due to an agreement with the user.

2.3. Discussion of Simulation Results

The model was run for a period of a full calendar year in accordance with the actual mine's work schedule (4,968 working hours).

A total of 11.3 million tons (Mt) of material was produced by the mine, which made up ±60% of the theoretical capacity. The effect of the interruption in the shuttle-car tipping by the control algorithm was negligible, however: only ±0.5 Mt of material was rejected by the system in the peak periods. That meant, first of all, insufficient productivity of the mining sections as well as irregular profiles of the shuttle car arrivals.

The example of the feed into bunker 7 is presented in Figure 4.

Direct feed in batches from conveyors A14 and A18 to the trunk belts during the first two shifts is shown in Figure 5. It can easily be seen that all the feed conveyors deliver maximum tonnage in the middle of a shift, starting with and dropping to zero at the beginning and at the end of the day. In Figure 4 an effect of the control algorithm can be noted with sharp reduction of the feed rates of all the conveyors at approximately the middle of the second shift.

High utilisation of all the conveyors feeding the bunkers was recorded, ranging from 77% to 900%, though still at lower-than-designed flow rates. Directly feeding conveyors due to the limited input from the mining sections and short transport delay caused idling for 74% to 79% of active time.

An example of the plots of the current level of material in the surge bunkers is given in Figure 6.

