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Transport with Large Conveyors in Surface Mines

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Summary

An attempt is made to differentiate between the criteria usually required for the design of conveyor systems under ordinary operating conditions and those with which open-pit planners will be increasingly concerned in the future.

Based on the well-known relationship between lifting power requirements, the calculation of the power requirement for conveyor equipment is shown by calculating the local belt pull.

The overall system utilization is shown to be a function of the characteristics of the mining machines and the availability of the conveyor systems.

Specific ways of considering parallel and slewing operation of conveyor systems are outlined.

Finally, attention is drawn to the dynamic behaviour of the transport of single large lumps and the possibility of assessing the respective forces is indicated, in view of the growing importance of handling these lumps in hard rock and also in soft rock open pit mining.

1. Introduction

In the early fifties, conveyors increasingly "gained ground" in lignite open-cuts. They were not only used as stationary but also as shiftable systems for mining and dumping faces. In this way, the typical conveyor characteristics of continuously transporting large bulk material masses up gradients, which until then had been considered too steep, could be utilized. However, the special conveying condition of individual flights over the length of a straight face had to be taken into consideration. Arrangement in curves, although possible in some cases, will not be dealt with in this connection.

By connecting conveyor flights into lines and combining these systems, the conveyor developed into a high capacity transport system which naturally led to a correspondingly complex distribution of mined material.

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In lignite open-cuts, conveyor systems essentially displaced rail transport from its then dominating position; and in other open-cuts they became a challenge to heavy truck transport owing to increasing mining depths and greater volumes of bulk material having to be handled. Finally, this situation also applies to hard rock open-cuts and not only to intermediate and discharge transport but also at the face itself.

From Fig. 1 it can be seen how the configuration of the deposit dictates the way of opening up a mine. It shows that considerable use of conveyor equipment is required where great quantities of overburden cannot be directly overthrown to the dump owing to the wide opening of the open-cut, or where the overburden has to be carried outside the mine in any case.

Lump size limitation for conveyor transport in hard rock open-cuts can be largely overcome by using crushers for "intermediate treatment".

The following deals mainly with the changing problems of designing conveyor systems with special reference to open-cut operations.

2. Basics for the Design of Conveyor Systems in Open-Cuts

Apart from solutions to individual problems of conveyor system components, initially the engineer planning open-cut concepts only had the DIN-formula to work out the behaviour of conveyors. This was refined, with increasing experience, in calculating power requirements according to the different resistance coefficients (Fig. 2).

The main objective was to meet the maximum power requirement. The only operating parameters considered were flow rate, conveyor length and difference in height between receiving and discharge points.

However, with increasing conveying distance and conveying over uneven ground these parameters did not cover all operating conditions. More and more, calculations are now used for the design of long distance conveyor systems which make due allowance for individual load distribution over the different flights. In this way the power requirement for any loads and profiles can be estimated for each section (Fig. 3).

The practical calculation is also mainly based on the DIN-formula and differs only in the distribution of loads according to the flights. The computation of local belt pulls in an

