CONVEYOR SPILLAGE – WHAT ARE WE MISSING?

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SYNOPSIS

This paper examines the implications of excessive spillage on conveyor systems primarily from a design perspective. The focus of the paper is on how spillage should be controlled and what the factors and influences are that govern these issues under operating conditions.

The subject of spillage on conveyors is examined in detail. The adequacy of modern day methodology for determining conveyor belt capacities is reviewed and examined for relevance. SO 5048 is used as the initial point of departure.

Transfer points in mines are discussed and key factors tabled for review regarding the use of brakes and flywheels on installations. Adequacy of transfer chutes is examined.

The principle of conveyor stopping is discussed in line with legal requirements. Controlled stops and emergency stops are reviewed for establishing relevance and application.

Conclusions are deduced and tabled in conjunction with the relevant recommendations on the way forward.

1. INTRODUCTION

Spillage on conveyors has been part of the bulk material handling industry to the point where it is considered as being a foregone conclusion. The only variable is the volume of spillage on any one conveyor system. The purpose of this paper is to review the principle causes of spillage with the aim of deriving corrective actions for overcoming this problem.

Spillage as a subject on its own is a major safety hazard. The safety aspects enter the equation as soon as the spillage is loaded back onto the ore handling system. This is a man-machine interface which, if it were not there in the first instance, would not be necessary, thus making the production environment a much safer place.

The buzzword in the industry is that a clean working environment is a safe working environment. This is a universally proven phenomenon. Spillage, however, results in the workplace not meeting this criterion. It is thus critical that the subject of spillage is reviewed with the aim of drilling down to expose the reasons behind spillage occurring in the first place.

This paper is structured by having an assumption section, discussion section, conclusion section and finally a section on recommendations. In the assumption section all the facts are noted with which the reader is considered to be familiar and which require no further explanation. The discussion section examines the topic in detail from various angles and the resulting conclusion section extracts the final logic from the discussions with appropriate recommendations being tabled accordingly.

2. ASSUMPTIONS

It is assumed that the reader is familiar with the fact that statistically the most dangerous and lethal section on a conveyor is the tail pulley. The second most dangerous area is the drive and take-up section.

3. DISCUSSION

But is spillage really an issue? Consider these photographs of spillage on conveyor systems.

Figure 1. Spillage photos

The aim of the photographs is to visually capture and act as a reminder as to the reality of the spillage issue. Expanding on the photos, spillage is initially discussed as a general topic. Subsequent to this, conveyor carrying capacities are examined in greater detail as well as transfer points. Note that it is not the intention to cover all the detail in full as this is a specialist subject and only salient points are addressed.

TALKING ABOUT SPILLAGE

Discussing spillage from a safety critical point of view is not difficult. Spillage, to say the least, is an unnecessary maintenance cost, safety risk and a nuisance factor on all conveyor installations. The key objective of a conveying system is to transport material from one point to another and the purpose is to deliver every single particle of the material to the final destination.

When material falls off the belt it ultimately needs to be put back onto the belt. At the point of origin, which is the mining face, the material is loaded onto the belt by mechanical means. This could typically be done by utilising feeder breakers, vibrating feeders loaded by means of load haul dumpers (LHD) and similar.

These machines are bulky and cannot be readily deployed for clearing spillage along the length of the conveyor or at the transfer points. Due to the cost of these machines it is simply not viable for them to be trammed next to the conveyor system to clear spillage. The operational side of this is that due to the bulk of these machines, they end up doing more damage to the conveyor structure than good.

Spillage is normally cleared by hand and this is where the danger lies and where spillage becomes a safety issue. Shovelling material is a tedious task and soon tires the individuals, especially underground. This leads to fatigue and under these conditions accidents are surely more likely to happen, and do happen. Clever ideas have been put into place like having a shovel with a removable front. The purpose of the removable shovel blade is that when it gets caught in the machinery the blade is readily dislodged from the handle ensuring to a large extent that the operator is not jerked into the system. Despite this, accidents happen.

How then, is spillage prevented? In essence there are only two situations that arise which cause spillage on a conveyor system. The first is differential material volumes at transfers and overloading of the outgoing conveyor system. The second is the central loading of the material at the transfer point and the tracking of the conveyor at this point. The irony is that none of this is rocket science and yet inevitably industry seems to be getting it wrong.

Holistically there are then two scenarios to consider. The first is the detail at the transfer point and the second is the detail of the carrying capacity of the conveyor system.

The Detail at the Transfer Point

At the transfer point the requirement is for the material to be deposited centrally on the receiving conveyor and for the material to land on the receiving conveyor preferably at a similar speed to which the receiving belt is travelling. Managing to get the speed requirement right leads to less turbulence in the material thus making it less likely for spillage to occur.

At every transfer point the height between the discharging and receiving conveyors is critical and from a design perspective is dependent on the speed of the material, discharge angle, the material trajectory, internal material friction values, impact pressure values, receiving conveyor angle, lump size and moisture content.

There are modern day computer based software and methodologies used by design engineers to predict the material characteristics. This situation is easy to observe and one can readily see when things go wrong in this area. Tell-tale evidence is the presence of dust at the transfer point as well as high noise levels. The general assumption is that transfers on average operate adequately under normal running conditions. The material flow can be observed and site interventions applied to achieve the required outcome.

The point that is not addressed is the differential stopping times between conveyors. The rule of thumb in the industry is that as long as the conveyor stops within 20 seconds there should be no spillage problems, but how true is this?

Conveyor specialists boldly state that a conveyor must be designed such that there are no shockwaves developed during the dynamics of the starting and stopping cycles. While the theory behind this is correct, it leaves the operator of the system not understanding when spillage needs to be cleaned up under operating conditions due to overruns.

It becomes essential that the user specifies or states that he requires the minimum differential in stopping time between consecutive conveyors. Fitting of brakes and flywheels thus needs to be considered for all conveyor applications.