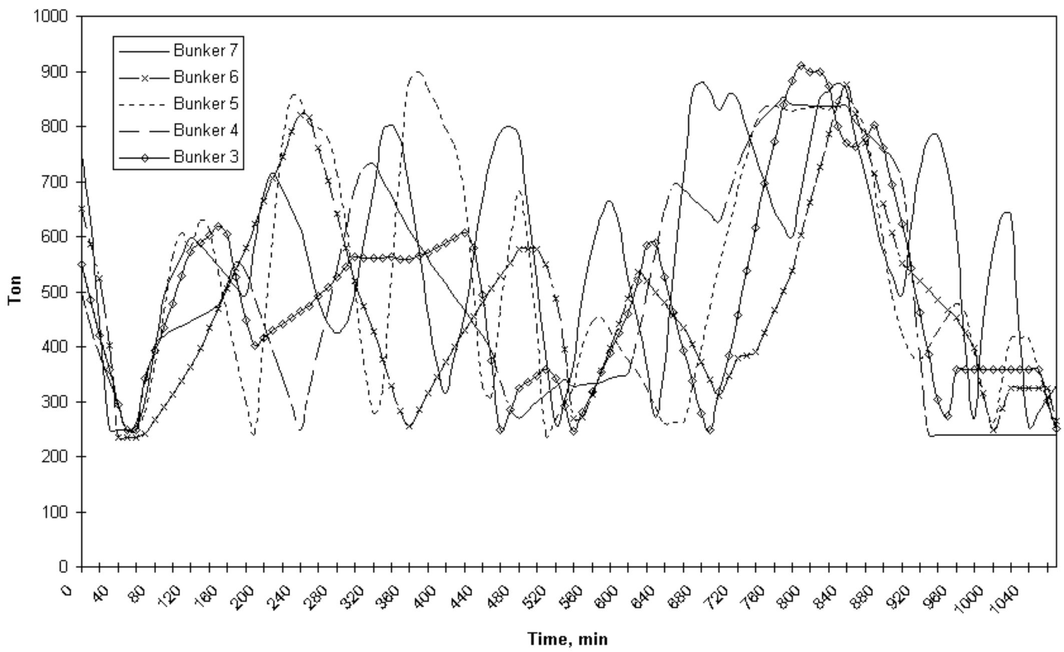

Figure 6. Current level of coal in bunkers

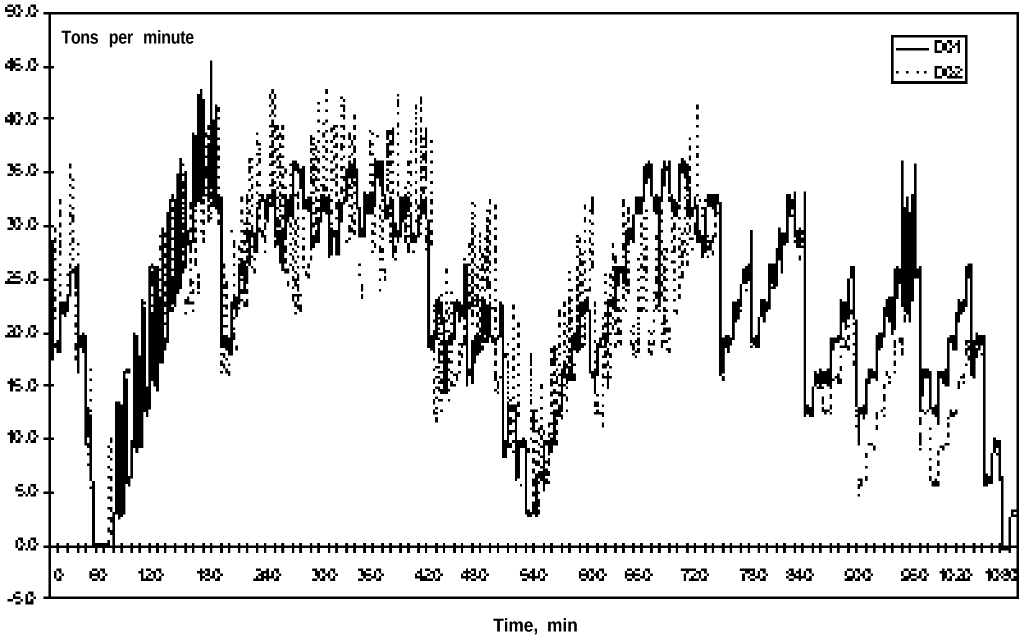

Figure 7. Readings of belt weighers

The most important observation from the plots is that the level of material in all the surge bunkers was below 900 tons (t). Full capacity is 1,000 t. This was confirmed by the simulation statistics gathered during the whole simulated period, shown in Table 1.

The average amount of material in surge bunkers 3 through 7 ranged from 500 to 585 tons. What is even more important is that these bunkers were never empty or full.

Current readings of the belt weighers installed in the head of conveyors D01 and D02 for one working day are plotted in Figure 7.

Virtually no peaks exceeding 42 t/m (2,500 t/h) have been noted during the whole simulation run, thus ensuring smooth operation of the trunk belts.

The utilisation of the trunk conveyors is presented in Figure 8. Bearing in mind that real trunk conveyor belts were divided into segments between the feed points, the codes on the bar chart correspond to the last segments of the relevant trunk belts (refer to Figures 1 and 2).

Plots in Figure 7 represent the behaviour of the trunk conveyors with no breakdown within a two-shift-per-day operation. The effect of breakdowns of conveyor C08 with a three-hour duration starting at 400 minutes from the very beginning, and of conveyor D01 with a two-hour duration starting at 700 minutes, is shown in Figure 9.

It is important to note that in the event of breakdown of the first conveyor in line (C08), the effect on the delivery rate of the whole line was negligible,

	Volume, tons					Time, %				
Name	In	Out	Currently In	Avg In	Avg Time In, Min	Flowing	Empty	Full	Partly Full	Off-Shift
Bunker 1	5,455,890	5,455,680	200	194.4	10.6	100	20.8	0.0	79.2	43.3
Bunker 2	5,501,750	5,501,550	192	197.4	10.7	100	20.0	0.0	80.1	43.3
Bunker 3	1,723,940	1,723,650	292	515.4	89.1	100	0.0	0.0	100.0	43.3
Bunker 4	1,189,370	1,189,050	315	499.7	125.2	100	0.0	0.0	100.0	43.3
Bunker 5	2,965,820	2,965,570	246	551.3	55.4	100	0.0	0.0	100.0	43.3
Bunker 6	969,778	969,546	233	518.1	159.3	100	0.0	0.0	100.0	43.3
Bunker 7	3,580,690	3,580,440	245	584.5	48.7	100	0.0	0.0	100.0	43.3