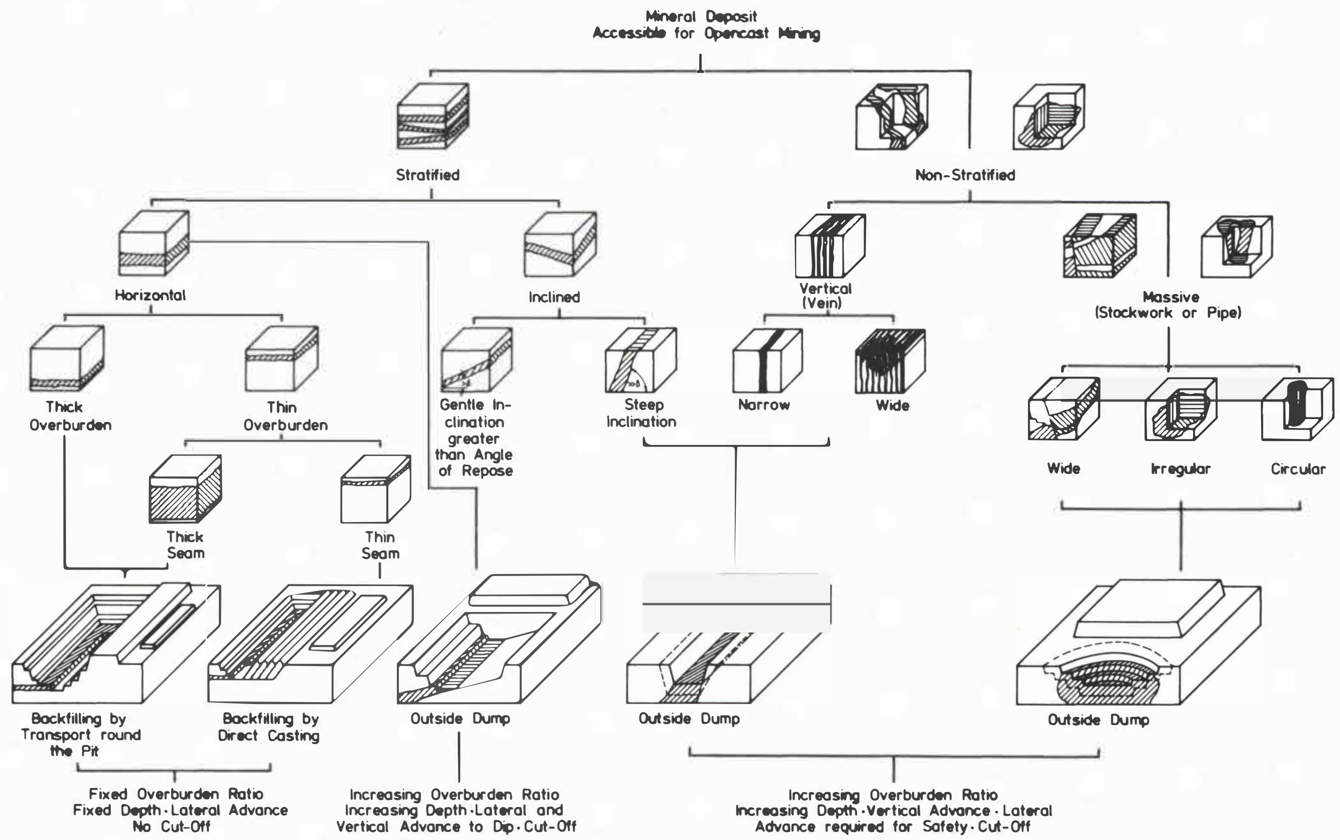


Fig. 1: Classification of Surface Mines

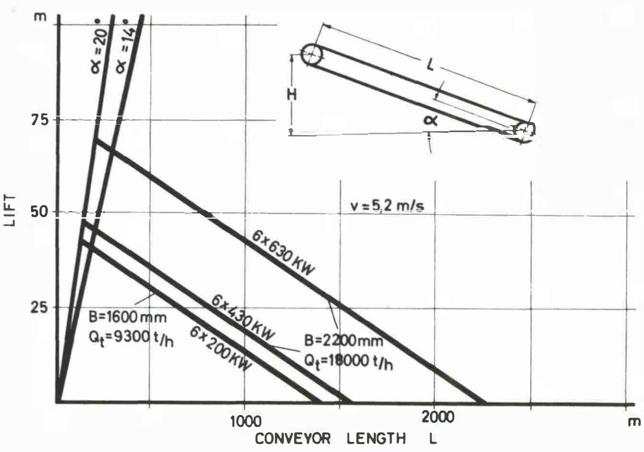


Fig. 2: Drive power requirements of a belt conveyor

"all-round calculation" makes it possible to determine all operating conditions quickly and with satisfactory accuracy.

It is based on a conversion of the DIN-formula and describes the expected local operational conditions satisfactorily. This method is gaining significance as with increasing length of systems and optional system profiles, local parameters become more decisive for the operational safety of the system. Further, it allows varying the position of the drives, a particular advantage when determining intermediate drives.

A particular criterion in the assessment of belt conveyor systems in open-cuts is their degree of utilization.

Total utilization η_A of a belt conveyor system is the product of load degree and time degree:

$$\eta_A = \varphi \times \eta_T$$

The load degree is determined by the characteristics of feed, the time degree by the behaviour of the system as well as its tie-up in the complete conveyor system.

Usually a trouble factor $x = \dfrac{T_F}{T_B}$ is attributed to a system under certain operating conditions. From this factor the operating time degree is calculated as follows:

$$\eta_T' = \frac{1}{1 + x}$$

In a chain of conveyors (direct connection of conveyor flights in one line) the operating time degree is calculated as follows (Fig. 4).

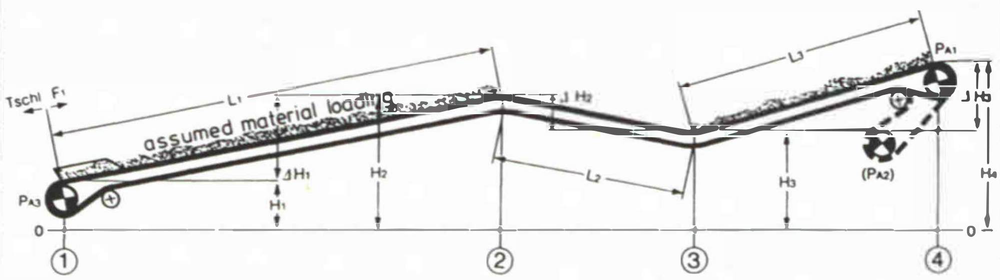
$$\eta_T' = \frac{1}{1 + \Sigma x_i}$$

Time delays due to the re-direction of material — depending on its type — also result in changes in the operating time degree:

$$\eta_T' = \frac{1}{1 + x_{vert} + \Sigma x_i}$$

The operating time degree is that which is typical for the feeding branches in a conveying intersection.

profile of system L m	H m
0	+ 10
1000	+ 60
1500	+ 40
2100	+ 80



system:

operating condition: operation

material loading: see sketch

remarks:

section	starting force F_1	section length $\Delta L = L_{i+1} - L_i$	movement resistance - system- $\overline{W_{oi}} \cdot \Delta L$	movement resistance - material- φ_A	$\overline{W_{Li}} \cdot \varphi_A \cdot \Delta L$	average height $\Delta H = H_{i+1} - H_i$	upward conveyance - belt- $m_{Bi} \cdot g \cdot \Delta H$	upward conveyance - material- $m_{Li} \cdot g \cdot \Delta H$	circumferential force load factor φ_u	F_u	sectional end force T_{i+1}	required change in belt pull ΔT = 20133	remarks
-	N	m	N	-	N	m	N	N	-	N	N	N	
1 - 2	30000	1000	5046	1	37890	+ 50	+ 14715	90742	-	-	178393	198526	F_{NEU} = 50133
2 - 3	-	500	2523	0	0	- 20	- 5886	0	-	-	175030	195163	-
3 - 4	-	600	3027	1	22734	+ 40	+ 11772	72594	-	-	285157	305290	-
	-	-	-	-	-	-	-	-	1	-166667	118490	138287	$T_{21} \geq$ 106667
4 - 3	-	600	3027	0	0	- 40	11772	0	-	-	109745	129878	-
3 - 2	-	500	2523	0	0	+ 20	+ 5886	0	-	-	118154	138623	-
2 - 1	-	1000	5046	0	0	- 50	- 14715	0	-	-	108485	128618	-
	-	-	-	-	-	-	-	-	0,94	,-78334	30151	50284	$T_{23} \geq 50133 = F_{NEU}$
$F_1 - T_{SOLL}$	-	-	-	-	-	-	-	-	-	-	[illegible]	[illegible]	-