More Information on Brakes and Flywheels

The first step is to develop a deeper insight into the constant torque machine that is being operated. The practical way to illustrate a principle is by means of an example. Consider a conveyor running at a belt speed of 3 m/s conveying material at the rate of 1620 tonnes per hour. This equates to a material loading of 150 kg/m. The material mass along the length of a conveyor that is 1000 m long is 150 x 1000 = 150 tonnes of material. The similar application where the conveyor is 100 m long is 150 x 100 = 15 tonnes of material.

Stopping logic needs to be applied to these two scenarios. In the first instance a mass of 150 tonnes needs to be stopped and the second option is to stop a 15 tonne mass. Briefly comparing them to trucks as a frame of reference the one application is a 150 tonne truck and the other a 15 tonne truck. Everyone understands that one cannot stop this magnitude of mass over a very short distance. It takes time. The shorter the stopping time, the greater the force that is required to exert the stopping action.

By further analogy, in the mechanical design of the equipment, the relevant pulley shafts are sized to function reliably under normal operating conditions. When considering the stopping forces acting on the system, a point is reached where the stopping force becomes the design criteria and not the functional requirement. It is at this point that the designer must make the call and state that the maximum stopping force will be equivalent to what the functional design requirement is. This approach invariably leads to a balanced design.

Should the requirement be for the conveyor to be stopped due to an emergency, possibly due to an accident, or as a material flow control measurement, the inertia of the system always results in the system coming to rest over a period of time. It is the writer's conviction that the reader is as safety conscious as himself. When the stoppage is required due to an accident, cold as it may sound, stopping the conveyor is ineffective as the damage to life and limb has already occurred due to the magnitude of the forces involved. The principle remains to take the individual out of harm's way as opposed to stopping this huge mass.

The right perspective is that if the equipment is stopping and the strength criteria is the magnitude of the stopping force, a point is reached where that equipment will fail and this will more than likely be a catastrophic failure with real danger to the innocent. Failure modes under these conditions could typically be belt breaks, pulley shafts snapping and structures failing. The rule of keeping personnel away from the equipment remains the best solution.

Based on this analogy, fitting of flywheels and brakes become a viable option. By way of an example and for illustration purposes, reference has been made to a 15 tonne and a 150 tonne truck. These trucks are assumed to be travelling at the same speed and can only coast to a stop once they are switched off. The heavier unit will invariable take longer to come to a stop due the inertia of its mass. Building further on this logic the following applies:

Fitting a suitable flywheel implies that the stopping time of the 15 tonne truck can mimic that of the 150 tonne truck. Fitting a suitable brake to the 150 tonne truck makes the stopping time comparable with that of the 15 tonne truck. The associated conveyor logic behind this is that the stopping time between consecutive conveyors should be the same. This has the effect of no overrun of material at the transfer point from one conveyor to the other. No spillage needs to be fed back onto the conveyor and thus no need for man-machine interface. The use of brakes in conjunction with flywheels further enhances the control of stopping time between conveyors which can only lead to positive results.

Fitting a combination of brakes and flywheels empowers the conveyor designer to design and develop a conveyor system with minimal to zero spillage. Within the application of reasonable braking and flywheel requirements and forces, conveyor designers are in a position to address design criteria by designing conveyor systems free from transient forces during starting up and stopping.

Concluding the thought process on differential stopping; the material overrun needs to be quantified. The long conveyor may take 16 seconds to stop while the short conveyor takes 8 seconds. The coasting time is considered as being linear and the average time is the volume of material to be considered in the overrun. Both units operate at the same speed being 3 m/s.

The long conveyor will thus still discharge 16 sec ÷ 2 = 8 seconds of material.

This is equal to 8 seconds x 150 kg/m x 3 m/s = 3600 kg of material.

The short conveyor will thus discharge 8 sec ÷ 2 = 4 seconds of material.

This is equal to 4 seconds x 150 kg/m x 3 m/s = 1800 kg material.

Should the long belt discharge on the short belt the material prone to spillage will be:

3600 kg – 1800 kg = 1800 kg material.

The belt is carrying 150 kg material per metre and now there is 1800 kg loaded on say, an extent of one metre. Spillage is inevitable.

Chute Design Considerations

The physical design of the chute plays an important part in limiting spillage. In principle all chutes must be designed to act as surge bins and feeders for material transfer.

The surge capacity inside the chute is required to store the overrun of material and must be based on the worst case scenario. The worst case scenario is the longest time the receiving belt is stationary while the feeding belt is still in the process of coming to a stop.

Once the overrun of material is in the chute, the chute outlet must be designed to centrally feed the overrun of material onto the receiving belt under controlled conditions. Effectively the outlet of the chute must be throttled so that, without blocking, it only allows the maximum volume of material which will fill the receiving belt through, preferably to the normal running material cross-section.

Needless to say, the receiving belt must start and operate as a feeder type belt until all the overrun is cleared and only then accelerate and brought up to the normal running speed of the conveyor. Starting as a feeder implies that it runs for a period of time as a feeder at less than typically 1 m/s until the overrun is cleared.

As alluded to earlier, another requirement for transferring material from one belt to another is for the material to be centrally loaded onto the outgoing conveyor. Yet again this is a broad statement to make in relation to the geometry at the transfer point. Inherent in every transfer is the fundamental property for the straight line inertia of the transferred material to continue along the path of its travel and discharge.

Within the transfer chute the flow of the material needs to be controlled. Typically impact plates and dead boxes are used for exerting some control over the material flow. Generally, in a 90 degree transfer, the dead box arrests the flow of the material but then, depending on the stream volume, the material reverses direction as it flows over the dead box back in the direction it came from. There is now a positive flow of material from the one side to the other side on the outgoing conveyor belt.

For illustration purposes, assume the discharge is at 90 degrees to the feeding belt. The forward inertia of the material is arrested in the dead box. Once the material starts flowing over the edge of the dead box the material has a positive force component from left to right onto the outgoing conveyor. Typically a thin stream of material lands on the left hand side and as the stream volume increases the material moves over from the left to the right.

The thin stream of material results in the belt being loaded more on the left hand side of the outgoing belt. Similarly, as the stream volume increases, the material begins to fill the full width of the belt up to the point where it starts overfilling the belt on the right hand side. Somewhere between these two extremes is an average value which ends up being used as being the perceived point for transferring the material, which is not always correct.