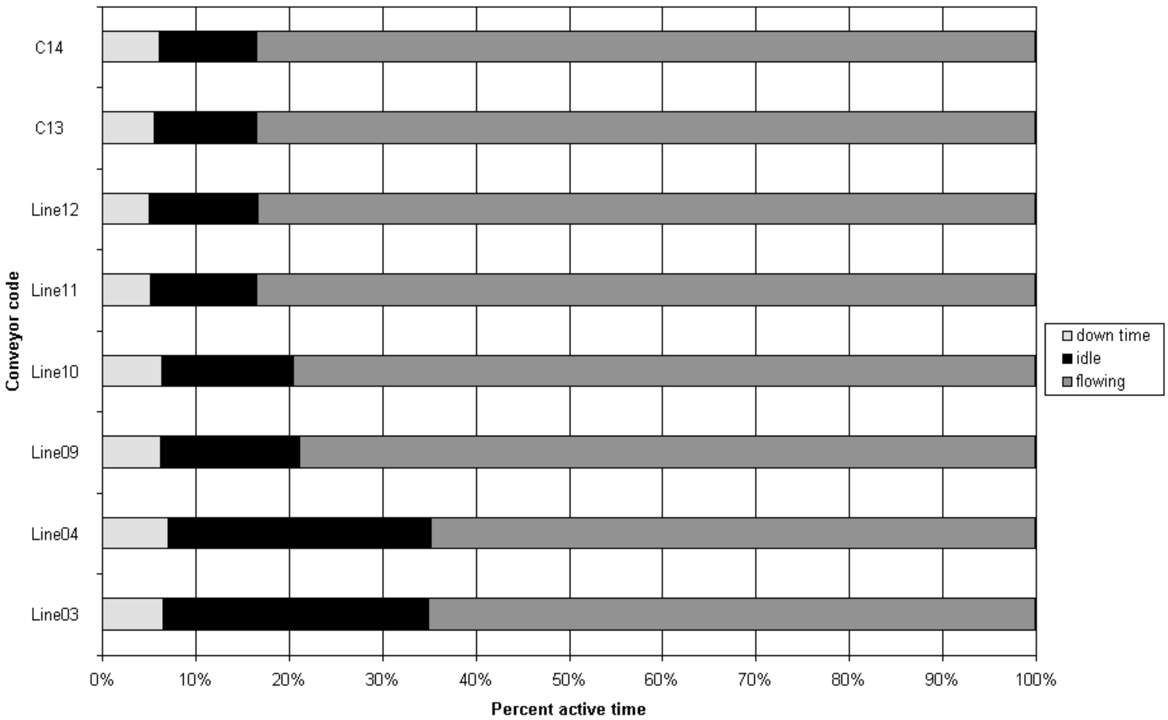

Figure 8. Utilisation of trunk conveyors

specifically taking into account the natural decline in production at the end of the day shift when all the bunkers were being emptied to the lowest permissible level. Then the control algorithm maintained the delivery of the trunk conveyor line at a rate very close to the target of ±31 t/m.

The effect of the breakdown of conveyor D01, the last one in the line, on delivery performance was much heavier as the entire line became blocked. However, once the healthy status of the conveyor was restored, the delivery was step-by-step brought to the target level.

The comparative statistics on the performance of the system in two scenarios is summarised in Table 2.

Insignificant reduction in the throughput of the system of some ±4% was due to inadequate performance of the mining sections delivering some ±40% less than the design capacity of system. In other words, while a conveyor was broken down, the mining sections still carried out production of material that was accommodated in the surge bunkers, which were always running with some available space inside. Once the conveyor's operational status was recovered, the trunk belts had enough spare capacity to remove the excessive y accumulated material from the system. In the event of the utilisation of all the equipment at the maximum design capacity, that result would have been much more difficult to achieve.

2.4. Results of the Case Study

(1) An algorithm was developed to control the feed of the mining sections into the surge bunkers and the discharge rate onto the trunk conveyor belts for an underground mine. The control objectives were to prevent the overflow of the bunkers, eliminate blockages of the trunk belts due to overload and maintain smooth, uninterrupted operation.

(2) The efficiency of the control algorithm was verified with the simulation model, and the results of experimentation with the model proved that the objectives of the control algorithm were achieved.

Table 2. Effects of breakdowns

Description	**Unit**	**No Breakdown**	**Breakdown Induced**
Total average delivery rate	t/h	2,660	2,555
Utilisation C07	%	92	81
Utilisation C08	%	91	76
Utilisation C05	%	94	85
Utilisation C06	%	95	95
Utilisation D01	%	96	86
Utilisation D02	%	96	98
Utilisation C13	%	96	86
Utilisation C14	%	96	98
Note: Broken-down conveyors are shaded			