Q_1 2000 t/h
L = 2100 m
H = +70 m
B = 1400 mm
v = 3 m/s

P_{A1} = 2 x 250 kW
P_{A2} = - kW
P_{A3} = 1 x 250 kW
P_A = 750 kW

α_1 = 180°
α_2 = - °
α_3 = 180°

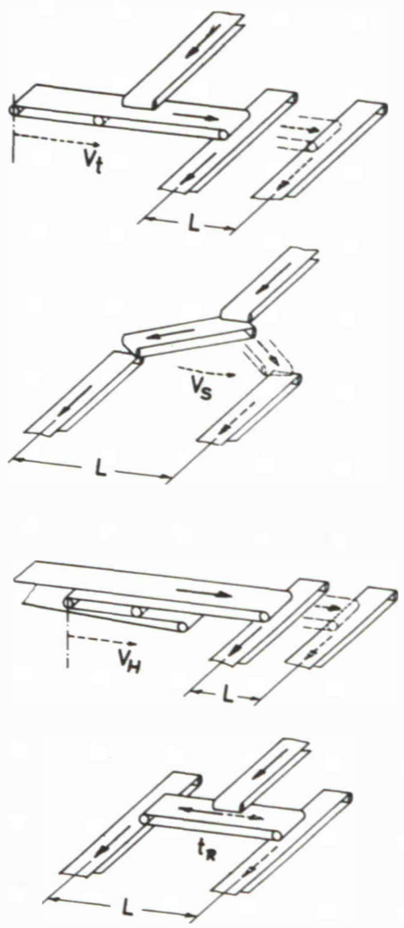
μ_1 = 0,3
μ_2 = -
μ_3 = 0,3

$T_{21} \geq$ 106667 N m_{M1} 30,7 kg/m
$T_{22} \geq$ - N m_{B1} 30 kg/m
$T_{23} \geq$ 50133 N m_{L1} 186 kg/m

C = 1,05
f = 0,018
C_M = 0,6 · C = 0,63
C_L = 1,1 · C = 1,16

relative movement resistance:
system: $\overline{W_{o1}} = 0{,}5\, C_M \cdot f \cdot g\, (2 m_{B1} + m_{M1})$ = 5,04 N/m
material: $\overline{W_{L1}} = C_L \cdot f \cdot g \cdot m_{L1}$ = 37,9 N/m
relative upward conveyance:
belting $m_{B1} \cdot g$ = 294,3 N/m
material $m_{L1} \cdot g$ = 1814,8 N/m

Fig. 3: Continuous calculation of belt tension



SYSTEM	TIME EFFICIENCY
SHUTTLE BELT (S)	$\eta' = \dfrac{1}{1 + K_{SB} + \dfrac{L/V_t}{T_{op}}}$
SLEWING BELT (SL)	$\eta' = \dfrac{1}{1 + K_{SL} + \dfrac{L/V_S}{T_{op}}}$
SHUTTLE HEAD (SH)	$\eta' = \dfrac{1}{1 + \dfrac{L/V_H}{T_{op}}}$
REVERSIBLE CONV. (SR)	$\eta' = \dfrac{1}{1 + K_{SR} + \dfrac{t_R}{T_{op}}}$

Fig. 4: Time-efficiency of different material distribution systems

The discharging branches of a conveying intersection or system are additionally subject to time limitation owing to non-loading due to the selection of materials (Fig. 5).

The loading degree φ depends on the production characteristics of the feeding equipment, an idea of which can be obtained from comparison with certain bucket wheel excavators. If the latter's capacity is related to and reduced by mining conditions, an idea of the loading degree value for the follow-on conveyor system will be obtained which is no longer identical with the continuous full work load of the available conveyor cross-section.

The loading degree derived from the joint feeding of a conveyor system by two machines can be ascertained from the probable load distribution of each. It appears that the sum of both maximum loads hardly ever occurs and therefore collective conveyor systems of lower capacity can be used. The common peak which occurs for short periods only can be absorbed by the overload capacity of the conveyor system. Up to the present, the effect upon the loading characteristic of throttling one machine in order to stay below the maximum loading capacity is still not investigated.

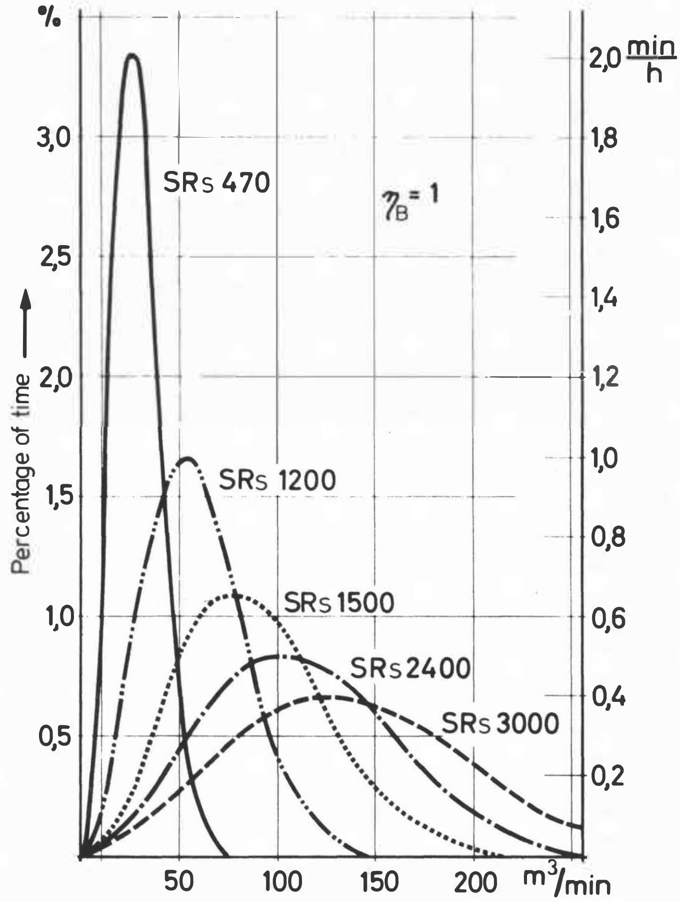


Fig. 5: Distribution of performance fluctuation of bucket wheel excavators

3. Utilization of Conveyor Lengths

When designing open-cut conveyor systems, special attention should of course be paid, as conveyor length increases, to the implications and also to the possible fluctuating flow of material and the latter's effect of reducing the average flow rate and, therefore, of leaving a considerably higher drive capacity reserve (Fig. 6).