The moisture content of the material shows similar characteristics. The angle of the material in the dead box now becomes steeper as the flow properties are affected. The increased angle causes the material to flow at a different speed over the dead box which affects the stream volume again. The difference in the speed component of the material flowing from left to right ultimately determines where the material lands on the outgoing belt. The end result is similar to that previously described.

Sufficient height is required to correctly guide the material from one conveyor onto the next one. Simplistically, the following happens on a correctly designed chute. The principle of application is to arrest the flow of the material being discharged over the head pulley of the discharging conveyor. With the material stream now effectively falling in a vertical plane it is thus contained with the end result being that it can effectively and repeatedly be guided in the direction of the outgoing belt.

Another buzzword in the industry is designing a chute using discrete element modelling. A discrete element model is an arrangement of chute plate work within which the flow of material is simulated. The model is a mathematical approach where developers analyse the flow of a material stream while simulating the interactions of all the particles between one another and the chute enclosure through which the material is passing. Without going into too much detail, the capability of DEM systems is now approaching the magnitude of 1 000 000 particles. This is an enormous achievement but the current systems are not capable of modelling very small particles as the numbers are too great for processing.

The results of a discrete element model are used to create a video file of the transfer chute. To run a two minute video file can easily take up to three days of dedicated computing time. The reader should now have a basic understanding of the discrete element approach.

It must be clearly understood that a model of this nature can only be used to simulate the flow. This process highlights problem areas within the chute which still need to be rectified. DEM, as it is currently practiced, will not definitively quantify the chute angle which needs to be altered from say, 45 degrees to 47 degrees. DEM only simulates flow and highlights a problem. Correcting the problem is still classed as a hit or miss situation. Once the chute angle is changed from 45 degrees to 47 degrees, the simulation must be run again and checked for acceptability. In real terms it remains a best educated guess.

Chutes should be designed by using a single particle analysis approach. The engineer analyses the entire flow pattern of the required chute. Once this is complete, the three dimensional model is then built with the aim of conducting a discrete element flow analysis. Should the designer notice a problem, the calculations are reviewed accordingly and chute angles adapted to suit. The DEM model is then rerun. Against the background of existing cutting edge technology, this is probably the most efficient way to design a chute, if not the only way.

But is efficiency of a transfer dependant on the chute design only? Just as important to note is the receiving section of the outgoing conveyor. Credit goes to those designers who now give consideration to the receiving section. At Beltcon, papers were presented on using deep troughing idlers and extending this application to the material loading section.

This logic must be consciously applied at the receiving section as well. To this end consideration is now being given to changing the standard 35 degree three equal roller idler and troughing it at 45 degrees at the loading point. This a good start but certainly not the full solution. More on this later. A full understanding of the carrying capacity of a conveyor is required first.

The Detail of the Carrying Capacity

As a clarification comment, the purpose of this paper is not to discuss the carrying capacity of conveyors relative to power but only to make reference to capacities related to spillage. It is thus assumed that the installed power is adequate to cater for all loading conditions.

It is common practice for conveyors to be designed with a minimum freeboard distance. This is normally quantified as being a ratio of the belt width and according to ISO 5048, this value is stated for the average belt width as being a minimum of 5% of belt width plus 25 mm. The reason for this is that conveyor belts will never run 100% on line all of the time simply due to varying loading conditions, varying material properties, conveyor geometry and starting and stopping dynamics.

This value is currently in place as an agreed distance used by all conveyor designers for minimising spillage. This paper reviews the relevance. To get to the bottom of this a recap is needed on the definitions currently being used in the industry.

There are two definitive loading conditions on a belt conveyor: the flooded and the full, and arbitrarily the third is the economical carrying capacity.

Flooded belt (definitive)
The flooded belt condition is when the material on the belt is loaded from the one edge of the belt to the other edge of the belt.

Fully loaded belt (definitive)
The fully loaded belt condition is when a conveyor belt is loaded to the point where the edge of the material coincides with the freeboard distance position on the belt.

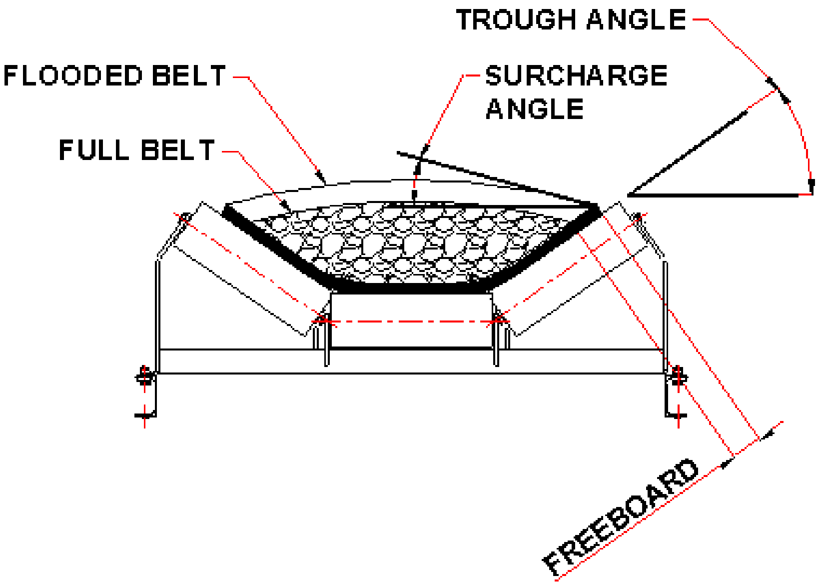

Figure 2. Flooded and full belt

Economical Carrying Capacity

The economical carrying capacity of a conveyor is always expressed as a percentage of the full capacity of the conveyor. There are various methodologies and approaches that can be used in this regard and they are all valid. The recommendation is for this capacity to be pitched at 90% of the fully loaded belt.

In the CMA conveyor design course there is an official range of considerations noted for determining this value. Among these are factors like making up for lost production, increasing capacity on the same system at a later date and lastly, simply allowing for spillage. For the convenience of the reader these considerations are noted below.