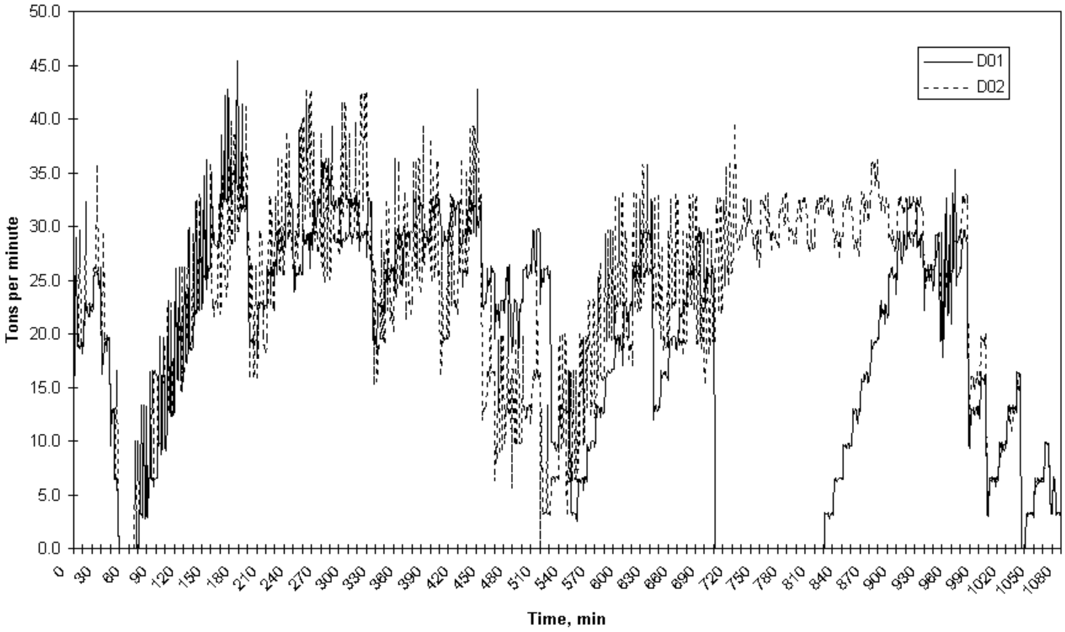

Figure 9. Effect of breakdowns on trunk conveyor delivery

(3) The algorithm was then implemented in a new PLC-based process control system installed on the mine after completion of this study. According to feedback from the operators, the surge bunkers have been running at ±60-70% average level and no overloads of the trunk belts have occurred.

(4) The problem identified at that stage was insufficient throughput of the mining sections resulting in the average delivery rate of the system at ±60% of its design capacity.

3. Case Study: Sizing Bulk Handling Equipment

3.3. Introduction to and objective of the Simulation Study

An existing Mine 1 mines material, which is conveyed into bunker 1, then via conveyors C1 and C2 into an existing stockyard 1, comprising 2 x 20,000 t stockpiles (see Figure 10). From the stockyard, material is reclaimed and conveyed via conveyor C3 into an 8,000-ton silo, feeding material to processing plants 1 and 2 via conveyor C4, link 2.

The expansion plans of the mine involve establishment of a new underground operation (Mine 2), deploying six to 16 mining sections that will feed the new incline conveyor C5. In each section, a feeder-breaker will be installed to allow for tipping of material from the shuttle cars onto a conveyor belt, C5 will feed a new bunker 2, which will feed material to a new overland conveyor C6. A new stockyard may be required to accommodate the feed from the new mine or, alternatively, the capacity of the existing stockyard may be adjusted accordingly.

It is supposed that material from the new mine will be predominantly supplied to plant 2, while plant 1 will consume material mainly from the existing mine. However, the old mine must be able to feed material to plant 2 if required; it is considered higher priority due to the absence of any entry buffer, and once a stockpile is exhausted, the plant will be idling.

The objectives of the simulation study were therefore as follows:

(1) Size the new surge bunker 2, taking into account the feed of the new incline conveyor C5 delivering material from six to 16 underground mining sections to be quantified;

(2) Configure and size the stockyard to accommodate material coming from the existing Cl and the new overland conveyors, and to provide feed into material processing plants 1 and 2 as uninterruptible as possible at required rates of 1,200 and 2,000 t/h, respectively. Configuration of the stockyard entails investigation of the sufficiency of the existing capacity and a definition of the new required capacity if that becomes necessary.

3.2. Simulation Approach and Assumptions

Due to the integral nature of the task, the following continuous modelling approach was used to simulate

Table 3. Parameters of simulation scenarios

Description	Scenario 1	Scenario 2	Scenario 3
Stockyards	Existing Only	Existing and New	Existing and New
Capacity of stockpiles (t)	2 x 20,000 and 2 x 40,000	Old: 2 x 20,000 New: 2 x 25,000	Old: 2 x 20,000 New: 2 x 25,000
Capacity of stacker (t/h)	4,500	Old: 1,500 New: 3,000	Old: 1,500 New: 3,000
Capacity of reclaimer (t/h)	3,200	Old: 1,200 New: 2,000	Old: 1,200 New: 2,000
Capacity of new overland conveyor (t/h)	3,000	3,000	3,000
Capacity of new surge bunker (t)	5,000	5,000	5,000
Max capacity of new incline conveyor (t/h)	3,600	3,600	3,600

the operation of the new overland conveyor system and the existing facilities.

The mining operation was simulated as some virtual source of material providing inputs of the material into the system with specified rates. Two inputs were modelled, feeding the existing conveyor Cl and the new incline conveyor C5.

To define the size of the new surge bunker 2, a level monitor was modelled. The reading of the monitor was updated every time it registered a higher level of material in the bunker. The following considerations were taken into account to size the new surge bunker:

(1) The size of the bunker should prevent blockages of the feeding conveyor C5; i.e., it must preferably never overflow;

(2) It is also important that the bunker never remain empty continuously, maintaining the required feed into the stockyard.