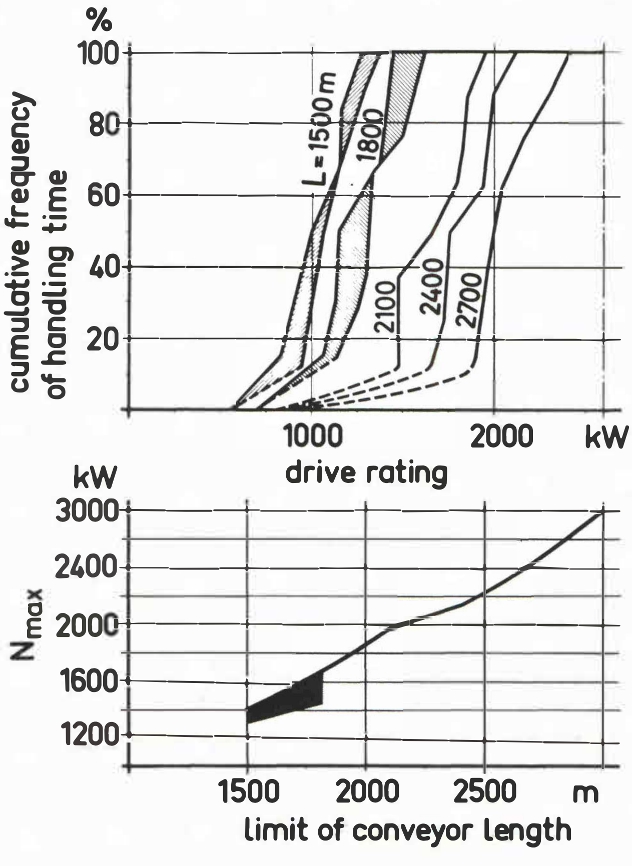


Fig. 6: Example of probable drive rating requirements and limits of conveyor length at non-uniform delivery of material from bucket wheel excavator

$$N = \frac{C \cdot f \cdot L}{367}\left[3{,}6 \cdot G_m \cdot v + Q_t\right]$$

$$\boxed{N_w = \frac{C_w \cdot f_w \cdot L}{367}\left[3{,}6 \cdot G_m \cdot v + k \cdot H_w \cdot v_{sw} \cdot s_w \cdot \gamma_w\right]}$$

C_w = coefficient

f_w = friction factor at idlers

L = conveyor length in m

G_m = weight of belting + idlers in kp per meter of carrying and return strand

v = belt speed, meters per second

H_w = slice height of bucket wheel cut

v_{sw} = slew speed of bw.-excavator

s_w = slice thickness

γ_w = bulk weight

k = conversion factor

Jndex w = indicates probability influences

As the required conveyor lengths depend on the configuration of the open-cut, planning engineers often have the problem of having to introduce additional drive stations, if the limiting conveyor length is exceeded. To avoid such expensive measures, the alternative would be to lengthen the existing system and reduce the average material flow within a given time or to alter any other power consuming factors.

Here, a reduction in height difference between receiving and discharge points is a possible solution.

Fig. 7 shows some practical examples of this. Engineers with open-cut experience are, of course, more inclined to make such compromises than engineers planning according to theory and without practical experience.

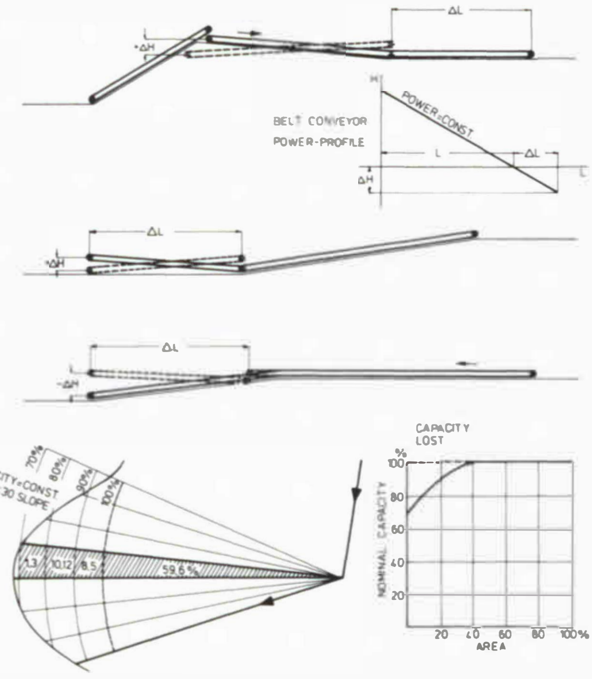


Fig. 7: Balancing of power requirement

4. System Designs

When working out technical concepts of conveyor equipment, it will be found that conveyor systems in module design — except where used up to nearly their limiting lengths — are more expensive than a "tailor-made" individual system. Module design will always be of advantage in cases where power parameters are changed after short periods of operation. The illustration shows, by way of example, how drive capacities are determined on the basis of the different mining phases of an open-cut (Fig. 8).

5. Parallel or Slewing Operation with Conveyor Systems

A basic question, which frequently crops up in open-cut planning, is whether it is more advantageous to use conveyor systems in parallel or in slewing operation. In the past, this question was mainly decided in favour of parallel operation on the basis of the volume of mined material related to the time resulting between two conveyor shiftings. However, a study of experienced open-cut operations with conveyor transport reveals that an increasing number of slewing operations are being introduced. If slewing operation is used, working at the mine face, for example, is made easier by the fact that the conveyor only has to be slewed around the drive station. Shifting of the conveyor itself is relatively independent of the advance of the drive station: transfer points need not be re-arranged, link conveyors do not have to be lengthened. Supplies to the face within the mine such as electric cables, drainage systems and also roads remain unchanged for longer periods (Fig. 9).

On the mining side, earlier shifting will be possible when digging in several block steps. A big advantage is that mining can be adapted to individual working levels.

It is apparent that in addition to a simple comparison of the volume of material, there is a number of other operating criteria to be considered which determine the technical concept.

6. Conveyor Systems Which Require Prior Crushing of the Material to be Transported

In recent years, heavy truck transport has increasingly been replaced by conveyor removal. Experience shows that as a general rule it can be said that with about 50 m depth to be mined, conveyor transport is economically superior to heavy truck transport even if waste material has to be broken up. The illustration shows the experience at the copper open-cut Twin Buttes Mine (Fig. 10).