Extract from the CMA conveyor design course material:

DESIGN CAPACITY (Power requirement)

If we summarise the determination of the design capacity then, we can say that:-

$$C_{dc} = (C_{fs} + C_3) \times C_1 \times C_2$$

Where

C_{dc} = *Design Capacity t/h*
C_{fs} = *Dry Capacity t/h*
C_1 = *Margin factor*
C_2 = *Feeder factor*
C_3 = *Entrained moisture t/h*

VOLUMETRIC SIZING FACTOR C_4 - THE BELT WIDTH (Spillage requirement)

In addition to the factors listed above, we can *artificially* increase the volume of the material, in order to give us a greater freeboard on the belt and therefore reduce the incidence of spilling. The standard factor for valuable materials, such as diamondiferous ores, is **1,2**, while higher values can be used on individual conveyors. For diamond concentrates, we use a factor of **1,65** as a minimum. The belt speed (S) in m/s is then obtained from the equation:

$$\boldsymbol{S} = \frac{(\boldsymbol{C_{fs}} + \boldsymbol{C_3}) \times \boldsymbol{C_1} \times \boldsymbol{C_2} \times \boldsymbol{C_4}}{\mathbf{3600} \times \boldsymbol{D} \times \boldsymbol{A_{100}}}$$

Where

A_{100} = The maximum cross sectional area (m^2) of the material on the belt, with the material edge on the standard freeboard as determined in accordance with ISO 5048.

D = The bulk density of the material (t/m^3).

With all this information and considerations being available, why does spillage then remain an issue? The perception is that there is a lack of understanding when interpreting these considerations in practice. In the first instance the problem is perceived to be the law of averages and in the second not making provision for lumps in the system.

Law of Averages

When the output of a mine is considered, it is expressed in terms of tonnes per annum. The tonnes per annum is converted back to tonnes per month, tonnes per shift and then tonnes per hour on the conveyor. This logic is flawed in two ways. The first being efficiency of personnel during the working cycle, and the second, capacity limitations induced by the equipment.

The first point to realise is that the production capacity is erratic. The conveyor starts and stops during the shift which has a major influence on the production capacity. The same can be said about the shuttle cars discharging the material onto the conveyor belt.

This point is best illustrated by means of an example. The following table is an extract from actual production conditions in a typical coal mine. It reflects the number of shuttle cars during one week's production from Monday to Friday. There are three production sections noted. (Table 1).

SHIFT HOUR	NUMBER OF SHUTTLES PER HOUR ACTUAL FIGURES FROM A COAL MINE - where average = 8														
	M	T	W	T	F	M	T	W	T	F	M	T	W	T	F
	SECTION 1 total shuttles 312					SECTION 2 total shuttles 295					SECTION 3 total shuttles 348				
1	1	3	0	11	4	2	4	0	0	0	6	0	0	22	7
2	2	1	0	7	17	15	8	0	0	11	14	0	3	17	8
3	8	22	0	18	1	1	17	0	0	22	14	0	11	8	1
4	15	8	0	5	0	5	5	0	10	20	5	0	18	6	5
5	14	26	0	19	12	19	0	0	20	20	5	0	18	21	21
6	4	6	1	21	21	2	4	0	17	20	17	0	18	14	21
7	2	17	0	9	16	0	15	0	18	24	3	0	11	21	19
8	5	7	0	0	9	2	2	0	6	6	2	0	4	4	4

Table 1. Spread of the actual production figures

Calculating the averages number of shuttle cars per hour from the above information, the answer is 8. The maximum number of shuttles per hour is 26.

Using the same information but only showing the incidents where the number of shuttles is less than the average the table looks as follows:

SHIFT HOUR	NUMBER OF SHUTTLES PER HOUR LESS THAN AVERAGE OF 8 - ACTUAL FIGURES FROM A COAL MINE														
	M	T	W	T	F	M	T	W	T	F	M	T	W	T	F
	SECTION 1 total shuttles 49					SECTION 2 total shuttles 39					SECTION 3 total shuttles 55				
1	1	3			4	2	4				6				7
2	2	1		7									3		
3					1	1									1
4				5		5	5				5			6	5
5											5				
6	4	6	1			2	4								
7	2										3				
8	5	7				2	2		6	6	2		4	4	4

Table 2. Spread of the number of shuttles less than average

Using the same information but only showing the incidents where the number of shuttles is equal to the average, but still less than twice the average, the table is as follows:

SHIFT HOUR	NUMBER OF SHUTTLES PER HOUR MORE THAN AVERAGE AND LESS THAN TWICE THE AVERAGE - ACTUAL FIGURES FROM A COAL MINE														
	M	T	W	T	F	M	T	W	T	F	M	T	W	T	F
	SECTION 1 total shuttles 86					SECTION 2 total shuttles 59					SECTION 3 total shuttles 80				
1				11											
2						15	8			11	14				8
3	8										14		11	8	
4	15	8							10						
5	14				12										
6														14	
7				9			15						11		
8					9										

Table 3. Spread of the number of shuttles equal to average and below twice the average value

Using the same information yet again but only showing the incidents where the number of shuttles is equal and larger than twice the average, the table is as follows:

SHIFT HOUR	NUMBER OF SHUTTLES PER HOUR EQUAL TO AND MORE THAN TWICE THE AVERAGE - ACTUAL FIGURES FROM A COAL MINE														
	M	T	W	T	F	M	T	W	T	F	M	T	W	T	F
	SECTION 1 total shuttles 177					SECTION 2 total shuttles 197					SECTION 3 total shuttles 213				
1														22	
2					17									17	
3		22		18			17			22					
4										20			18		
5		26		19		19			20	20			18	21	21
6				21	21				17	20	17		18		21
7		17			16				18	24				21	19
8															

Table 4. Spread of the number of shuttles equal to twice the average and above

The only value resembling an average is the total number of shuttles per week. The production team is obliged to produce the minimum number of tonnes per month. To this end they need to achieve the minimum number of shuttles per week.

These tables clearly demonstrate that the mean value does not apply and the sizing of the equipment must realistically be based closer to the peak values as opposed to the average values.