Table 4. Table of availability

Description		Schedule 1 (Mine)	Schedule 2 (Plant)
Applicable to equipment		C5	C1, C4, C9
Basis of stoppages		Weekly	Weekly
Working time per week (min)		4,800	8,640
Availability 85%	MTBF (min)	EXP (4,080)	EXP (7,344)
	MTTR (min)	GAMMA (0.5, 1,440)	GAMMA (0.5, 2,592)
Availability 87%	MTBF (min)	EXP (4,176)	EXP (7,512)
	MTTR (min)	GAMMA (0.5, 1,248)	GAMMA (0.5, 2,256)
Availability 90%	MTBF (min)	EXP (4,320)	EXP (7,776)
	MTTR (min)	GAMMA (0.5, 960)	GAMMA (0.5, 1,728)
Availability 93%	MTBF (min)	EXP (4,464)	EXP (8,035)
	MTTR (min)	GAMMA (0.5, 672)	GAMMA (0.5, 1,210)

Table 5. Comparison of material production

Description	Scenario 1		Scenario 2	Scenario 3
Stockpiles	2 x 20,000 t	2 x 40,000 t	2 x 20,000 t + 2 X 25,000 t	2 x 20,000 t + 2 X 25,000 t
Volume entered (tpa)	18,862,300	20,239,500	18,184,000	15,533,701
Volume shipped (tpa)	18,846,800	20,216,400	18,148,400	15,515,378
Current volume in system (t)	15,526	23,096	35,606	35,352
Average volume in system (t)	16,249	23,808	39,978	32,536
Average time in system (rein)	453	618	1,156	1,066

The capacity of the existing stockyard should maintain the maximum utilisation of the reclaimer and the reclaim conveyor; i.e., the reclaimer must always have some material in one of the two stockpiles to reclaim. Therefore, the definition of the required size of the stockyard was based on the minimisation of the idling time of the reclaimer-conveyor combination.

Three scenarios were simulated after discussions of the model with the mine and are shown in Table 3.

Scenario 3 was considered specifically to deal with interruptions of the feed into plant 1. In that event, it was suggested to shut down Cl and route to Plant 1 material coming from C6 via link 3, C2 and link 1, because Plant 1 was highly prioritised.

The existing mining operation delivers material to the Cl conveyor at a-rather stable rate provided by the belt weigher and surge bunker 1 feeding up the conveyor to maintain a 1,400- to 1,500-t/h feed. The feed rate of conveyor Cl was therefore assumed to be a uniform distribution with a minimum of 23.5 and maximum 25.0 t/m (average 1,455 t/h).

The feed of a new mining section was described as a square negative parabola with a maximum in the middle of a shift and zero production at the beginning and at the end of the shift. The average feed of a section over a one-shift period was assumed at ±220 t/h, as per a similar operation in one of the operating underground mines.