Conveyors have been recently increasingly installed, too, in hard rock open-cuts. This mostly requires the application of crushers prior to delivery of the material on to the conveyor. Unless the material is screened, the capacity of such a system exclusively depends on the capacity of the crusher. This mainly depends on the required lump size and it is therefore of advantage to select a lump size which is close to the upper limit for the conveyor width used. Although conveyor systems in hard rock open-cuts are capable of handling material including a large number of coarse lumps, their maximum dimensions are clearly defined.

Owing to the crushing process, the material will, however, have sharper edges than material in soft rock operation, where larger lumps are mostly picked up in their natural form. The conclusions resulting from these problems will have a bearing on the required wear characteristics of the belt cover and its resistance to penetration of sharp-edged pieces as well as the behaviour of single lumps at maximum conveyor gradients, or on the conveyor system in general.

Without wishing to deal with impact loads on transfer points of the conveyor by single lumps, attention is drawn to the increase in dynamic loads on the conveyor caused by single lump transport, a problem which will undoubtedly become more important. This, in particular, as in future a considerable capacity increase of the crushers used is to be expected (Fig. 11).

7. Loads on Idlers When Transporting Large Lump Material

Formulations normally used in the past only allowed for dynamic effects on carrying idler loads when transporting single lumps by a coefficient of the average material load which increases by the square of the speed.

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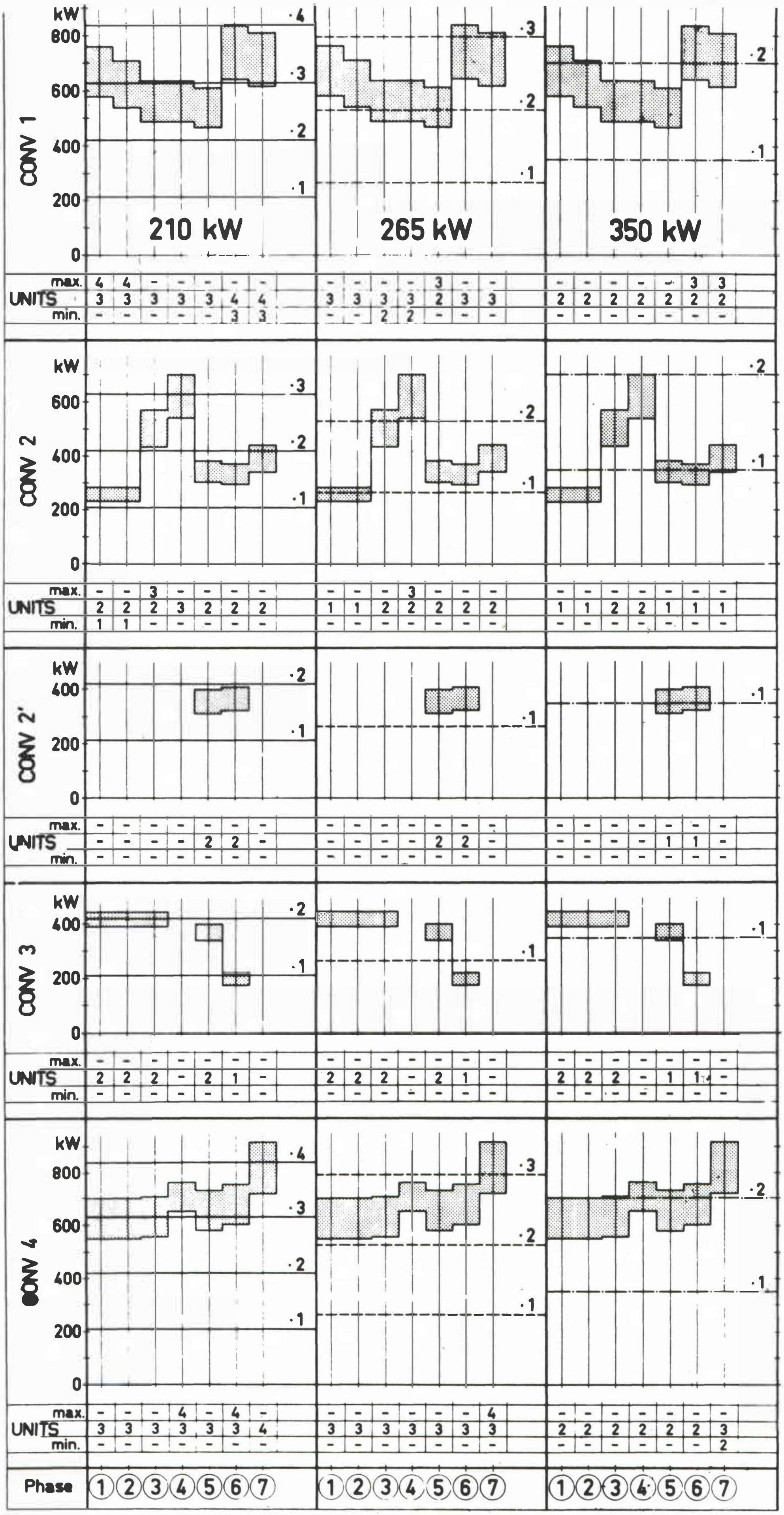
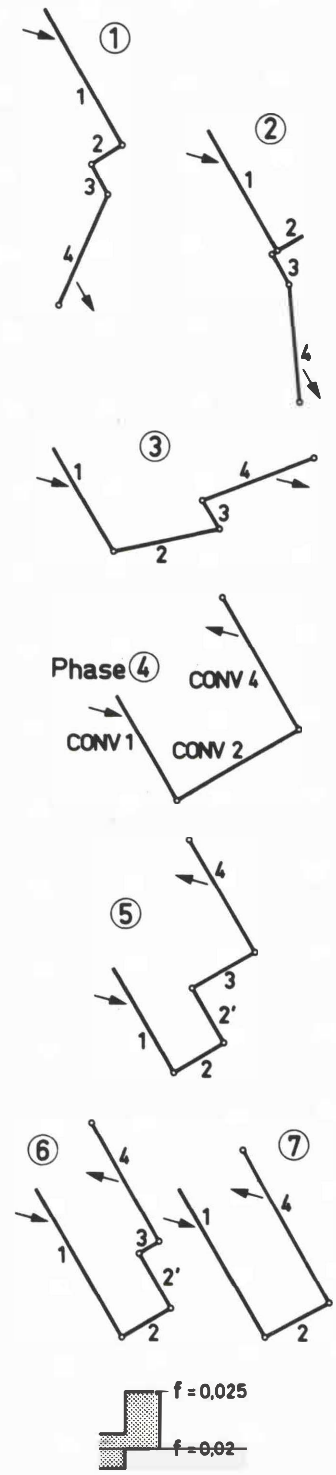




$$N = \frac{1}{\eta} \cdot \left[\frac{C \cdot f \cdot L}{367} \cdot (3{,}6 G_m \cdot v + Q_t) + \frac{Q_t \cdot H}{367} \right]$$