Then there is the limitation induced by the mining equipment. The continuous miner can mine at a rate of 2000 tonnes per hour. The shuttle car has a capacity of 20 tonnes and transports chunks of material to the feeder breaker. It takes seconds for the shuttle car to dump the payload into the belly of the feeder breaker. The feeder breaker is set to discharge the material onto the receiving conveyor at a rate of 600 tonnes per hour.

In real terms, the utilisation of the continuous miner is in the region of 45% of its capacity. The 2000 tonnes per hour cutting machine works intermittently because the shuttle cars cannot take the mined material away fast enough to the waiting feeder breaker. The two or three shuttle car configuration takes the material from the miner for depositing it into the feeder breaker. The closer the miner is located to the feeder breaker, the quicker the turnaround time is for discharging the material. The shuttle cars find themselves either waiting to be loaded or alternatively, to be unloaded during the shift. The less time they spend waiting at either end the more efficient the system becomes.

In this system, the maximum output is the 600 tonnes material discharge onto the outgoing section conveyor belt. Should the tramming distances become too long, the material is discharged onto the outgoing conveyor typically in 600 tonnes per hour slugs with each slug having a total mass of 20 tonnes.

Doing the sums there are 600 ÷ 20 = 30 shuttles required per hour. Should there be two shuttle cars, this equates to 15 trips each per hour. When applying the logic of slow start at the beginning of the shift and the highest volume toward the end of shift, the number of shuttles varies accordingly.

Introduce into the scenario a belt stoppage during the shift. Everything now comes to a grinding halt. Due to the sequencing interlocks of the ore clearance system it takes time to get production flowing again and for the conveyors to be restarted sequentially. These stops do occur and overall production still needs to be pushed out. Invariably the setting on the feeder is upped a notch and before you know it the system is prone to becoming overloaded, and then there is spillage.

The aforesaid scenario takes into consideration the physical capacity of the mining machinery. As a statement of fact there is ongoing competition between the suppliers of this equipment. The end result of this healthy competition is that the capability and the production capacity of the existing machinery is correspondingly enhanced.

The situation on the various mines is never static. The machinery is serviced and refurbished on an ongoing basis and positive consideration is given to upgrading and modernising the equipment. Invariably this leads to increasing the production capacity of the kit and the ore clearance system is left wanting.

However, it remains a factual requirement for ensuring that there is an allowance built into the ore clearance system for increasing production capacity at a later stage in the lifecycle of the mine. This allowance can be twofold. Either the system can be oversized from the outset or alternatively, it may be the intention to speed the belt up at a later stage. The speed of a conveyor belt can be readily increased, for example by increasing the diameter of the driving pulleys. One needs to hasten to add that the installed power must still remain adequate for the application and the drives checked for sufficiency.

The only way to prevent this type of spillage is to have the system sized so that there is spare capacity to allow for the volumetric overload. Putting this into practice means ensuring that the maximum volumetric loading condition is duly considered when designing and sizing the ore clearance system.

Up to this point everything seems to indicate that spillage is an issue that will eventually find a place where it will manifest itself. What is the solution then? Key issues need to be reviewed again.

The economical carrying capacity is the place to start. This capacity is expressed in terms of the full capacity of the conveyor which is not far from the mark. But what is then forgotten? The physical lump is rarely, if ever, brought into the equation.

Minimum Belt Width Requirement

The Conveyor Equipment Manufacturer's Association in the USA (CEMA) tables minimum belt width as being a ratio of the maximum lump size. It also considers the condition where there is a mix of fines and lumps and qualifies the belt width accordingly. (Figure 3).

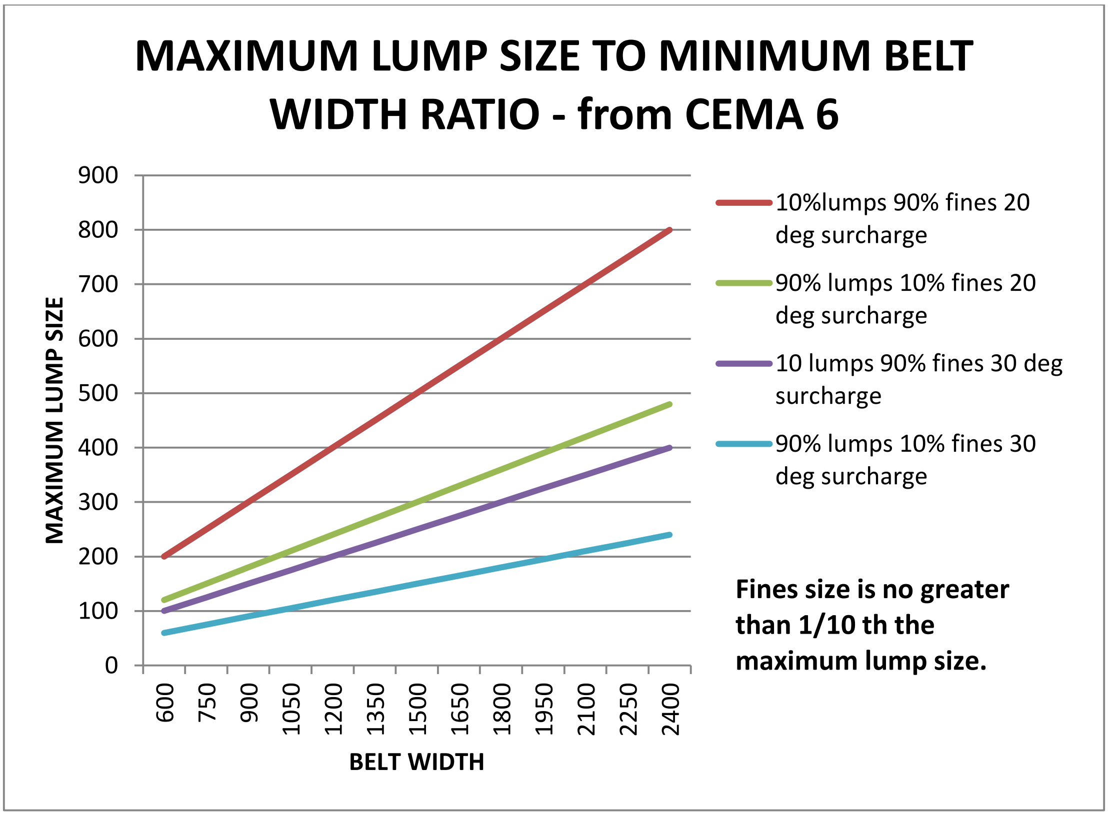

Figure 3. Ratio of maximum lump size

For an application where there are 350 mm lumps in the system with a particle size distribution of 10% lumps and 90% fines, the belt width will be:

3 x 350 mm lump = minimum 1050 belt width.