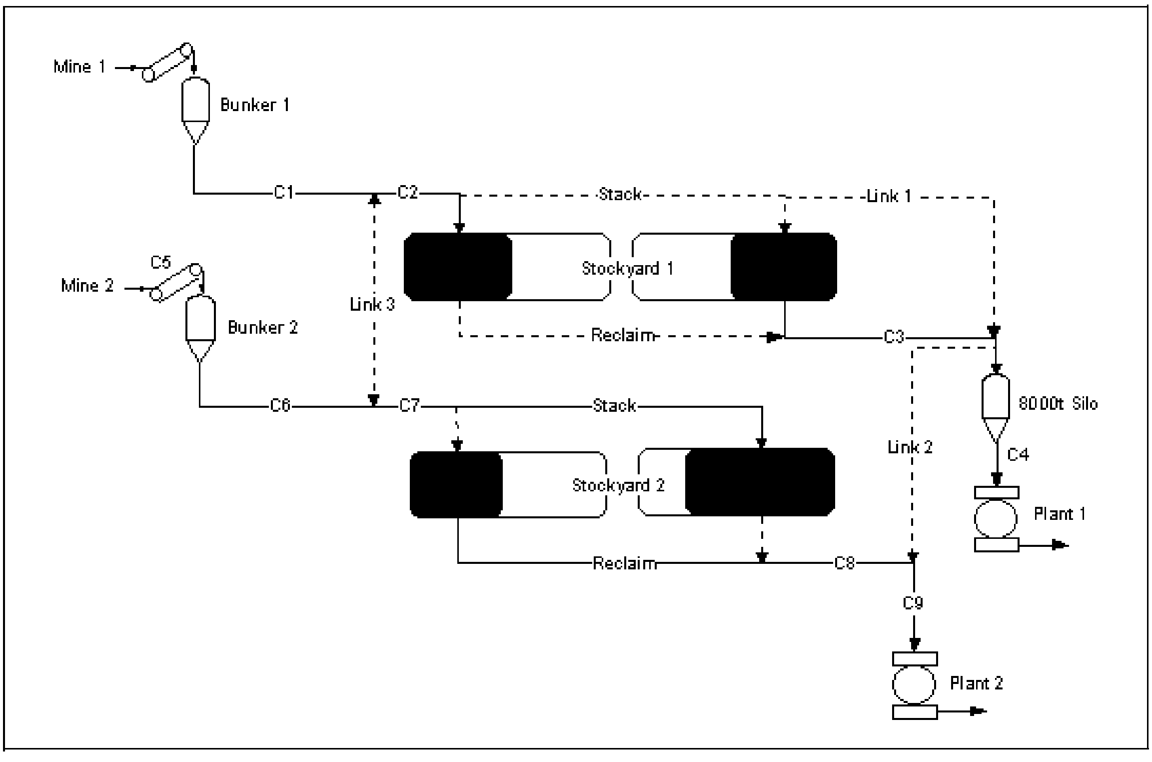

Figure 10. Material flowchart

Table 6. Results of simulation of conveyors

Scenario	Code	Volume				Avg Time in Syst	Percentage of Time			
		In	Out	Curr In	Avg In		Down Time	Idle	Flow-ing	Off-Shift
1:2 x 20,000 t	C1	9,783,150	9,782,940	205	205	9	0.00	5.65	94.35	18.63
	C5	9,055,130	9,054,980	150	147	4	0.00	7.71	92.29	53.79
	C6	9,054,980	9,054,980	0	314	15	0.00	52.38	47.62	18.63
	C2	8,199,040	8,199,040	0	9	0	0.00	64.96	35.04	18.63
	C3	16,372,200	16,372,200	22	16	0	0.00	26.94	73.06	18.63
	C7	o	0	0	0	0	0.00	100.00	0.00	18.63
	C8	0	0	0	0	0	0.00	100.00	0.00	18.63
	C9	10,587,700	10,587,600	100	96	4	0.00	3.34	96.66	18.63
	C4	8,259,220	8,259,120	100	97	5	0.00	3.29	96.71	18.63
1:2 x 40,000 t	C1	10,361,100	10,360,900	205	205	8	0.00	0.08	99.92	18.63
	C5	9,834,350	9,834,200	150	147	4	0.00	0.04	99.96	53.79
	C6	9,834,200	9,834,200	0	288	13	0.00	46.47	53.53	18.63
	C2	9,356,300	9,356,300	0	10	0	0.00	55.85	44.15	18.63
	C3	18,859,800	18,859,700	22	19	0	0.00	14.66	85.34	18.63
	C7	0	0	0	0	0	0.00	100.00	0.00	18.63
	C8	0	0	0	0	0	0.00	100.00	0.00	18.63
	C9	13,072,100	13,072,000	100	95	3	0.00	4.63	95.37	18.63
	C4	7,144,510	7,144,410	100	85	5	0.00	14.04	85.96	18.63
2	C1	8,559,960	8,559,750	205	205	10	0.00	16.53	83.47	18.63
	C5	9,572,030	9,571,880	150	147	4	0.00	2.24	97.76	53.79
	C6	9,574,880	9,574/880	0	292	13	0.00	47.82	52.18	18.63
	C2	4,024,580	4,024,580	0	11	1	0.00	61.09	38.91	18.63
	C3	8,062,220	8,062,200	22	22	1	0.00	5.67	94.33	18.63
	C7	9,574,880	9,574,880	0	17	1	0.00	50.80	49.20	18.63
	C8	9,594,880	9,594,640	236	160	7	0.00	30.53	69.47	18.63
	C9	9,594,640	9,594,440	200	136	6	0.00	30.80	69.20	18.63
	C4	8,554,040	8,553,940	100	100	5	0.00	0.00	100.00	18.63
3	C1	8,315,082	8,314,978	205	202	10	0.00	17.34	82.66	18.63
	C5	7,191,705	7,191,601	150	147	5	0.00	16.13	83.87	53.79
	C6	7,193,050	7,192,843	403	350	21	0.00	49.45	50.55	18.63
	C2	4,031,440	4,031,440	0	11	1	0.00	61.04	38.96	18.63
	C3	8,065,862	8,065,862	22	21	1	0.00	5.65	94.35	18.63
	C7	6,685,812	6,685,812	0	20	1	0.00	55.38	44.62	18.63
	C8	6,693,214	6,693,110	236	112	7	0.00	49.91	50.09	18.63
	C9	8,601,982	8,601,930	100	100	5	0.00	0.00	100.00	18.63
	C4	6,913,500	6,913,448	27	49	3	0.00	49.53	50.47	18.63