REQUIREMENT

		①	②	③	④	⑤	⑥	⑦
350 kW	max.	-	-	-	-	-	8	7
	UNITS	7	7	8	6	7	7	6
	min.	-	-	-	-	-	-	5
265 kW	max.	-	-	-	9	12	-	9
	UNITS	9	9	10	8	11	11	8
	min.	-	-	9	7	-	-	-
210 kW	max.	11	11	11	10	-	13	-
	UNITS	10	10	10	9	12	12	10
	min.	9	9	-	-	-	11	9
Phase		①	②	③	④	⑤	⑥	⑦

Fig. 8: Selection of drive units for conveyor systems

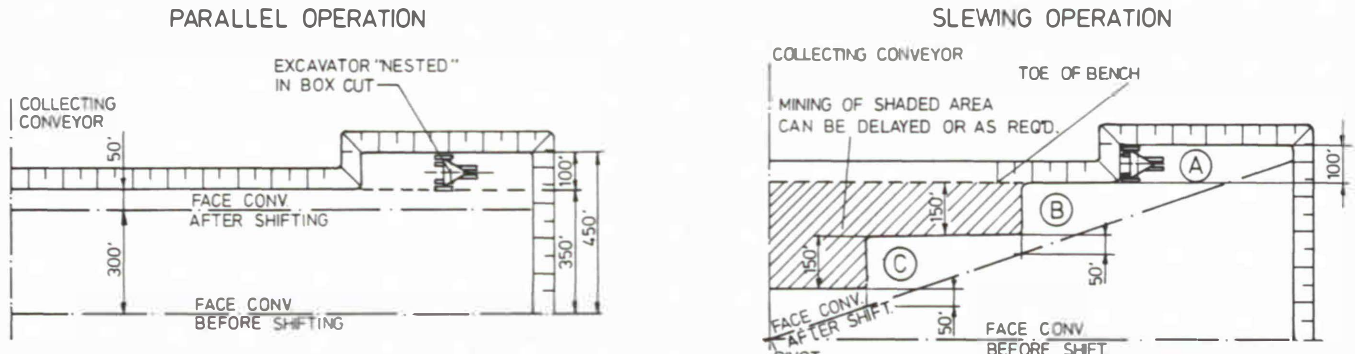


Fig. 9: Parallel/slewing operation of belt conveyors

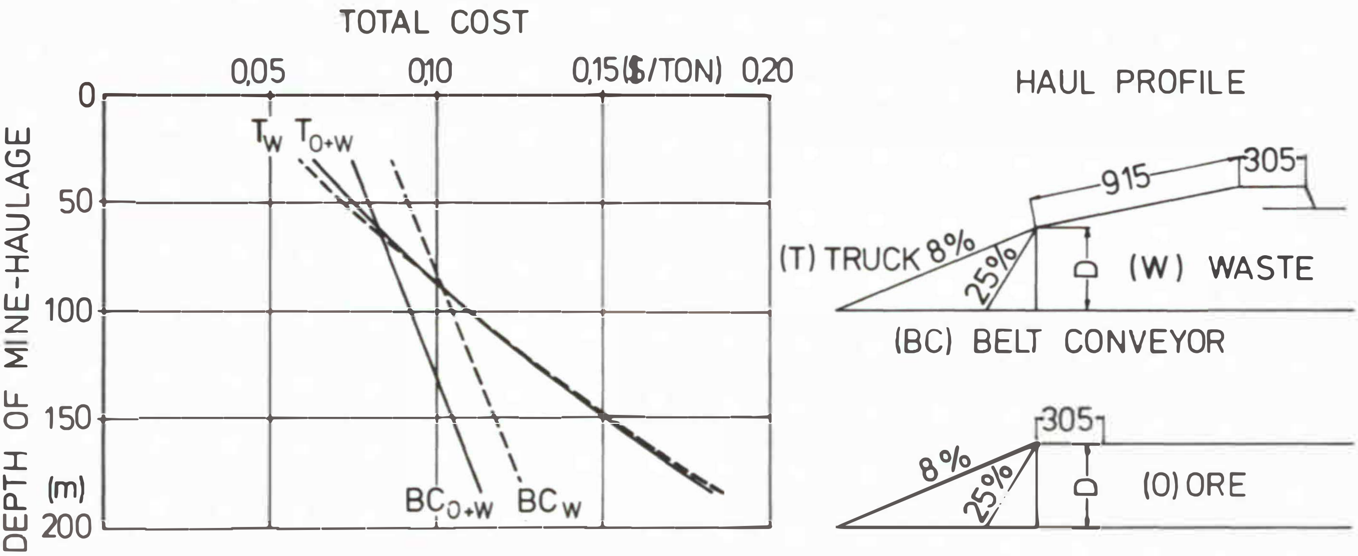


Fig. 10: Cost of waste and ore haulage (Twin Buttes Mine)