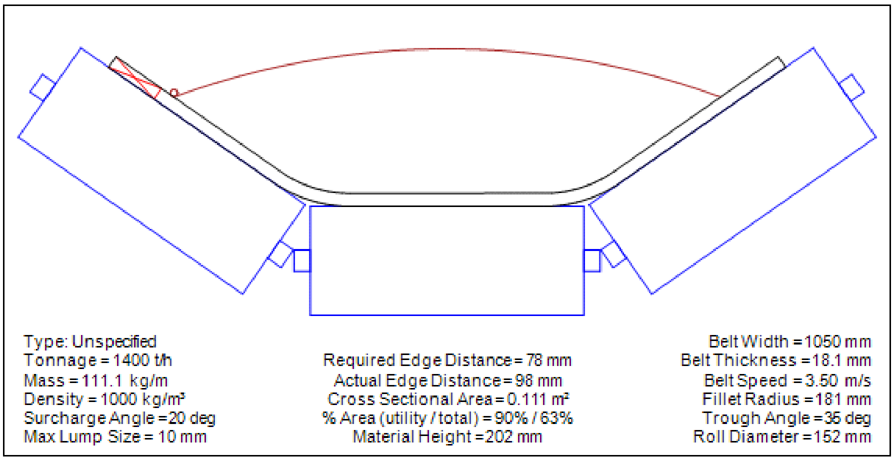

Figure 4. Cross-section without lump size consideration

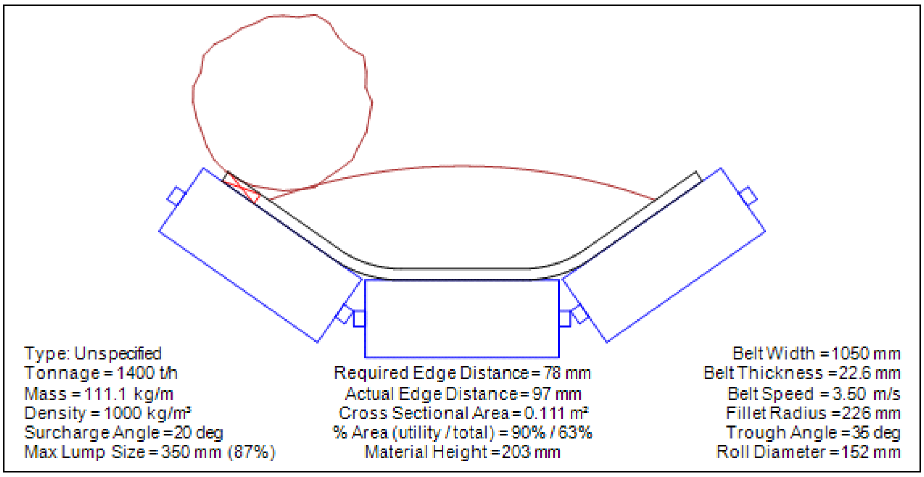

Figure 5. Cross-section with 350 mm lump size consideration

This is the standard approach used for determining the appropriate belt width and speed for the specific application. However, in order to make the system work, the volumetric capacity where the support for the lump is under the freeboard or full line needs to be increased. In this instance the speed of the belt is increased to compensate accordingly.

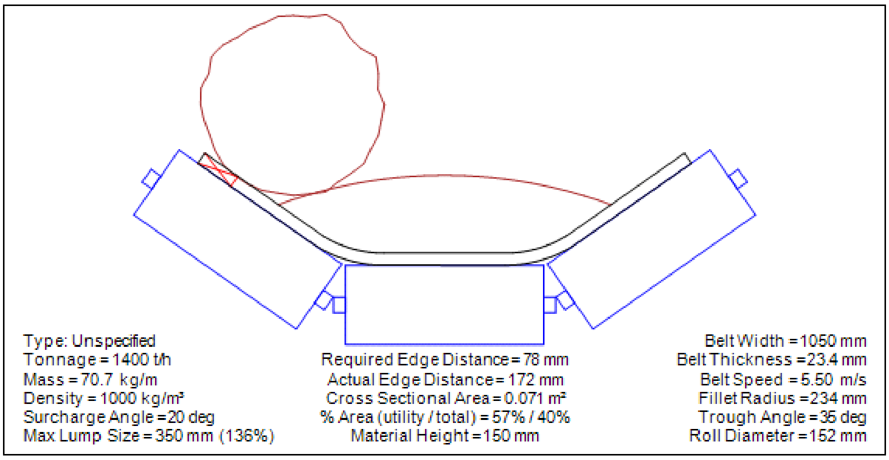

Figure 6. Belt running at 5,5 m/s

The speed needs to be increased from 3,5 m/s to 5,5 m/s for the lump to become properly supported.

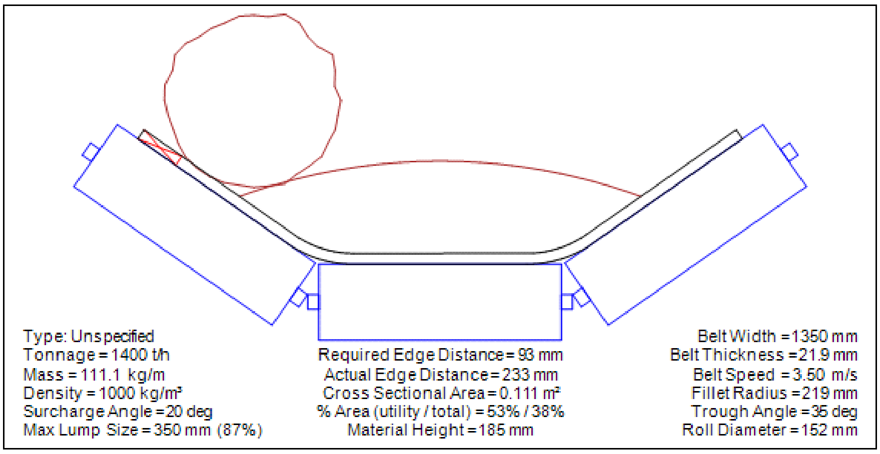

Figure 7. Increased belt width

Alternatively the width of the belting needs to be increased from 1050 mm to 1350 mm for the lump to become properly supported.