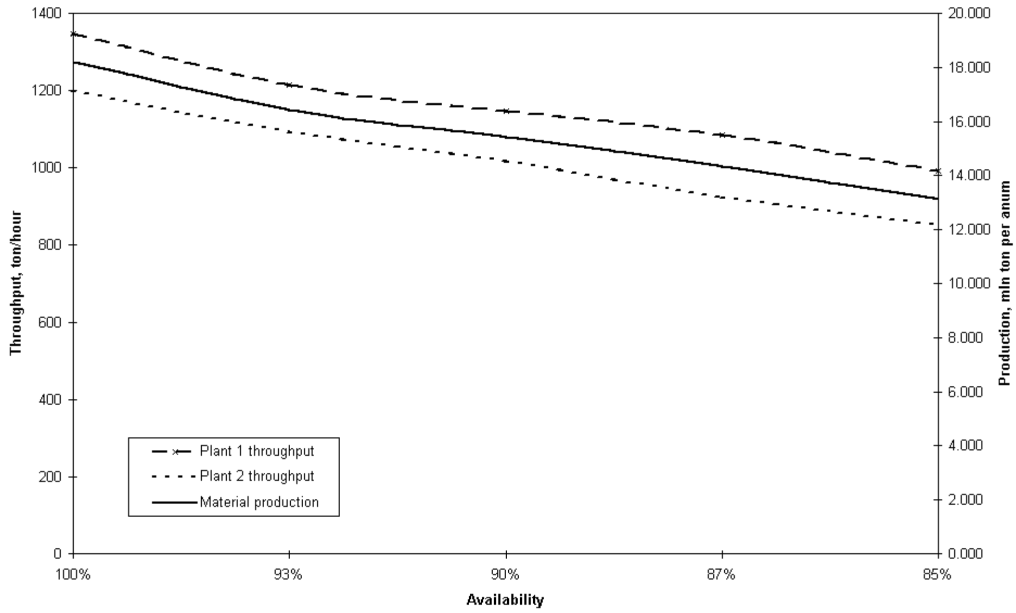

Figure 11. Effect of equipment availability on production and throughput of material

Conveyor C5 conveys material into the new surge bunker 2, feeding the new overland conveyor at the controlled average rate of 3,000 t/h.

Reclaim rates from the stockpiles and discharge rates from the surge bunkers were sampled from a normal distribution with mean values as per the relevant parameters of Table 3.

Stacking and reclaiming took place on the alternative basis with reclaiming as first priority (a stacker could not interrupt a reclaimer; however, once reclaiming was finished, the stacking had to be stopped and the reclaimer moved to the other stockpile). A change-over time of 45 minutes was allowed for stacker and reclaimer motion and setup.

No simulation of plants 1 and 2 was required; however, defining the effects of the breakdowns and maintenance stoppages on the throughput of the plants was delegated to conveyors C4 and C9 feeding the plants.

Failures of both the mine and the plant were assumed to occur on a weekly basis. The MTBF was simulated by the negative exponential distribution, and the MTTR by the Gamma distribution to obtain the above availability as per standard statistical methods (see Table 4).

Further research on the availability effect was undertaken with respect to the key equipment in the stock yard - namely, stackers, reclaimers with associated conveyors, and the new overland conveyor. The 98% availability of that equipment was assumed and calculated exactly in the same way as for the above equipment.

As in the previous case study, a full calendar year operation according to the actual mine work schedule was simulated.

3.3. Discussion of Simulation Results

The level monitor in bunker 2 registered ±4,900 t maximum volume of material; therefore, it was decided to retain the size of the bunker at 5,000 tons with a maximum of 11 underground sections concurrently mining material.

The simulation report on production of material is presented in Table 5.

The best production figure is obtained for the existing stockyard with 2 x 40,000-ton stockpiles (see scenario 1 in Table 3).

Performance of the system in scenario 1 (with 2 x 20,000 t stockpiles) and in scenario 2 is similar. Although intuitively one would think that the throughput capacity of a stockyard with 2 x 20,000 t and 2 x 25,000 t stockpiles would be better than for the 2 x 20,000 t stockpiles, it is necessary, however, to keep in mind that the loss in the throughput of a system with one stacker and one reclaimer will be smaller than that of the system with two stackers and two reclaimers due to the time spent on changing over. Another fact is that the shorter the average change-over time of stackers/reclaimers, the better the utilisation of the new incline conveyor C5 will be.

A further decrease in material production in scenario 3 is explained by the fact of shutting down conveyor Cl and re-routing material coming from the