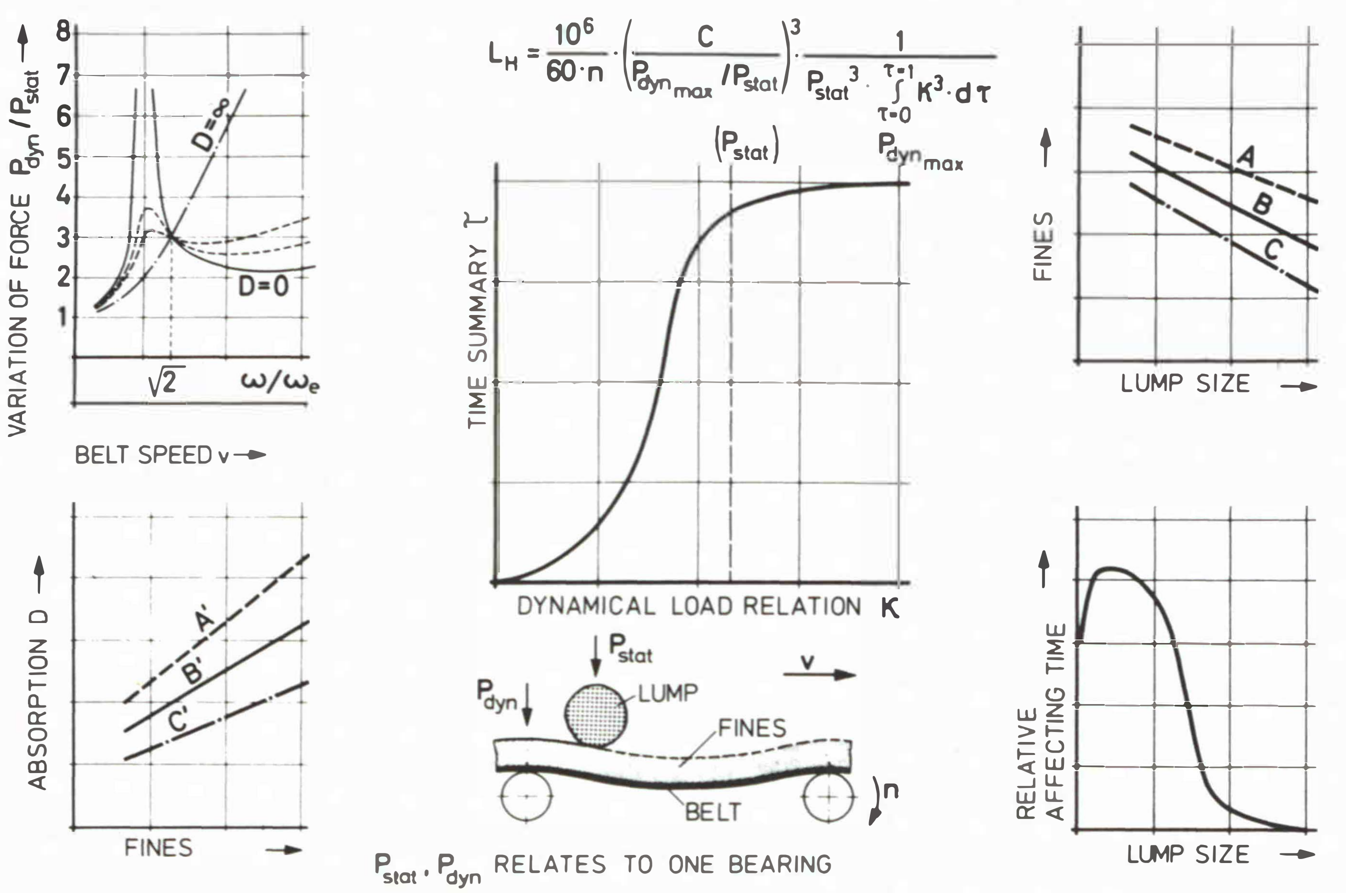


Fig. 11: The dynamic load for a lifetime calculation of an idler bearing

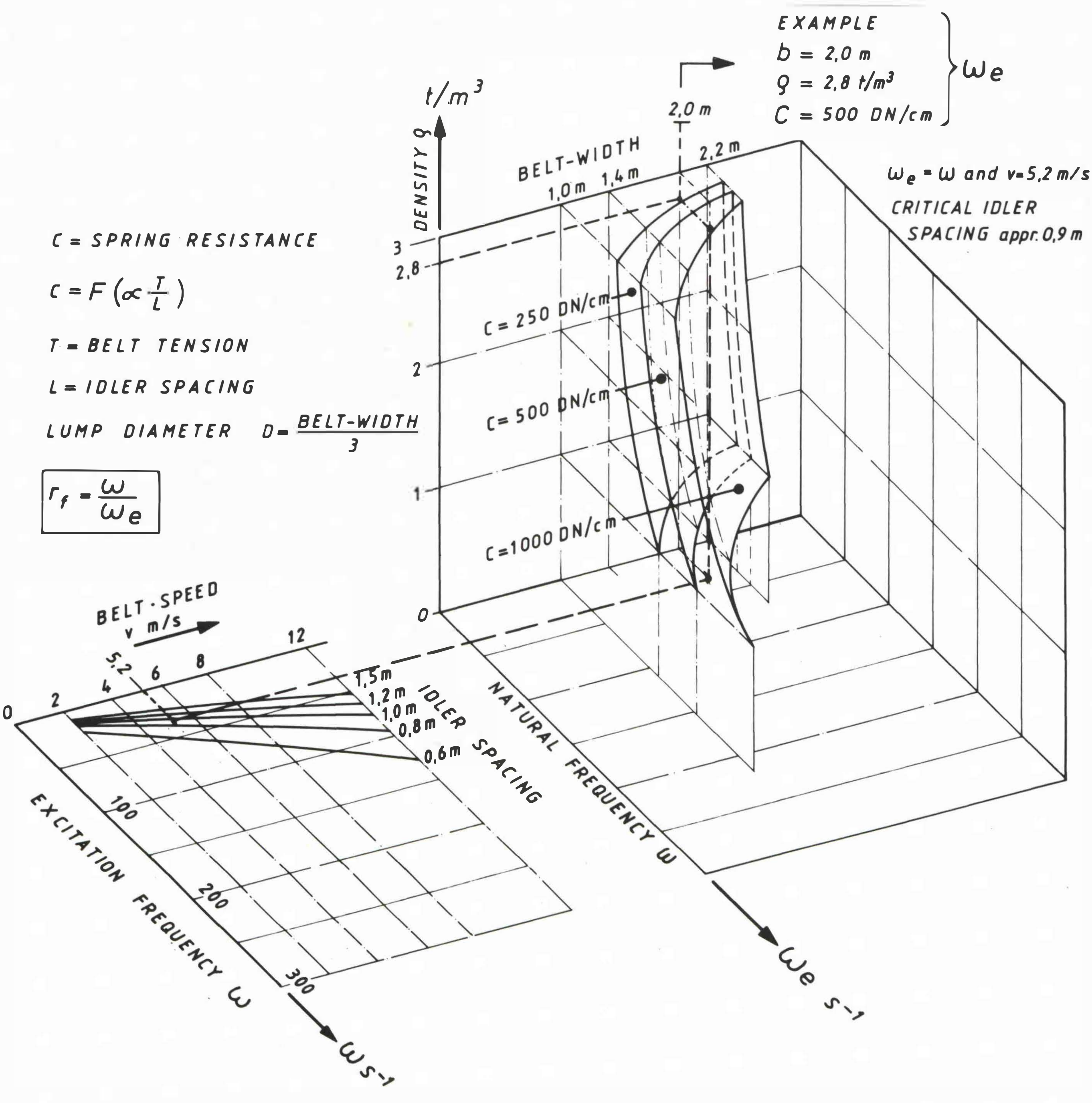
However, if dynamic forces are studied from a vibration model, the effects from conveying single lumps are represented by a resonance diagram and are, therefore, dependent on the dynamic behaviour of the system, the characteristic frequency and the above normal frequency.