What can be deduced from this? During the conveyor design process designers do not have the understanding to include and make the necessary allowance for lump sizing when determining the volumetric carrying capacity of a conveyor system. The essence is that the belt must be wide enough to still support the lump within the boundaries of the full belt condition as can be seen in Figure 7.

Figure 7 shows the lump as a sphere. This is hardly ever the case in reality. For purposes of this paper it is sufficient to mention that the lump will have a length, breadth and depth. These can be quantified in terms of a shape factor and belt conveyor designers need to be cognisant of this. At this point the discussion needs to be stopped as this subject matter can be readily expanded and could quite easily become the content of a dedicated seminar.

To conclude this line of thought, lumps need to be included as a spillage hazard. Where spillage of small particles at low heights is not life threatening, spillage of larger lumps is definitely dangerous. Using the aforementioned examples, note the simplistic approach can be as easy as to equate the carrying capacity as a cross-sectional volume and then placing a lump on the side. Should the tangent of the lump on the belt fall within the extent of the fully loaded belt, it can be considered as being adequate for preventing spillage. The shape factor of the lump must ultimately be factored in the consideration as well.

It is sensible to include in this discussion the cross-sectional belting profile with respect to the idler configuration. In South Africa at least 90% of installations operate and make use of equal roller applications. The troughing rollers will typically be rollers of the same length and size, the impact rollers will be similar with return rollers being the same size as well.

The reasoning behind this is that the fewer the idler types, the more economical it becomes from a stockholding and maintenance perspective. From a loading and reliability perspective this is however, not always the case. This loading perspective will be dealt with later on in this paper but the reliability perspective will be the subject matter at one of the papers at Beltcon 18 in 2015.

Now, carrying on from where it was previously noted, one of the key requirements for preventing spillage is to have the material loaded centrally on the belt. This point can best be explained by means of an example. To this end an 1800 wide belt is selected.

For purposes of this discussion the belt speed is 3 m/s, design capacity 2800 tonnes per hour, material density is 1000 kg/m3, 35° trough angle and rollers to SANS 1313. The loading percentage relative to the full condition is 70%. This is a conservative design and spillage should never ever become an issue.

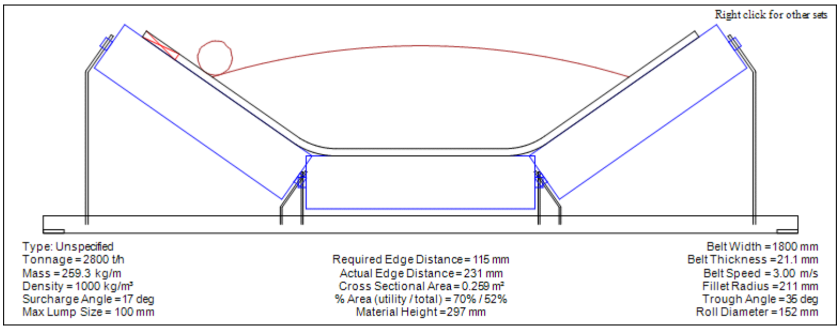

Figure 8. Standard SANS 1313 equal length roller application

But when the material is not loaded centrally onto the outgoing belt, the following happens.

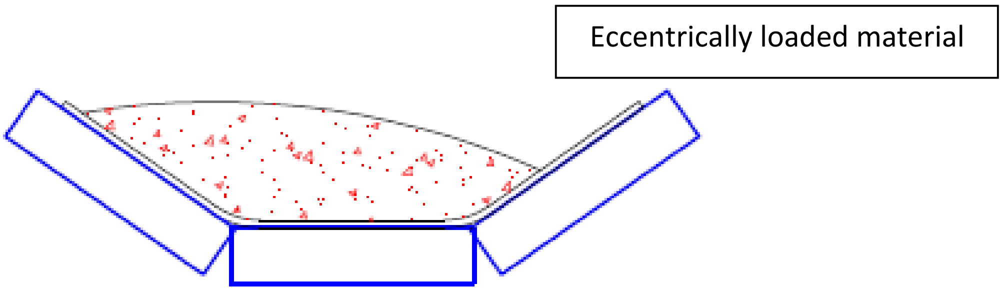

Figure 9. Section at the loading point

Note how the **material is loaded on the side** of the conveyor belt

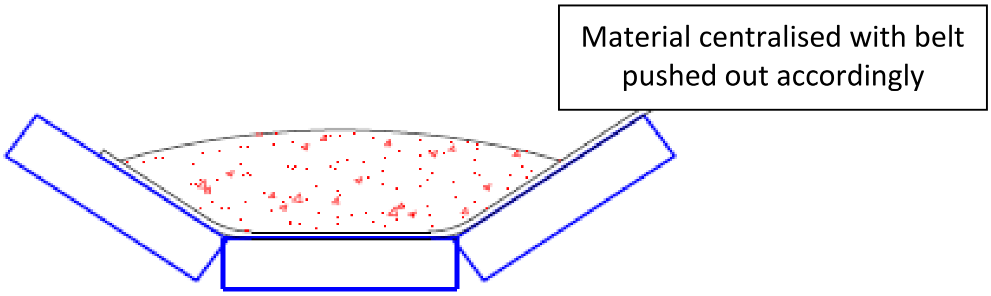

Figure 10. Section after material has centralised

Figure 11. Photographic confirmation of the aforesaid

In order to assist with central loading, the geometry of the receiving belt can be readily altered and changed at the loading point.