Table 7. Results of simulation of storage facilities

Scenario	Name	Volume				Avg Time in Sys	Percentage of Time			
		Entered	Shipped	Curr In	Avg In		Empty	Full	Partly Full	Off-Shift
1: 2 x 20,000 t	Bunker 2	9,054,980	9,054,980	0	1,522	72	46.52	6.76	46.72	18.63
	S/pile 1/1	8,167,680	8,167,330	342	7,742	405	13.73	5.21	81.06	18.63
	S/pile 1/2	8,219,040	8,204,920	14,119	7,811	406	14.16	5.27	80.56	18.63
	S/pile 2/1	o	0	0	0	0	100.00	0.00	0.00	18.63
	S/pile 2/2	o	0	0	0	0	100.00	0.00	0.00	18.63
	Bunker 1	8,259,340	8,259,220	118	126	7	6.76	0.00	93.24	18.63
1: 2 x 40,000 t	Bunker 2	9,834,200	9,834,200	2	706	31	56.73	0.03	433.24	18.63
	S/pile 1/1	9,485,000	9,468,210	16,792	12,150	548	7.42	0.13	92,45	18.63
	S/pile 1/2	9,396,300	9,391,570	4,730	12,179	554	8.08	0.01	91.91	18.63
	S/pile 2/1	o	0	0	0	0	100.00	0.00	0.00	18.63
	S/pile 2/2	o	0	0	0	0	100.00	0.00	0.00	18.63
	Bunker 1	7,145,140	7,144,510	625	320	19	61.65	0.00	38.35	18.63
2	Bunker 2	9,574,880	9,574,880	0	914	41	55.32	1.95	42.73	18.63
	S/pile 1/1	4,040,000	4,020,000	20,000	10,831	1,147	2.15	11.12	86.73	18.63
	S/pile 1/2	4,044,580	4,042,240	2,343	10,826	1,145	2.18	11.11	86.71	18.63
	S/pile 2/1	4,125,140	4,125,140	0	4,996	518	34.67	0.06	65.27	18.63
	S/pile 2/2	5,474,740	5,469,730	5,005	6,065	474	25.54	0.02	74.44	18.63
	Bunker 1	8,560,970	8,554,040	6,925	6,753	337	0.00	10.90	89.10	18.63
3	Bunker 2	7,193,154	7,193,050	167	1,854	110	39.25	18.43	42.33	18.63
	S/pile 1/1	4,035,969	4,026,849	17,624	10,789	1,144	2.85	10.95	86.19	18.63
	S/pile 1/2	4,041,792	4,039,049	5,301	10,792	1,142	2.89	10.95	86.17	18.63
	S/pile 2/1	3,349,174	3,349,097	154	4,734	605	34.01	0.08	65.91	18.63
	S/pile 2/2	3,349,573	3,344,133	10,506	4,626	591	34.97	0.04	64.99	18.63
	Bunker 1	6,913,552	6,913,500	88	52	3	54.37	0.00	45.63	18.63

new overland conveyor C6 directly to the transfer tower while there was no feed into plant 1. When Cl was stopped, it didn't deliver any material into the system, which contributed to the loss in production.

Performance parameters of conveyors are broken down in Table 6. The overall best utilisation of conveyors was observed for scenario 1 with 2 x 40,000 t stockpiles, which fully correlates with the results of material production.

The utilisation of conveyor C8 in scenario 2 is rather worse than that of C3. This indicates that the operation of the new stacker and reclaimer was less efficient that of the existing ones. The reason for that could be the size of the new stockpiles, which caused the stacker and reclaimer to move more frequently than in the existing stockyard (compare 16.7 hours required to reclaim the full old stockyard to 12.5 hours to reclaim the full new one).

In scenario 3 the situation reversed: conveyor C4 performed better than C9 due to the preferential feed of the former (remember, the control algorithm had an objective to maintain uninterrupted feed to plant 1).

Parameters of the stockpiles and bunkers are summarised in Table 7.

3.4. Throughput of the Plants

It is interesting to note that with much better throughput of plant 1 in scenario 2 with 2 x 40,000 t stockpiles, the productivity of plant 2 scaled down by some 15%. In scenario 2, both plants performed very similarly, although plant 1 was supposed to process more material. The only obvious reason for that could be less efficient operation of the new stacker and reclaimer as discussed in Section 3.3. Overall material production and plant throughputs versus equipment availability are plotted in Figure 11.

3.5. Recommendations from the Case Study

As a result of simulating experimentation, the following conclusions can be made:

(1) The absolute maximum number of underground sections that can mine and feed material into the new surge bunker concurrently is 11, with the capacity of the new incline conveyor of 3,600 t/h.

(2) The new surge bunker sized at 5,000 t will be able to accommodate the irregularity of the feed coming from the underground mining operation.

(3) A capacity of the new overland conveyor of 3,000 t will suffice to deliver the amount of material required for the plants and to balance the feed coming into the surge bunker with the above two recommendations followed.

(4) The existing stockyard at its current capacity of two stockpiles of 20,000 t each will maintain the annual production of material exceeding 18 million tons and sufficient throughput of the material plants at 100% availability of equipment if the capacity of the existing stacker can be upgraded to 4,500 t/h and of the reclaimer to 3,200 t/h.

(5) The stockyard with two new stockpiles of 25,000 t each, a new stacker of 3,000 t/h, a new reclaimer of 2,000 t/h and associated conveyors will also provide the production of material of more than 18 million tons per annum and a sufficient feed through the plants with no breakdowns in the system.

(6) If the availability of equipment degrades to 85°/0, a loss is expected of ±4 million tons of annual production of material, from 18 million to 14 million tons.

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