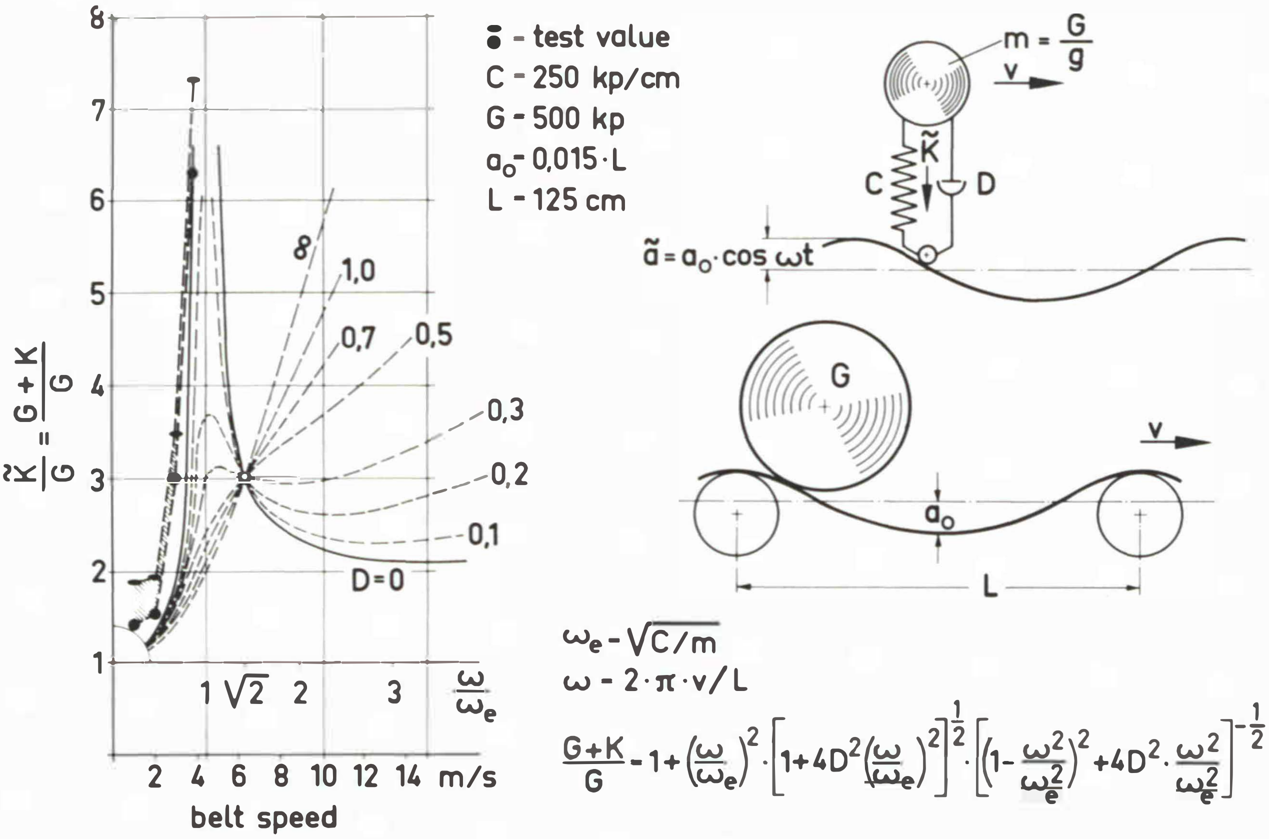
As the characteristic frequency of the vibration system changes with belt pull, locally limited areas can be determined in a conveyor system where single masses build up vibration and become particularly destructive. The greater the fluctuations in the belt pull of a system, the greater the number of such critical areas which will occur.

Fig. 12 shows as a summary a first field calculation to determine the characteristic frequency behaviour from which, by means of belt rigidity values, the critical belt pull areas can be determined which are subject to a particularly high build-up of vibration. The lumps were assumed to be spherical.

If single lumps are embedded in a layer of fine material, the results can analogously be traced back theoretically to the idler load after introducing a damping value (Fig. 13).

Fig. 12: Conveying of single lumps. Determination of the frequency relation r_f




- test value
C - 250 kp/cm
G - 500 kp
a₀- 0,015·L
L - 125 cm
m = G/g
v
K̃
C
D
ã = a₀·cos ωt
G
v
a₀
L
ωe - √C/m
ω - 2·π·v/L
(G+K)/G - 1+(ω/ωe)²·[1+4D²(ω/ωe)²]^½·[(1- ω²/ωe²)²+4D²·ω²/ωe²]^-½
K̃/G = (G+K)/G
1,0
0,7
0,5
0,3
0,2
0,1
D=0
ω/ωe
belt speed
m/s