Below is exactly the same loading condition as before but now using a different roll configuration.

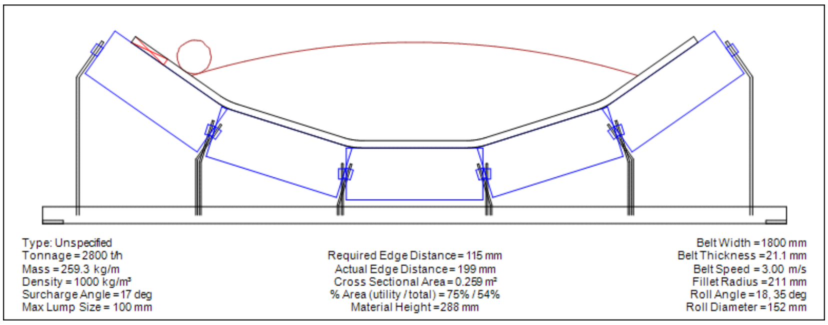

Figure 12. Five-roll configuration with 35°trough

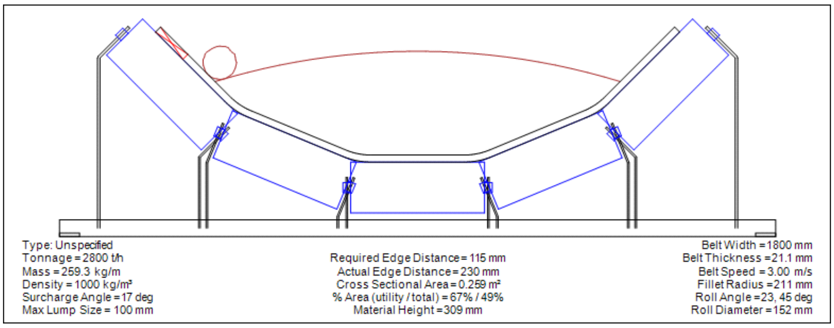

Figure 13. Five-roll configuration with 45° trough

There is nothing preventing the use of a specially designed idler base also troughing at 45 degrees along the following lines. Note that standard SANS 1313 roll lengths are still being utilised. The idler base becomes a 'special' design and accept that this is neither a production issue and nor is it a relevant cost issue.

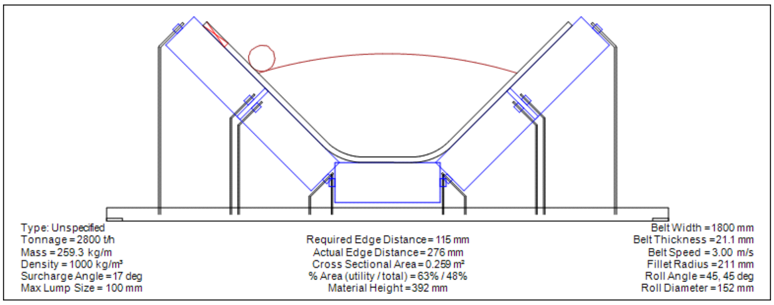

Figure 14. The way forward

Note that the capacity of the five-roll unit is less than that of the three-roll unit troughing at the similar angle. Then observe that the five-roll 45 degree has more capacity than the three-roll unit troughing at 35 degrees. The last configuration is the optimal solution.

This arrangement provides the best option for centralising the material at the transfer as well as offering the largest cross-sectional area of material, thus minimising the possibility of spillage.

IDLER CONFIGURATION	Loaded percentage	Actual freeboard distance
3-roll 35 degree trough	70% of full	231 mm
5- roll 35 degree trough	75% of full	199 mm
5-roll 45 degree trough	67% of full	230 mm
5-roll 45 degree trough	63% of full	276 mm

Table 5. This table confirms the improved approach

4. CONCLUSION

There is a direct correlation between spillage on conveyor systems and the safe operation of these systems. Spillage implies double handling and results in human intervention to get the material back onto the ore handling system. By eliminating spillage the human interface is no longer required and in doing so makes the system safer from an operating perspective.

There are various design approaches which have been considered by conveyor designers as being good practice which have been now highlighted as requiring reconsideration.

- Brakes and flywheels form an integral part of conveyor at transfer points
- Volumetric capacities of conveyor belts need to be carefully considered
- Material centralising at transfer points is a requirement
- Maximum lump sizes to be duly considered during conveyor design
- Single particle analysis forms the backbone of adequate chute design
- Full material control is required for successful transfer chutes
- Consideration is required for all conveyors to become feeders during the starting cycle.

Safe operating principles require to be duly considered at the initial design of all conveyor systems. Conveyor designers are thus required to become more knowledgeable with the operating side of the application and not to be theorists on what should and should not be with little or no regard for the practical and maintenance considerations.

5. RECOMMENDATIONS

There is room for improvement on the majority of conveyor installations in South Africa with the view of reducing spillage. These include:

- Transfer chutes need to act as feeder arrangements as well
- All transfer points must allow for the containment of overrun of material during stopping.

Ore clearance systems should not be designed to operate close to 100% of the full volumetric loading capacity. Due consideration must be given to reserve capacity relative to lost production requirements, mining equipment capacities and maximum loading conditions as opposed to average loading conditions.

REFERENCES

ISO 5049

CEMA 6

CMA conveyor design diploma notes

ABOUT THE AUTHOR

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To date the author has been directly involved in the conveyor industry for over 34 years. During this time he has been exposed to all facets of the conveyor industry ranging from mechanical design, manufacture, installation, commissioning, visual and forensic ore clearance audits and feasibility studies. The engineering of conveyor systems for both underground and surface applications are second nature to him. As current chairman of the Conveyor Manufacturers Association he is also actively serving on the SABS' technical committees responsible for reviewing the national standards relative to all conveyor related issues.

Highlights of his career are the four full patents and two provisional patents that he has registered as well as various firsts in conveyor installation conveyor systems like the first powered tripper drive in South Africa. Another milestone worth mentioning is the class 1250 PVC dual booster conveyor designed for an underground application at an overall length of 7 300 metres.

Current employment is with Flexco as project development manager responsible for developing new products for the company.

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